A TECHNICAL AND ECONOMIC STUDY OF CANDIDATE UNDERGROUND MINING SYSTEMS FOR DEEP, THICK OIL SHALE DEPOSITS PHASE I REPORT

United States
Department of the Interior
Bureau of Mines

USBM Contract Report SO 241074

July 1975

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Bureau of Mines Open File Report 23-76

DISCLAIMER NOTICE

SUBJECT INVENTIONS

Cameron Engineers, Inc. does hereby certify that no patents or inventions have resulted from this study.

FOREWORD

This report was prepared by Cameron Engineers, Inc., 1315 South Clarkson Street, Denver, Colorado, 80210, under U.S. Bureau of Mines Contract Number SO 241074. The contract was initiated under the Bureau of Mines Program for Advancing Mining Technology--Oil Shale. It was administered under the technical direction of 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 recently completed as part of this contract during the period July 1974 to July 1975. This report was submitted by the authors on 11 July 1975.

The current decline in the domestic reserves of easily obtained crude oil, coupled with a drastic increase in the price of imported oil, has spurred the evaluation of all possible sources of oil. The 1600-square mile Piceance Creek Basin in Colorado contains the largest single known hydrocarbon resource in the world. This resource occurs in the Parachute Creek Member of the Green River Formation as kerogenetic marlstone or, as it is commonly termed, oil shale. Although this potential source of oil could not feasibly replace America's current six million barrel-per-day oil deficit, it is possible that shale oil could replace up to one-third of this amount.

Current American oil shale mining technology is based on operations at three prototype room and pillar mines and a prototype modified in situ recovery project. In an effort to evaluate the economics of mining oil shale and to develop further technology, the USBM has initiated three independent contract studies to evaluate underground, open pit, and modified in situ mining systems.

The subject contract study involves the evaluation of all potentially feasible systems for large scale mining of the thicker oil shale deposits in the deeper central portion of the Piceance Creek Basin. The study is divided into two phases: (1) an initial investigation and subsequent engineering evaluation of all potential underground mining systems for

TABLE OF CONTENTS

TABLE OF CONTENTS con't

TABLE OF CONTENTS con't

TABLE OF CONTENTS con't

LIST OF FIGURES con't

LIST OF FIGURES con't

LIST OF FIGURES con't

LIST OF FIGURES con't

LIST OF TABLES

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LIST OF SYMBOLS AND ABBREVIATIONS

γ - unit weight

$\Delta\sigma_h$ - total horizontal stress change due to extraction assuming premining horizontal stress equals premining vertical stress.

$\Delta\sigma_v$ - total vertical stress change due to extraction

ε_v - vertical strain

ζ - material constant

ν - Poisson's ratio

ξ - material constant

σ_b - minimum allowable compression

σ_e - maximum compressive stress

σ_f - compressive bending stress in upper fiber of beam

σ_h - horizontal stress

σ_o - rock mass cohesion

σ_p - maximum failure stress

σ_t - tensile strength

$\hat{\sigma}_v$ - peak pillar stress

ϕ - angle of internal frictional resistance

b - bolt length

c - cohesion

C - centroidal distance

C_1 - strength of core sample

C_2 - strength of pillar

cfm - cubic feet per minute

cfs - cubic feet per second

cfsm - cubic feet per square mile

C_o - unconfined compressive strength

D_1 - diameter of core sample

D_2 - diameter of pillar

E - Young's modulus

FEL - front end loader

fpm - feet per minute

gpm - gallons per minute
H - height of overburden above pillar
H_1 - height of core sample
H_2 - height of design pillar
h - pillar height
Δh - change in pillar height
ℓ - roof beam span
LHD - load-haul-dump
M - bending moment
mg/l - milligrams per liter
MM - million
mtpy - metric tons per year
R - percent extraction
r - correlation coefficient
$\tan\beta$ - passive pressure coefficient
tph - tons per hour
tpy - tons per year
W - pillar width
yd^3 - cubic yards
$\hat{Y}$ - distance into pillar locating peak pillar stress
mps^2 - miles per second per second

EXECUTIVE SUMMARY

The objective of the Phase I study was to evaluate the technical feasibility and cost of selected candidate mining systems most adaptable to large scale mining of the deep oil shale deposits of the central Piceance Creek Basin. The study was divided into three investigative sections:

- General review of the geology, hydrology, and resources of a representative mine site in the central portion of the Basin.
- Evaluation of underground rock mechanics design techniques and synthesis of engineering physical properties of Green River oil shale.
- Selection, preliminary design and costing, and evaluation of candidate mining systems.

A resource evaluation of four potential representative mine sites in the central Piceance Creek Basin resulted in the selection of an area near the confluence of Ryan Gulch and Piceance Creek, Rio Blanco County, Colorado. Approximately 1,100 feet of oil shale, averaging 20 gpt and under an average of 1,000 feet of overburden, is projected over most of the 30 square mile site. Significant quantities of dawsonite and nahcolite are also present. An evaluation of the groundwater hydrology indicated an estimated groundwater inflow from 4,000 to 9,000 gpm for a four square mile mined out area.

A review of rock mechanics underground design methods indicated that the most important physical properties for design purposes are compressive strength, tensile strength, deformation modulus, internal friction, cohesion, and Poisson's ratio. A synthesis of the available data on Green River oil shale indicates that as Fischer assay increases, strength decreases. Quantitatively, as Fischer assay increases, compressive and tensile strength, deformation modulus, and internal friction decrease, while cohesion and Poisson's ratio increase. These quantitative data were used in the preliminary design of pillars and roof beams.

In the selection and adaptation of mining systems to deep, thick oil shale deposits the following constraints were considered:

- Mining systems must be capable of a minimum of 85,000 tpd and be easily expandable to twice that amount.
- Mining systems must be well within the health and safety standards of todays mining codes.
- The possibility of the presence of methane gas must be considered.
- The mining system must be capable of handling large quantities of water inflow.
- The underground disposal of spent shale may be considered.
- Resource recovery must be as high as possible.
- Environmental impact should be minimized.
- Equipment used in the designs should be currently available or technically feasible for near future development.

The mining systems evaluated for adaptability to large scale mining of oil shale were: square set stoping, shrinkage stoping, cut and fill stoping, sublevel caving, longwall mining, sublevel stoping, room and pillar mining, and block caving. The systems selected as the most proved and adaptable were sublevel stoping, room and pillar mining, and block caving. Sublevel stoping was modified to include a caving system having a 95% resource recovery and a spent shale backfill design having 55% recovery with the ability to dispose 70% of the spent shale underground. Room and pillar mining was also modified into two designs, advance entry and pillar and chamber and pillar mining. The chamber and pillar mining design had the highest resource recovery of the two and is also specially designed for spent shale backfill. Standard block caving systems, having 300-foot square by 550-foot high blocks were evaluated using both slushers and LHD's and also considering the cost of inducing caving. Production costs, including manpower, capital investment, depreciation, interest, power and water, and preproduction development costs, ranged from $1.04 per ton for chamber and pillar mining to $1.35 per ton for block caving with slushers.

A ranking analysis using the DARE system was conducted considering the following major categories: technical feasibility, mining costs, resource recovery, reclamation, environmental impact, and health and safety. The results of the ranking analysis are as follows with the highest ranking system first:

1. Chamber and pillar mining
2. Sublevel stoping with spent shale backfill
3. Sublevel stoping with full subsidence
4. Block caving using LHD equipment
5. Block caving using slusher equipment
6. Advance entry and pillar mining

SUMMARY OF CONCLUSIONS AND RECOMMENDATIONS

The large scale mining of oil shale by underground methods is technically feasible. The mining systems ranked and selected as most promising are: chamber and pillar mining, sublevel stoping with spent shale backfill, sublevel stoping with full subsidence, and block caving using LHD's.

Recommendations are made to do a more detailed technical analysis and perform an economic evaluation of the four most promising mining systems using data from the representative mine site.

The reserves of oil shale in the general area of the potential mine site not only are more than adequate for the contemplated rates of mining by any of the selected systems, but also contain appreciable amounts of the associated saline minerals nahcolite and dawsonite, which could significantly improve the overall economics of such mining.

SECTION 1

INTRODUCTION

The subject of this report is a technical and economic evaluation of candidate underground mining systems for deep, thick oil shale deposits phase I. This study was awarded by the U.S. Bureau of Mines, under the direction of the Denver Mining Research Center, to Cameron Engineers in July, 1974.

The apparent decline in this nation's proved reserves of easily obtainable hydrocarbons has resulted in a search for alternative fuel sources. A dramatic increase in the cost per barrel of imported oil has also changed the economic climate, resulting in the possibility of recovering higher cost hydrocarbons. One such source of hydrocarbons is found in a sedimentary rock termed kerogenetic marlstone, or commonly, oil shale. The Piceance Creek Basin in Colorado contains the world's largest single known reserve of oil shale that is located within an area of approximately 1,600 square miles. The depth of this resource below the surface ranges from zero to as much as 1,200 feet. It is estimated that if the oil produced from this resource were mined and processed at a rate of 2,000,000 barrels per day, the resource would last more than 400 years.

In an effort to determine the best methods of exploiting this potentially valuable and needed resource the U.S. Bureau of Mines recently funded three research contracts. These contracts involved individual studies on the technical and economic feasibility of mining oil shale by in situ rubblization, open pit, and underground methods. This report concerns the latter contract.

1.1 SCOPE

This contract study is divided into two phases, the second contingent on the results obtained from the first. The first phase (12 months) involves the initial investigation and subsequent engineering evaluation of all potentially feasible systems for large scale, underground mining of the thicker oil shale deposits. The second phase, of 8 to 12 months duration, will be a more detailed technical and economic analysis of the most promising mining system or systems as determined under the initial phase.

The general approach used to achieve the objectives of the phase I study included the following steps:

- A geological, hydrological, and resource evaluation of the Piceance Creek Basin oil shale deposits.
- A review and synthesis of the engineering physical properties of Green River oil shale.
- A review of possible candidate mining systems and selection of the most feasible.
- The preliminary design and costing of the most feasible candidate mining systems.
- A ranking analysis and selection of the most promising mining systems.

Three of the several candidate systems investigated showed sufficient promise for further evaluation by preliminary design and costing. Final selection and ranking was made on the following modifications of these three general systems:

- Sublevel stoping -
  - Sublevel stoping with full subsidence
  - Sublevel stoping with spent shale backfill
- Room and Pillar Mining -
  - Advance entry and pillar mining
  - Chamber and pillar mining
- Block caving -
  - Block caving using slushers
  - Block caving using LHD's

The data used in this study come from publicly available sources and in most cases are insufficient to provide the basis for detailed designs. The majority of the published data is from the U.S. Bureau of Mines, U.S. Geological Survey, and the prototype oil shale mines located near Rifle, Colorado. The limitations of this study are a result of insufficient data on ground water hydrology, structural geology (jointing and faulting), in situ stresses, and the physical properties of oil shale and associated minerals in the central portion of the Piceance Creek Basin.

1.2 CONTENT

This report includes a discussion of the geology and hydrology of the Piceance Creek Basin, a resource evaluation of a potential mine site, and the synthesis of published data on the engineering physical properties of Green River oil shale. Following these sections are the contract studies, including the review, selection, design and costing of the candidate mining systems. The report closes with a ranking analysis of the mining designs, the conclusions and recommendations, a phase II outline, appendices, and references cited.

SECTION 2

GENERAL GEOLOGY AND HYDROLOGY OF THE OIL SHALE DEPOSITS IN THE CENTRAL PICEANCE CREEK BASIN

The Piceance Creek Basin includes about 1600 square miles in Garfield and Rio Blanco counties in northwestern Colorado, Figure 2.1. It is a northwest trending structural basin which ranges topographically from 1000 to 4000 feet above the surrounding lowlands. Elevations in the basin are from about 6000 to 9400 feet above sea level with parts of the basin margin being defined by precipitous escarpments. The land surface within the basin consists of a series of steep sided ridges and valleys that slope down from the margins toward the basin interior.

Precipitation, as measured in small towns in the surrounding lowlands, averages about ten inches annually and from 15 to 25 inches at elevations above 7000 feet.

The Roan Plateau, an east-west trending structure in the southern part of the basin, forms a drainage divide between tributaries of the Colorado River to the south and the White River to the north. The northern drainage system is the larger of the two and the principal streams in this drainage are Piceance Creek and Yellow Creek. Principal streams in the southern drainage system are Parachute Creek and Roan Creek.

Vegetation over most of the interior of the basin is sparse and consists of sage, pinon, and juniper. On the north and east margin of the basin, north-facing slopes and other isolated patches are covered by heavy growths of scrub oak and buck bush. There are some scattered heavy stands of aspen, lodge-pole pine, and blue spruce on the Roan Plateau.

Major towns in the vicinity include Grand Junction, Rifle, Meeker, and Rangely, Colorado.

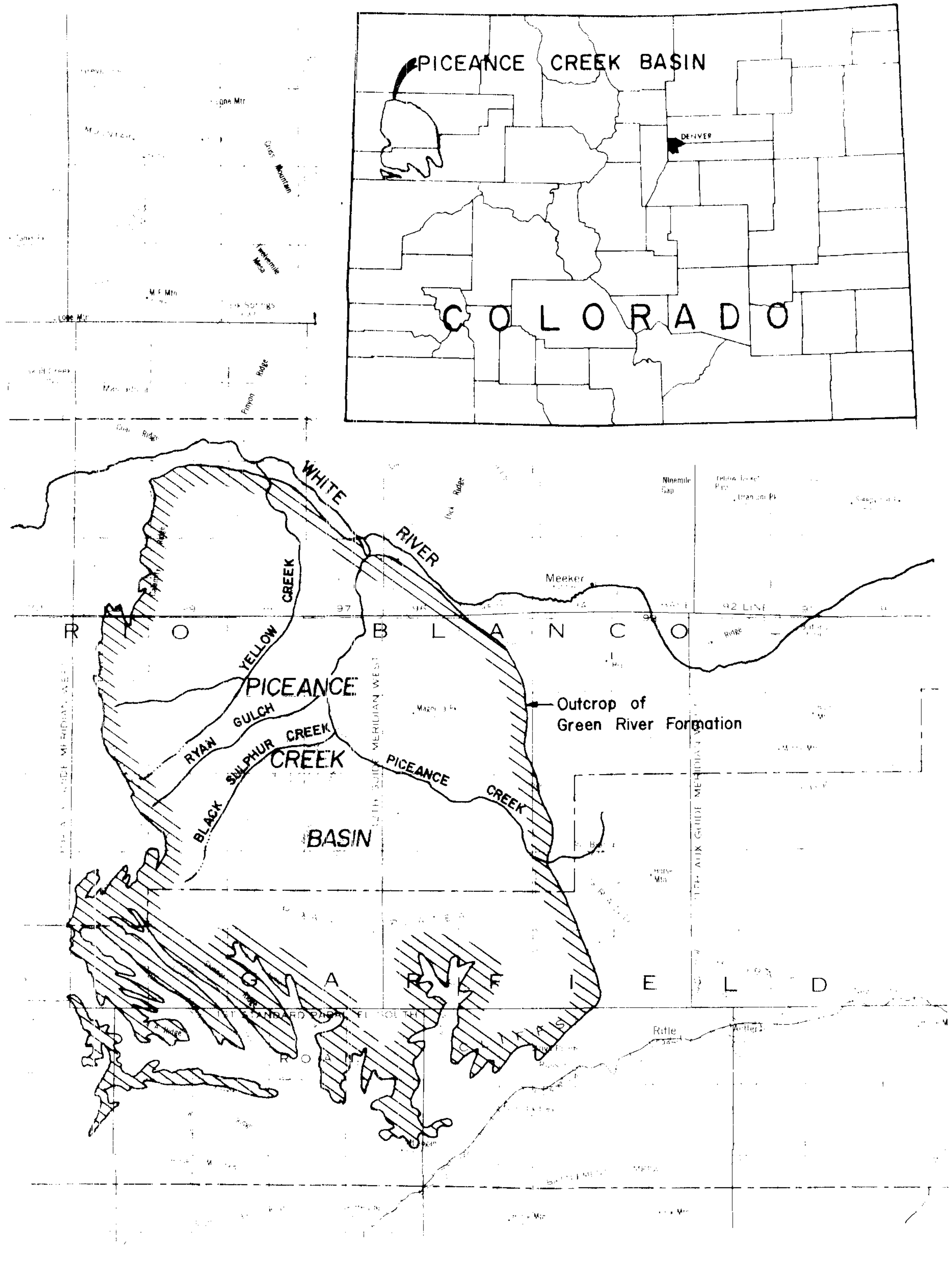


Figure 2.1 Location Map of Piceance Creek Basin, Northwest Colorado

2.1 STRATIGRAPHY

Strata exposed in the Piceance Creek Basin range in age from Late Cretaceous to Tertiary. The oldest rocks are the Late Cretaceous Mesaverde group which is composed of sandstones, shales, and some coal beds. These rocks are resistant to erosion and form a series of prominent benches which outcrop continuously around the southern and eastern margin of the basin. Fossil content indicates that deposition occurred near strandlines in fresh, brackish, and saltwater environments. Overlying the Mesaverde is the Paleocene Series, consisting of the Ohio Creek Conglomerate and an unnamed unit composed of feldspathic sandstones, shales, and thin coalbeds which are considered Fort Union Formation equivalents. Above the unnamed unit is the Wasatch Formation which forms the lowlands between the more resistant strata of the Mesaverde and the overlying Green River Formation. The Wasatch Formation consists of a thick sequence of lenticular sandstones and red, purple, gray, green, and yellow shales with some coal beds. Numerous fossils indicate that the Wasatch strata were deposited in a fluviatile environment. Above the Wasatch Formation is the Green River Formation, deposited during the middle Eocene Epoch of the Tertiary Period. A more detailed description of the Green River Formation and the overlying Uinta Formation is shown in Figure 2.2.

2.1.1 Green River Formation

The Green River Formation is composed of kerogenetic marlstones, marlstones, shales, sandstones, siltstones, and limestones which have been divided into three distinct members based on lithology. The lowermost, or basal unit, is designated as the Douglas Creek Member and is composed predominantly of sandstone, limestone, and shale. Above the basal unit is the Garden Gulch Member which is composed of dark, finely laminated shale. The upper unit, the Parachute Creek Member, is composed largely of kerogenaceous dolomitic marlstones, or commonly, oil shale.

In the eastern and southeastern part of the basin a clastic facies equivalent to the Douglas Creek, Garden Gulch, and lower Parachute Creek members has been designated the Anvil Point Member by Donnell (1961).

[illegible]			Thickness vari[illegible] from 0 to 1,200 feet	Present as cap rock over most of the basin. Marlstone, sandstone[illegible], shale and siltstones.
[illegible] RIVER FORMATION	[illegible] Creek Member	[illegible] Sequence	Thickness varies from 400 to 600 feet	The Mahogany Zone is at the base of the upper sequence. The upper sequence is present throughout the basin. Both sequences are composed of dolomitic to calcareous kerogenitic marlstones (oil shale). The lower sequence contains significant amounts of [illegible]colite, halite and daw[illegible] in the central part of the basin.
		[illegible] Sequence	Thickness varies from 0 to 1,800 feet	
	[illegible] Member		Thickness varies from 0 to 900 feet	With the exception of the eastern part of the basin this member is present everywhere. It consists of dark, finely laminated shales (illite) and marlstone both of which contain some kerogen.
	[illegible] Creek Member		Thickness varies from 0 to 800 feet	This member is present only in the southern, western and central parts of the basin. It consists of sandstones, shales, and limestones. Contains minor oil shale.
	Anvil Point Member		[illegible] from 0 to 1,870 feet	This member is present only in the [illegible] and southern part of the basin. It is composed of sandstones, shales, siltstones, marlstones and limestones.

Figure 2.2 Generalized Description of Green River Formation and Overlying Uinta Formation, Piceance Creek Basin, Colorado

Overlying the Green River Formation is the Uinta Formation that includes sandstones and siltstones which were formerly designated as the Evacuation Creek Member. With the exception of a few local structural disconformities observed by Donnell (1961), these formations represent a continuous depositional sequence.

2.1.1.1 Douglas Creek Member

The Douglas Creek Member is composed of sandstones, shales, and limestones that conformably overlie the variegated shales and sandstone units of the Wasatch Formation. This member has been recognized only in the southern, western, and central parts of the basin. In the eastern part of the basin the clastic facies of the Douglas Creek Member are assigned to the Anvil Point Member. To the northwest, the Douglas Creek Member is coalescent with shales of the overlying Garden Gulch Member. The maximum reported thickness of the Douglas Creek Member is about 800 feet in the southwestern part of the basin. Resistant strata of the Douglas Creek Member form a series of benches along the base of the Green River escarpment. The color of the outcrop commonly ranges from brown to buff.

2.1.1.2 Garden Gulch Member

The Garden Gulch Member is composed of dark, finely laminated shales and marlstone, some of which contain kerogen. With the exception of the eastern part of the basin where equivalent sandy beds are assigned to the Anvil Point Member, the Garden Gulch Member is present throughout the basin. The thickest sequence of the Garden Gulch Member is 900 feet in the northwestern part of the basin, where it conformably overlies the Wasatch Formation. From there southward, the member thins. The Garden Gulch outcrops as a gray, steep slope between the white cliffs of the Parachute Creek Member and the brown and buff benches of the Douglas Creek Member. Both the upper and lower contacts of the Garden Gulch Member in the subsurface are well defined on resistivity logs. The lower contact, defined as the base of the low resistivity zone is known as the Orange Marker (Roehler, 1974). A similar low resistivity zone, *Lower Oil Shale Zone.* The lower oil shale zone extends from the top of the Garden Gulch Member to the base of the Mahogany Zone. In the southeastern part of the basin the lower zone is interfingered with, and in some instances replaced by, the Anvil Point Member. In the northwest part of the basin the lower zone is absent and the upper oil shale zone rests conformably on the Garden Gulch Member, Donnell (1961). Elsewhere the thickness of the lower zone ranges from a minimum of 20 feet in the southwest to over 1300 feet in the north-central part of the basin. The thickening toward the center of the basin is attributed in part to increased organic content and the occurrence of bedded and disseminated evaporites.

Cashion and Donnell (1972) subdivided the lower oil shale zone into a series of alternating lean and rich zones. This designation is based on shale oil yields from the modified Fischer assay. The lower zone can also be subdivided into two zones, leached and unleached, on the basis of removal by dissolution or leaching of the water soluble minerals nahcolite and/or halite. Although leaching is not confined to any one area of the basin, the characteristic low resistivity zone produced on electric logs in the leached section is confined to the central part of the basin in the area of maximum evaporite deposition. The unleached or saline zone is the lower of the two. This zone contains possible economic deposits of nahcolite and dawsonite. The thickness of the unleached zone in the central part of the basin ranges from about 500 to over 1100 feet.

The interface of the leached and unleached zone is a somewhat poorly defined plane termed the dissolution surface. In reality, it is probably a dissolution zone, as numerous cores have shown nahcolite to be present above the dissolution surface while vugs and cavities are found below. At best, any designated dissolution surface is an approximation.

Above the dissolution surface is the leached zone which is characterized by cavities, vugs, and collapse breccia. The thickness of the leached zone in the central part of the basin varies from about 400 to over 1100 feet. Leaching of the evaporites from the lower zone has upgraded the oil shale, produced an aquifer, and reduced the bulk strength of the rock.

Upper Oil Shale Zone. The upper oil shale zone is variable in thickness, ranging from 500 to 600 feet in the south to 400 feet in the extreme northeast part of the basin. From the southern margin northward, this zone interfingers with the overlying Uinta Formation and decreases in thickness.

The most consistantly rich oil shale zone in the basin is within the lower 100 to 200 feet of the upper oil shale zone. This zone, designated the Mahogany Ledge in the outcrop and Mahogany Zone in the subsurface is a readily identifiable unit (Bradley, 1931). The very resistant nature of the Mahogany Zone results in a precipitous outcrop. The Mahogany Zone is also bounded on the top and bottom by barren zones termed the 'A' and 'B' Grooves, respectively. These barren zones are less resistant to erosion, forming slopes on outcrops that resemble grooves, hence their designation. In the subsurface both the 'A' and 'B' Grooves produce distinct lows on resistivity logs (the red and black markers respectively). The thickness of the Mahogany Zone varies from approximately 100 feet near the margin of the basin to about 200 feet in the north central part of the basin. The thickening is largely due to an increase in clastic content. Generally, the grade of oil shale in the Mahogany Zone exceeds 20 gpt with the richest sequence comprising approximately 130 feet of 30 gpt shale.

Above the Mahogany Zone is 300 to 500 feet of leaner oil shale. This sequence is thickest in the southern part of the basin, and due to interfingering with the overlying Uinta Formation, thins to less than 300 feet in the northeast.

2.1.2 Uinta Formation

The Uinta Formation, formerly the Evacuation Creek Member, consists of barren marlstones, shales, siltstones, and sandstones overlying most of the basin. The outcrop of the Uinta Formation forms a buff to light brown rounded cap receeding from the white cliffs of the Parachute Creek Member.

The maximum thickness of the Uinta Formation is unknown because the top has been removed by erosion; however, from subsurface information it is known to exceed 1200 feet. Because of the lenticular nature of the siltstones and sandstones, the lower boundary varies from location to location. With the exception of a few local disconformities the Unita Formation rests conformably on the Parachute Creek Member of the Green River Formation.

2.2 STRUCTURE

The Piceance Creek Basin is a large, assymetrical, northwest trending, structural downwarp. Beds that dip into the basin range from less than three degrees on the southern margin to 27 degrees on the northern rim, Figure 2.4. The basin is bordered on the east and the northwest by the White River Uplift and the Axial Basin anticlines, respectively. On the northwest edge, the basin axis is a syncline located between the Massadona Anticline and the Rangely Anticline, extending south from the western boundary of the Piceance Creek Basin. The southern boundary is formed by the Uncompaghre Uplift southeast of Grand Junction, not shown in Figure 2.4.

Within the basin itself there are a number of small, parallel, northwest trending anticlines. Among these local structures, the most prominent is the Piceance Creek Dome located in the northeastern part of the basin.

Numerous northwest trending, high angle normal faults of small displacement are present northwest of the Piceance Creek Dome. The faults commonly occur in pairs and usually bound downthrown blocks or grabens. Faulting on the crest and flanks of plunging anticlines are probably a result of failure due to tensional stress exerted by folding of the strata after deposition and lithification. A series of these faults form en echelon and parallel graben systems.

2.3 GEOLOGIC HISTORY

The geologic history of the Piceance Creek Basin is reflected in the cross sections shown in Figures 2.5 and 2.6 (Roehler, 1974). The Piceance Creek Basin was a center of deposition at the beginning of the period of deposition of the Green River Formation. This is reflected in the basinward thickening of the underlying Wasatch Formation. About 50 million years ago in the Eocene Epoch drainage outlets for the basin

OUTCROP SYMBOL

Base of Eocene Green River Formation

STRUCTURE SYMBOLS

Anticline - showing direction of plunge

Syncline - showing direction of plunge

Monocline

Normal Fault - bar and ball on downthown side

KEY TO NUMBER SELECTED STRUCTURES

1. Red Wash syncline (stucturally deepest part of Piceance Creek basin on troughline near number)
2. Piceance Creek dome
3. South Rangely syncline
4. Sulphur Creek anticlinal nose
5. Hunter Creek syncline
6. Douglas Creek anticline
7. Crystal Creek anticlinal nose
8. Clear Creek syncline
9. Rangely anticline
10. White River dome
11. Massadona anticline
12. Axial Basin anticline
13. Grand Hogback monocline

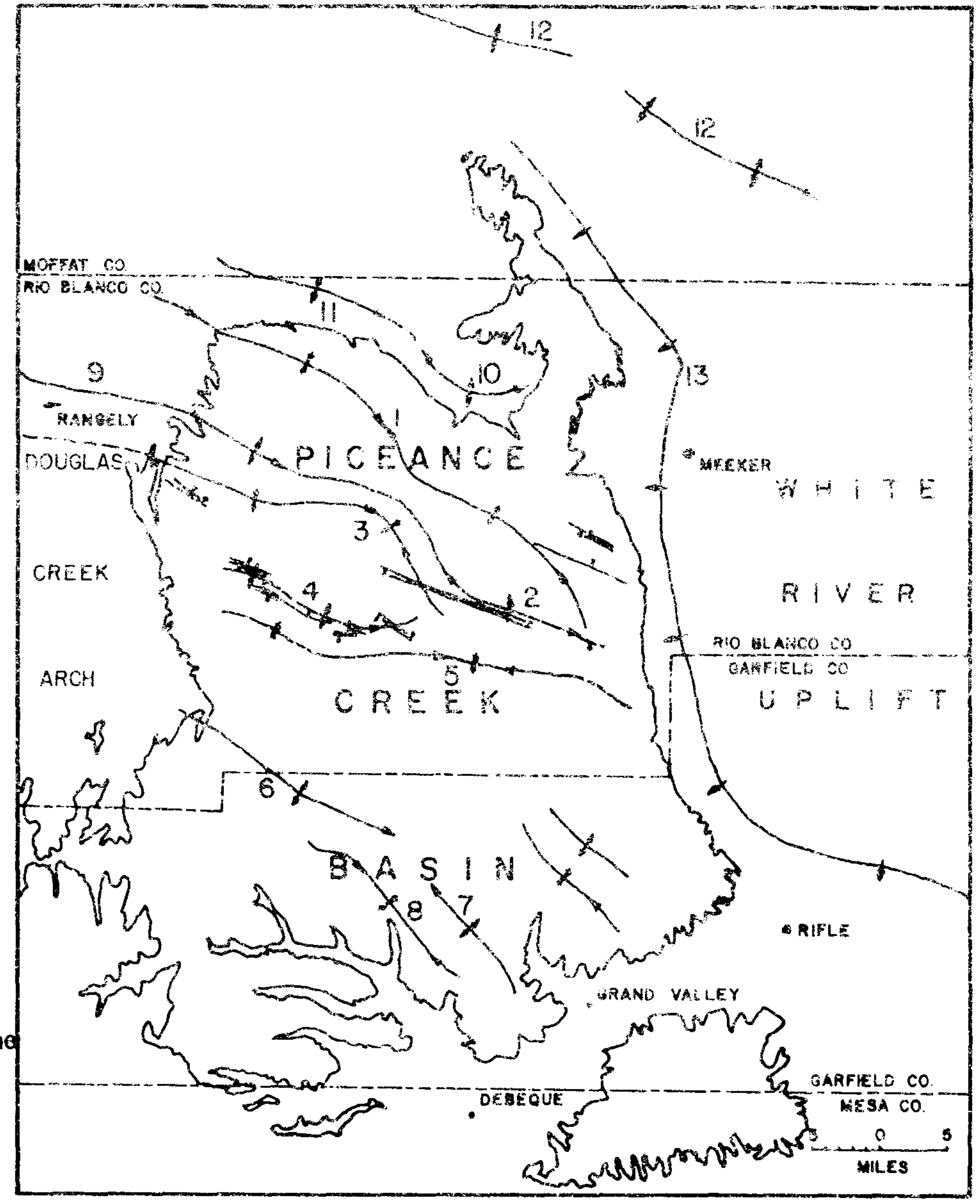


Figure 2.4 Tectonic Map of Piceance Creek Basin, Colorado -- from Murray and Haun (1974)

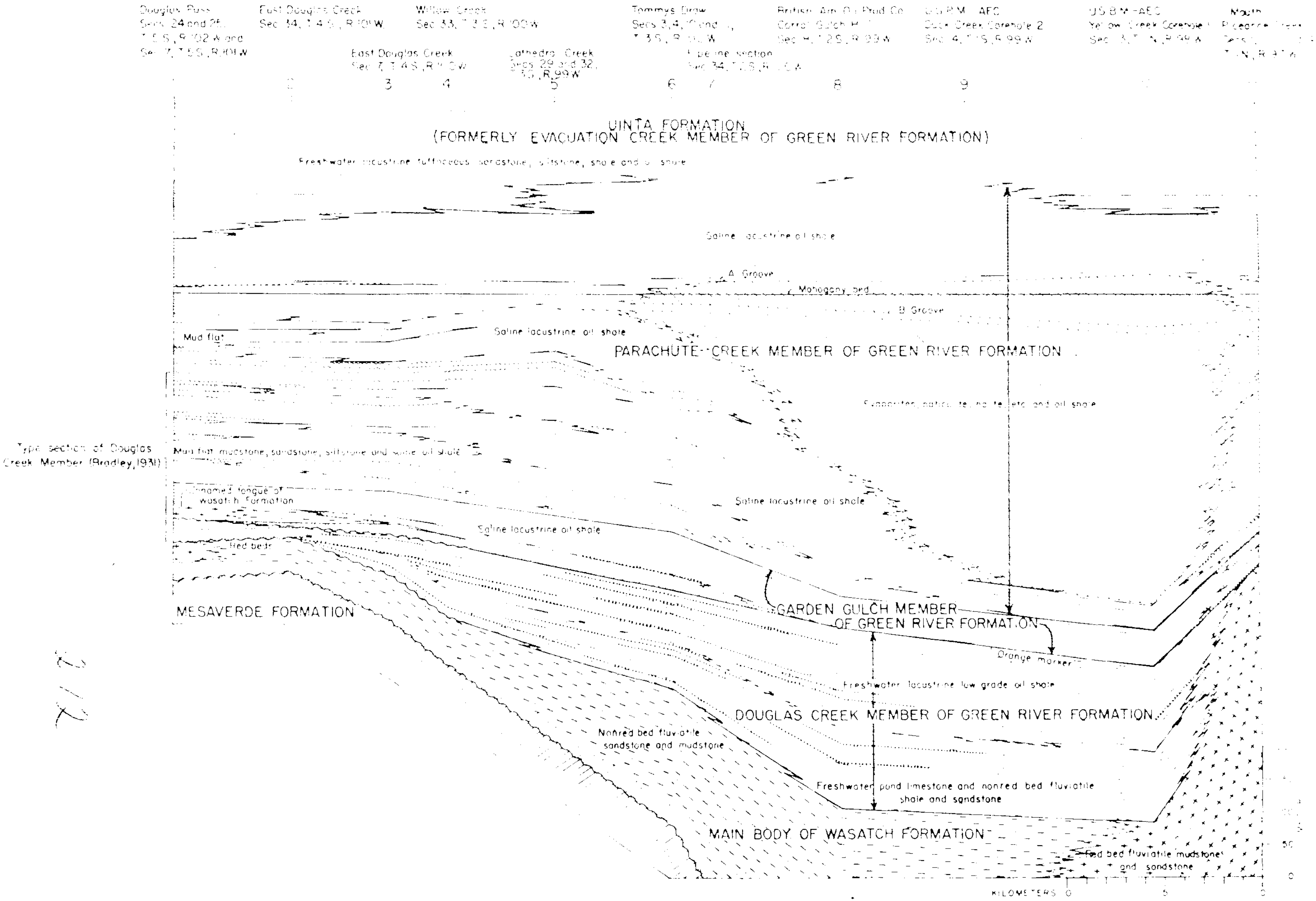


Figure 2.5 SW-NE Cross section of Eocene Rocks in the Piceance Creek Basin, Colorado -- from Roehler (1974)

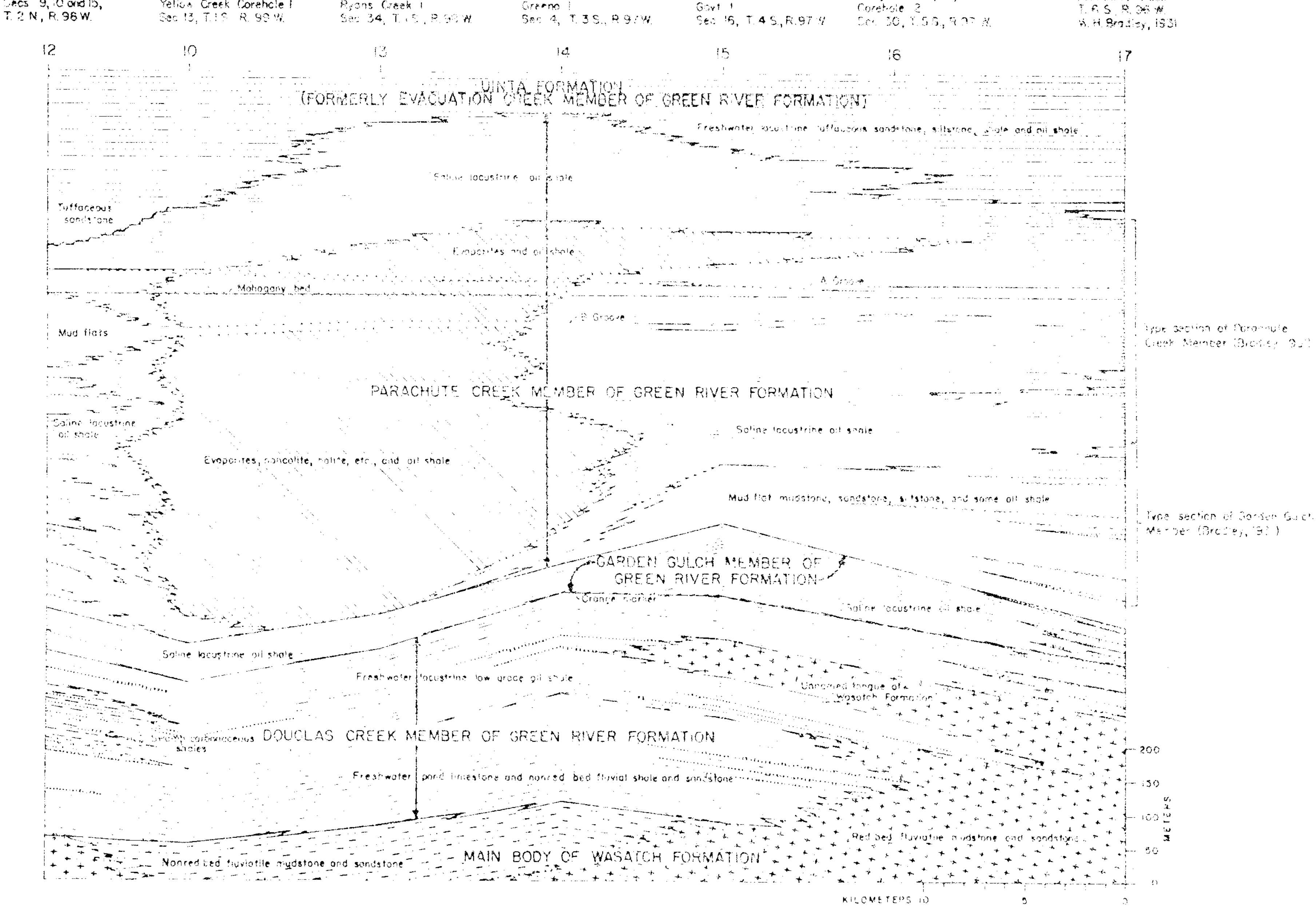


Figure 2.6 NW-SE Cross Section of Eocene Rocks in the Piceance Creek Basin, Colorado -- from Roehler (1974)

were effectively blocked, possibly by subsidence of the basin, uplift of the adjacent areas, or a combination of both. During early Douglas Creek time drainage into the basin produced fluviatile and paludal conditions as reflected in the strata in the lower part of the Douglas Creek Member. These strata consist of fresh water limestones, gray and brown carbonaceous shales, thin beds of coal, and some gray lenticular sandstones. The drainage into the basin was from the east and southeast. This condition continued throughout most of the depositional history of the Green River Formation and is reflected in the clastic facies of the Douglas Creek, Garden Gulch, and lower Parachute Creek Members, designated as the Anvil Point Member. Eventually the numerous small ponds coalesced to form a large fresh water lake called Lake Uinta. This lake continued to expand throughout the remaincer of the period of deposition for the Douglas Creek Member. The upper part of the Douglas Creek Member is composed of strata that were deposited in a fresh water lake. These strata are usually light brown, varved, papery, kerogenaceous shale with the high organic content derived from the seasonal mortality of blue-green algae.

At the beginning of Garden Gulch time the climate altered and the lake changed from freshwater to saline. Sediments deposited in the saline conditions were limey organic muds. This change is recorded in the organic content of the shale in the Garden Gulch Member, which is generally higher than the organic content of the freshwater shale of the Douglas Creek Member. Brines settled to the deeper part of the lake during extremely quiescent conditions. During Garden Gulch time the carbonate content of the lake waters also increased, setting the stage for the deposition of the Parachute Creek Member.

Deposition during Parachute Creek time was characterized by vast quantities of kerogen-rich, dolomitic to calcareous marlstone with bedded evaporites toward the center of the basin. During this time the lake expanded to its maximum size as indicated by the widespread Mahogany Zone.

As a result of widespread volcanic activity to the northwest, climatic conditions changed and a decrease in precipitation and temperature resulted in the disappearance of Lake Uinta. This is recorded by the numerous tuffaceous rocks present in the upper part of the Parachute Creek Member and in the overlying Uinta Formation. Subsequent post-depositional tectonics and erosion has produced the topographic features of the Piceance Creek Basin as seen today.

2.4 ECONOMIC RESOURCES OF THE PICEANCE CREEK BASIN

The three economic resources of interest in the Green River Formation of the Piceance Creek Basin are shale oil, nahcolite, and dawsonite. All three of these are found in significant quantities in the Parachute Creek Member.

2.4.1 Shale Oil

Shale oil is the product of the destructive distillation of the kerogen in oil shale. In the central part of the basin the thickest and richest oil shale sequence is found in the lower part of the Parachute Creek Member. Recently, a new system of classification for total resources was adopted by the USBM and the USGS. Key criteria for this classification are the extent of geologic knowledge of the resource and the economic feasibility of recovery. An estimate for the total shale oil resource of the Piceance Creek Basin is 1,200 billion barrels (National Petroleum Council). Of this, 117 billion barrels are contained in favorably located and defined deposits averaging 30 to 35 gallons per ton over a continuous interval of 30 feet or more. Less favorably situated and not as well defined deposits of the same grade and thickness are thought to contain another 167 billion barrels of oil. Poorly defined deposits of lower grade, ranging down to 15 gallons of oil per ton constitute the remaining 916 billion barrels.

2.4.2 Nahcolite Resources

Nahcolite ($NaHCO_3$) is considered a potentially valuable by-product of oil shale processing due to the large quantities that exist in the central part of the basin and its postulated use in emission control systems of coal-fired, electrical power generation plants. Although nahcolite is found elsewhere in the basin, the significant deposits are restricted to the lower part of the Parachute Creek Member in the central part of the basin. Nahcolite occurs as: (1) fine crystals disseminated in the oil shale matrix or in fractures, (2) as roughly spherical masses called rosettes, or (3) as coarse to microcrystalline, bedded deposits. The coarsely crystalline, bedded variety is associated with halite ($NaCl$). Beard, Tait and Smith (1974) estimate about 29 billion tons of nahcolite are present in the center of the basin.

2.4.3 Dawsonite Resources

Dawsonite ($Na_3Al(CO_3)\cdot 2Al(OH)_3$) resources of the Parachute Creek Member of the Green River Formation have been estimated by Beard et al, (1974) to be 19 billion tons (6.5 billion tons of alumina). Dawsonite could be produced as a by-product in the oil shale retorting process as envisioned by Weichman (1974). In light of recent price increases by the major bauxite producing countries, the development of dawsonite resources may be economically attractive.

Dawsonite occurs finely disseminated in the oil shale matrix. Significant dawsonite deposits are apparently confined to the north-central part of the basin and are restricted to an interval ranging from 1000 to a little less than 300 feet. Reserve estimates range from more than 120 million to less than 20 million tons per square mile.

2.5 RESOURCE EVALUATION OF PROPOSED MINE SITE

A resource evaluation of four potential mine sites, proposed by the USBM, was made to determine the most favorable location for a representative mine site. Figure 2.7 is a map showing the location of the proposed sites Each site was evaluated for shale oil content, nahcolite and dawsonite content, overburden thickness, oil shale thickness, and leached and unleached zone thickness. On the basis of these evaluations, the Bureau of Mines selected site 2 for more detailed evaluation. The following discussion covers the detailed evaluation of site 2 only.

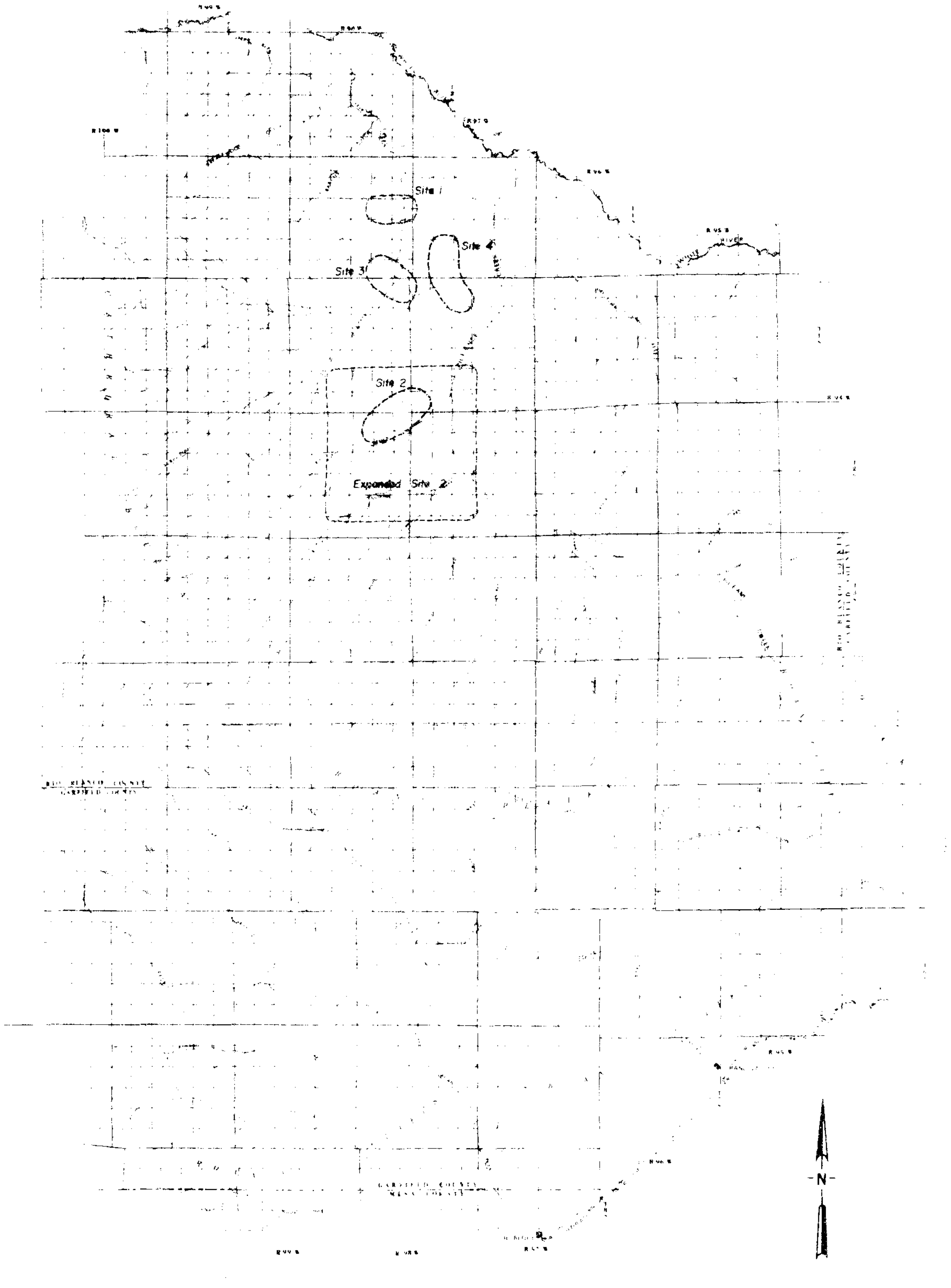


Figure 2.7 Location Map of Potential Mine Sites

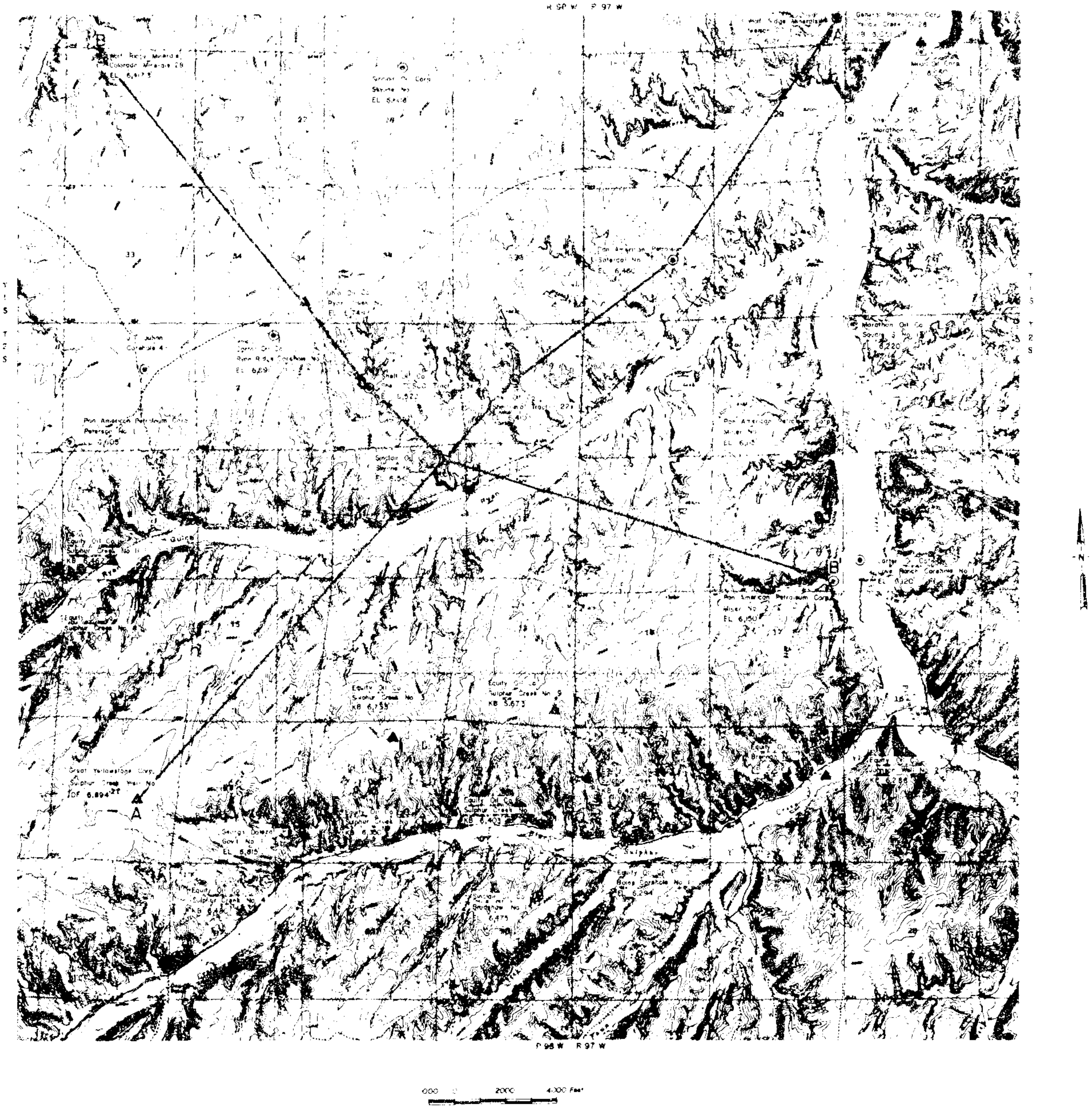
2.5.1 Site 2 Evaluation

Site 2, as originally selected includes about five square miles in portions of T1S and T2S, and R97W and R98W, Figure 2.8. Corehole data are available from Shell Oil Company coreholes 22x and 23x, (Tables 2.1 and 2.2), Pan American Saterdal No. 1, and Sinclair Skyline No. 2. The USBM has since designated this site as the primary area of interest and it has been expanded to include about 30 square miles in portions of the previously mentioned townships.

Figure 2.9 is an overburden thickness map to the top of the Mahogany Zone. The overburden thickness ranges from over 1400 feet in the northwest to less than 600 feet in the southeast with the minimum thickness along Piceance Creek. The overburden consists of the oil shales in the upper part of the Parachute Creek Member, and sandstone, siltstones, and marlstones of the overlying Uinta Formation.

The structural contours on top of the Mahogany Zone, Figure 2.10, indicate the southern boundary of the area is delineated by a system of grabens. Another graben is shown in the northeast portion of the area. This graben is a northwest extension of the same graben found on the Piceance Creek Dome, located to the east of site 2. The dip of the strata is generally about one degree to the north with local variations due to folding and faulting.

Figure 2.11 shows that the thickness of the total oil shale interval from the base of 'A' Groove to the top of the Blue Marker varies from over 1,500 feet in the north to less than 1,100 feet near the southern boundary. The thinning is only partially a depositional feature as removal of the saline minerals by dissolution and subsequent collapse of former saline rich zones, has decreased the thickness as shown in Figure 2.12. The effects of leaching on the total oil shale interval can be seen in Figures 2.13 and 2.14 which show a relatively rapid thickening and thinning of the leached and unleached zone, respectively, away from the north central part of the area.



LEGEND

— Corehole - Information publicly available

— Corehole - Information not publicly available

— Assayed Well - Information publicly available

— Assayed Well - Information not publicly available

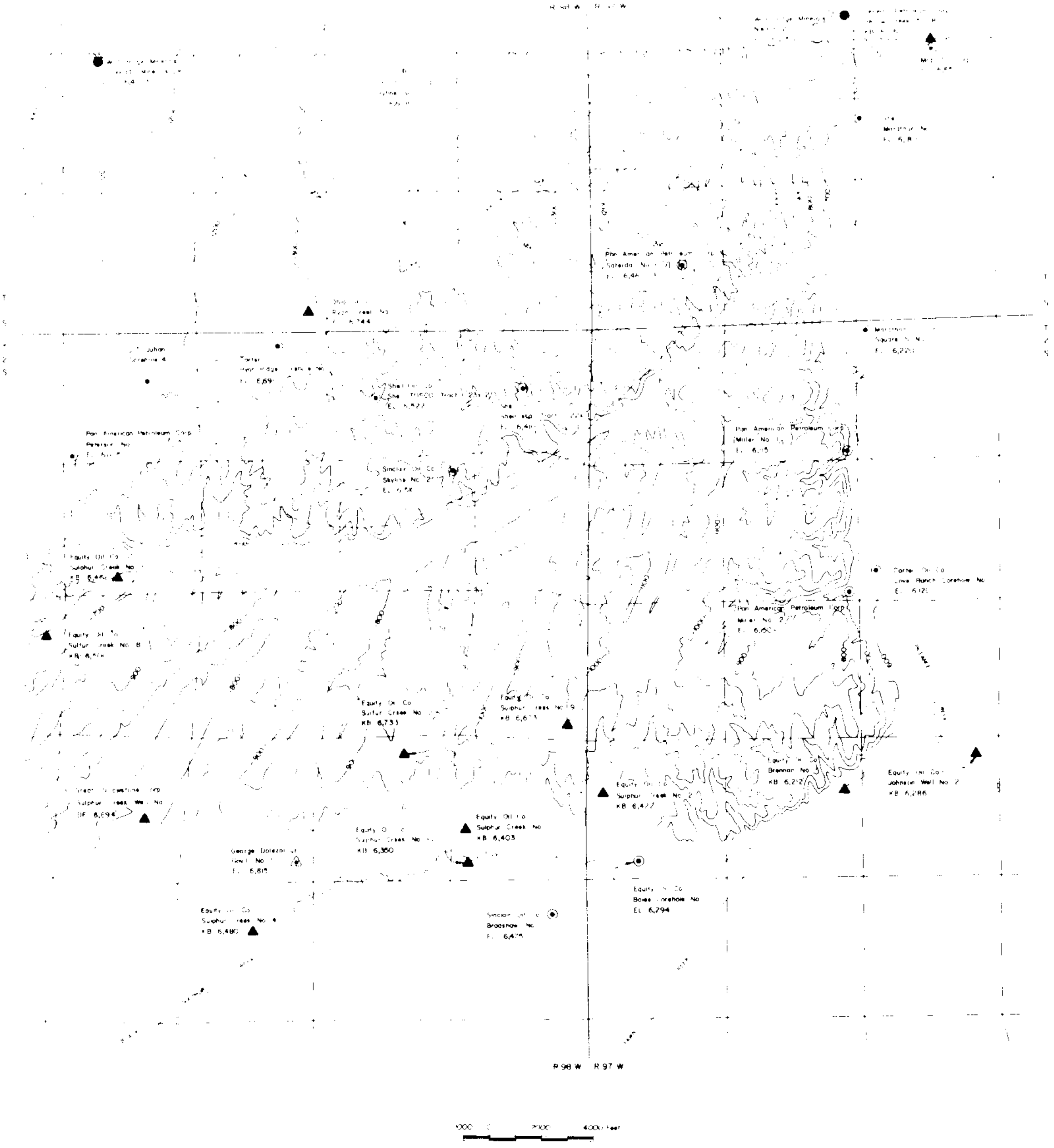
Figure 2.8 Topographic Map of Potential Mine Site 2

Table 2.1 Shell Oil Company Corehole Number 23x-1, Rio Blanco County, Colorado, Sec. 1, T2S, R98W

Zone	Depth Below Surface (ft.)	Elevation of Top (ft.)	Thickness (ft.)	Grade (gpt)
Mahogany	977	5,443	175	27.7
'B' Groove	1,153	5,258	17	4.7
R-6	1,169	5,240	171	24.7
L-5	1,339	5,071	155	11.6
R-5	1,494	4,915	344	18.7
L-4	1,838	4,571	142	18.2
R-4	1,980	4,429	161	33.0
L-3	2,141	4,268	23	16.7
R-3	2,164	4,245	149	24.9
L-2	2,313	4,096	21	15.6
R-2	2,334	4,075	79	34.6
[illegible]e Marker	2,413	3,996	---	---

Table 2.2 Shell Oil Company Corehole Number 23x-2, Rio Blanco County, Colorado, Sec. 2, T2S, R98W

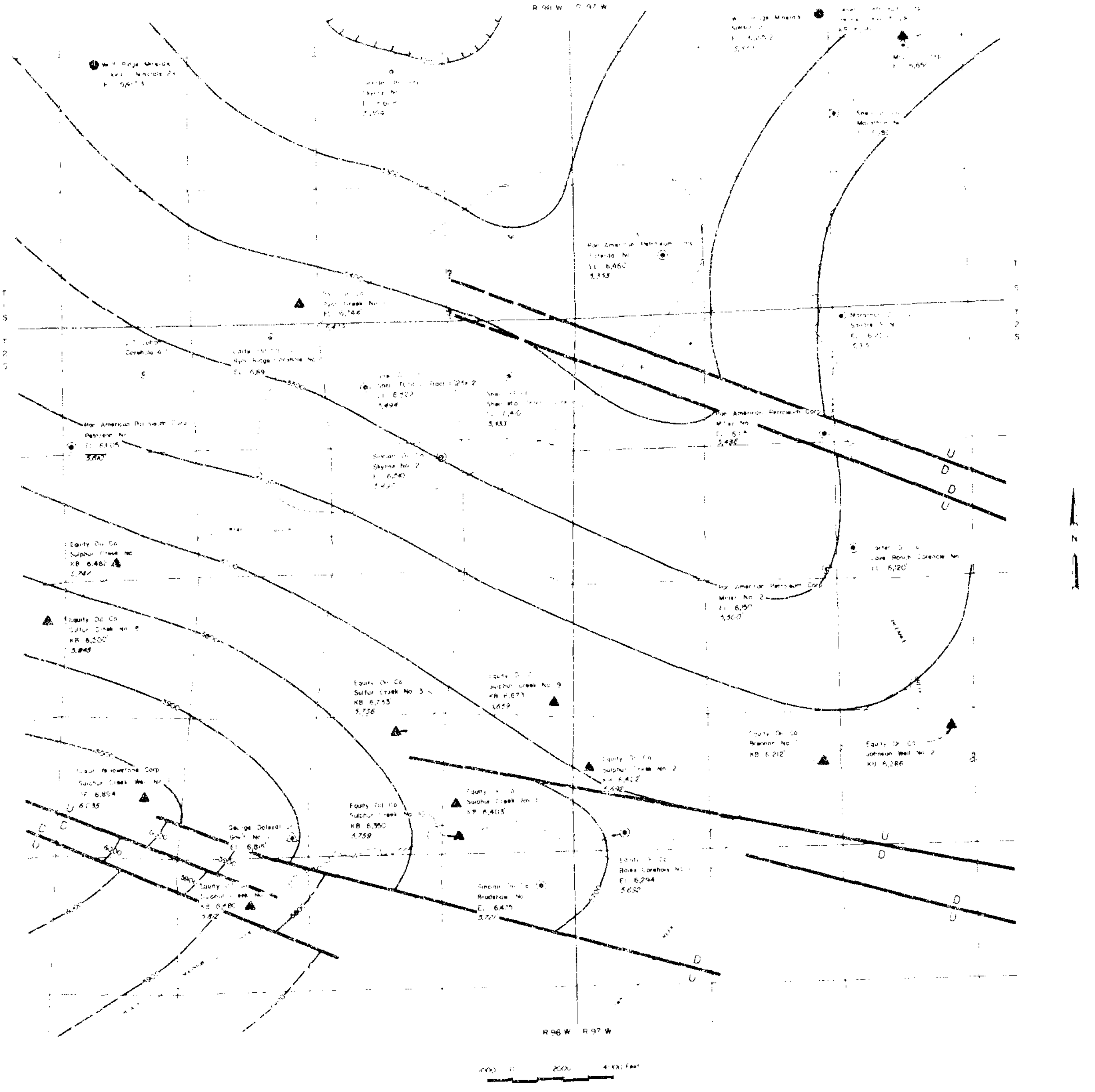
Zone	Depth Below Surface (ft.)	Elevation of Top (ft.)	Thickness (ft.)	Grade (gpt)
Mahogany	1,040	5,492	178	27.5
'B' Groove	1,218	5,314	21	5.7
R-6	1,239	5,293	167	24.1
L-5	1,406	5,126	108	16.6
R-5	1,514	5,018	362	18.9
L-4	1,876	4,656	150	19.1
R-4	2,026	4,506	164	37.1
L-3	2,190	4,342	25	21.3
R-3	2,215	4,317	159	24.8
L-2	2,374	4,158	28	18.7
R-2	2,402	4,130	88	34.9
Blue Marker	2,490	4,043	---	---



L E G E N D

- Corehole - Information publicly available
- Corehole - Information not publicly available
- Assayed Well Information publicly available
- Assayed Well - Information not publicly available
- 1000 - Thickness Contour Line - Contour Interval 100 feet

Figure 2.9 Isopach Map of Overburden to Top of Mahogany Zone, Site 2



LEGEND

- Corehole - Information publicly available
- Corehole - Information not publicly available
- Assayed Well - Information publicly available
- Assayed Well - Information not publicly available
- Structural Contour Line - Contour Interval 100 feet
- U D — Fault
- 5,762' — Elevation of Top of the Mahogany

Figure 2.10 Structure Contour on Top of Mahogany Zone, Site 2

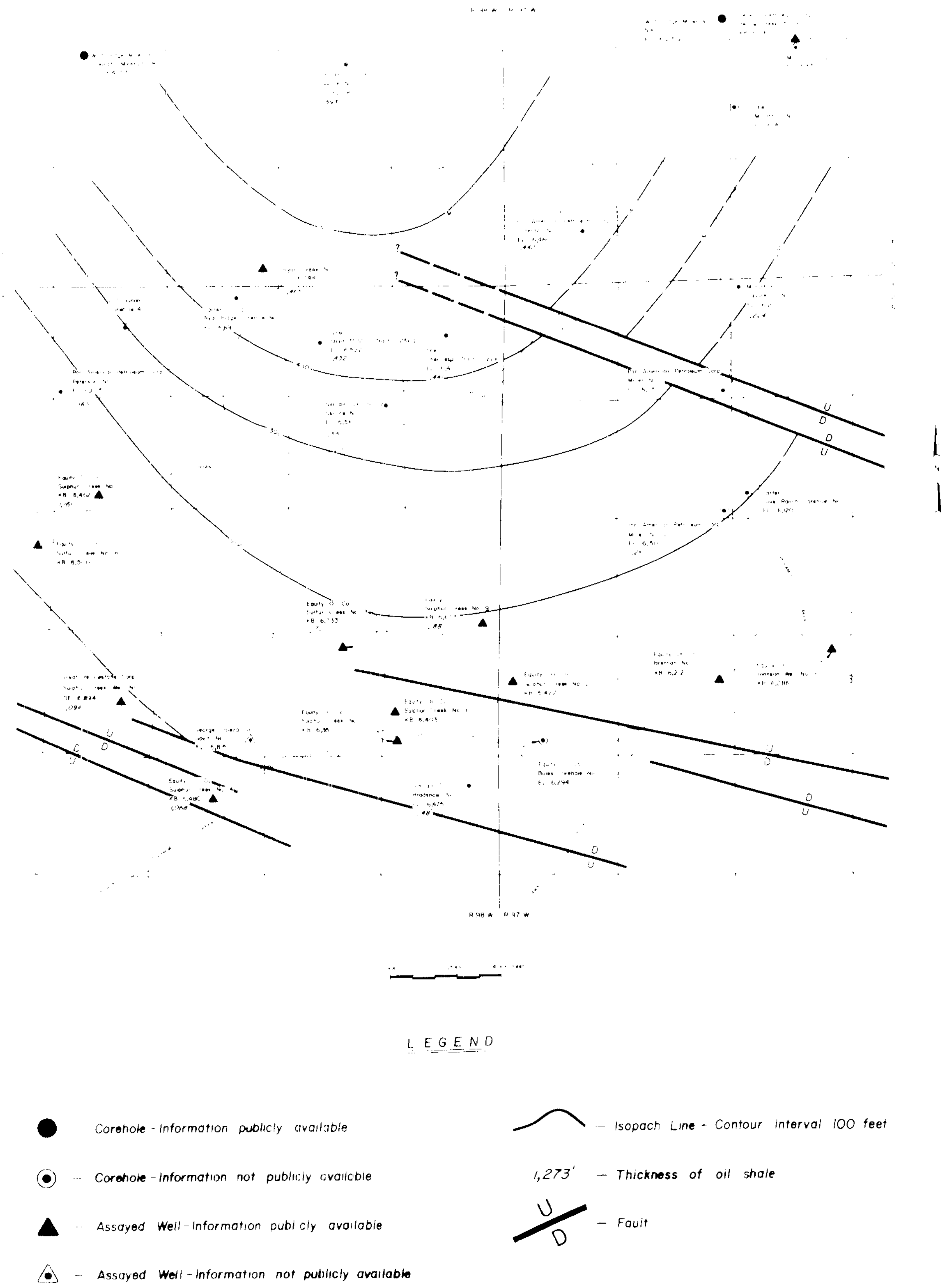


Figure 2.11 Isopach Map of Oil Shale Interval from Bottom of 'A' Groove to Top of Blue Marker, Site 2

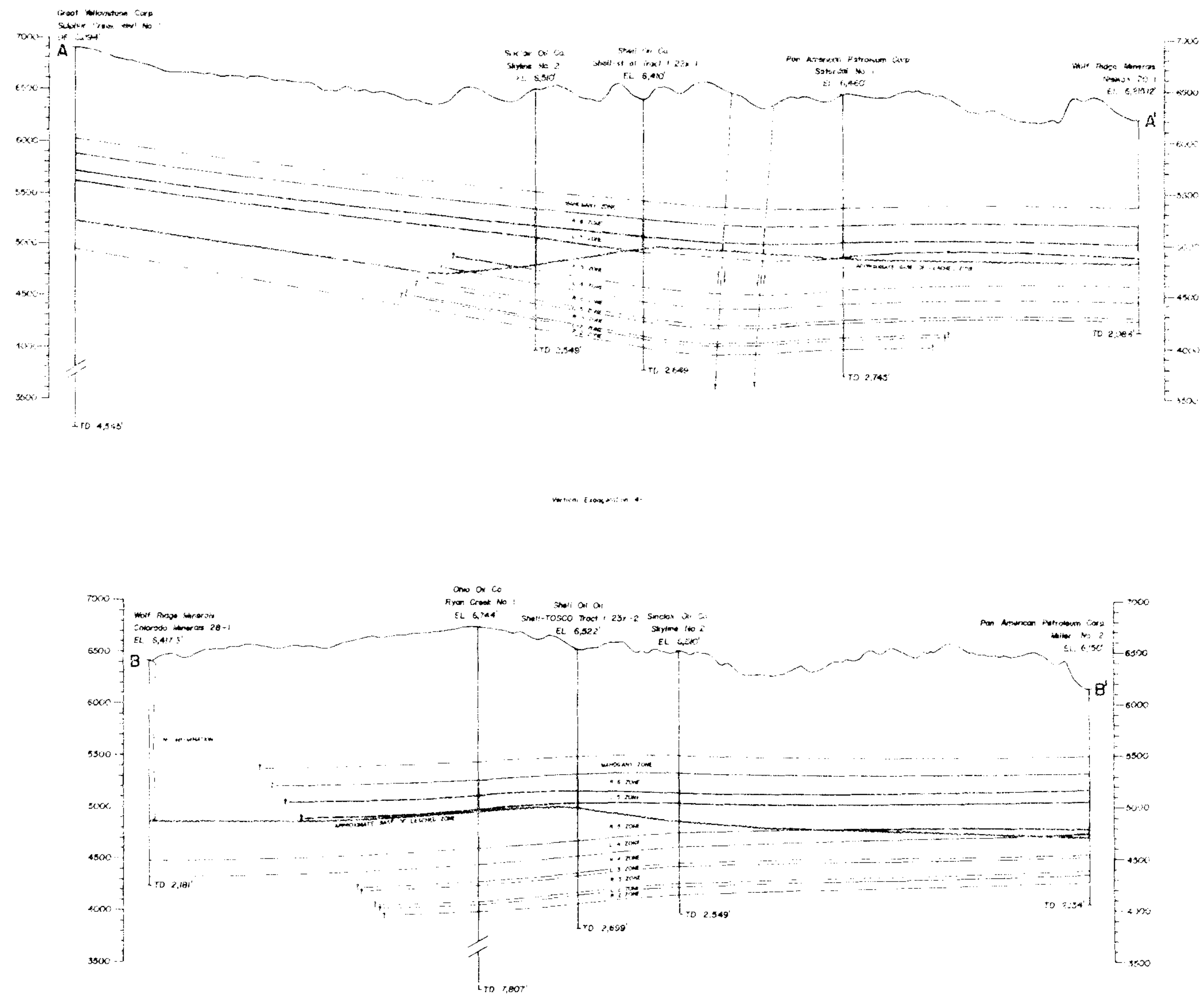


Figure 2.12 Cross Sections Along A-A' and B-B', Site 2

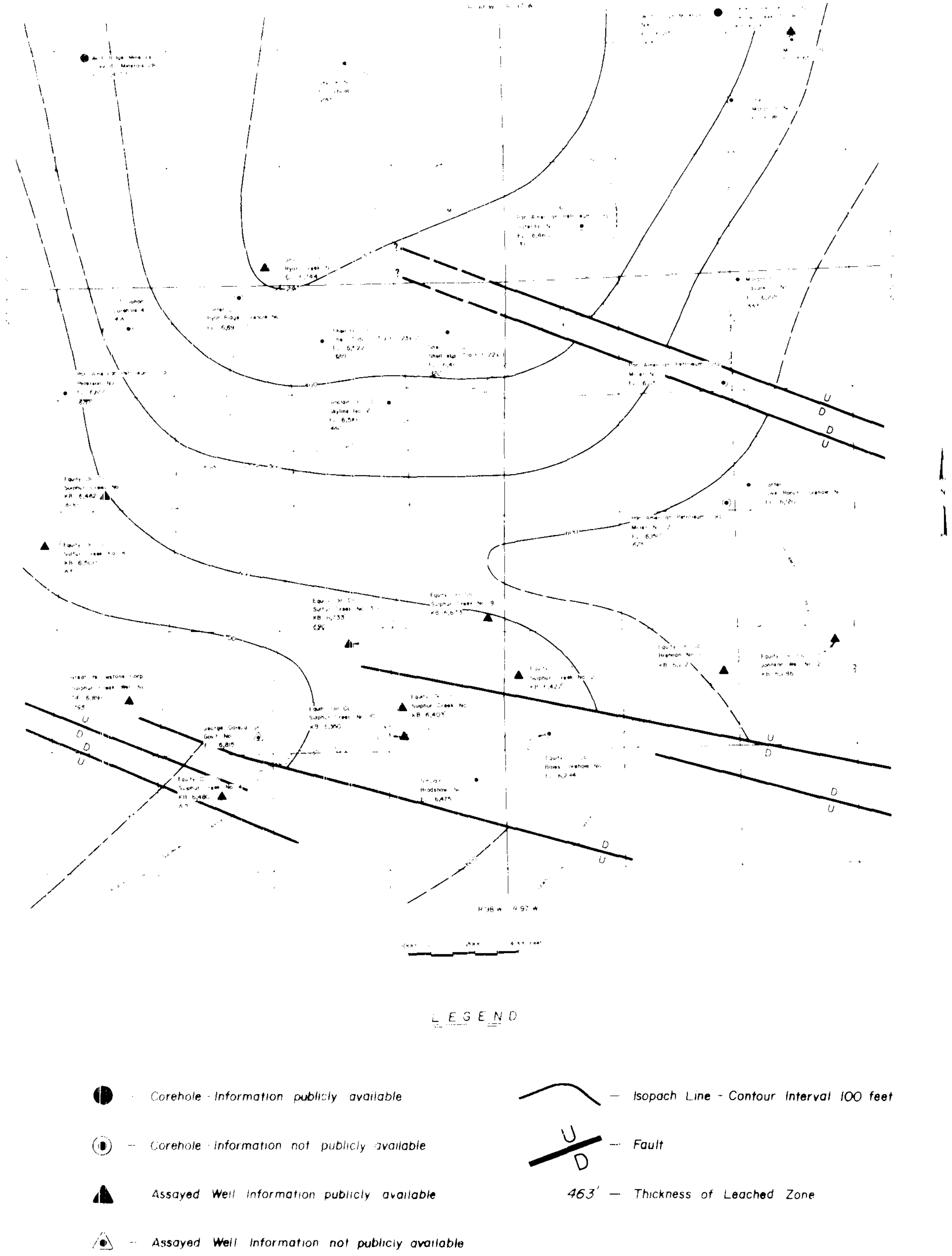
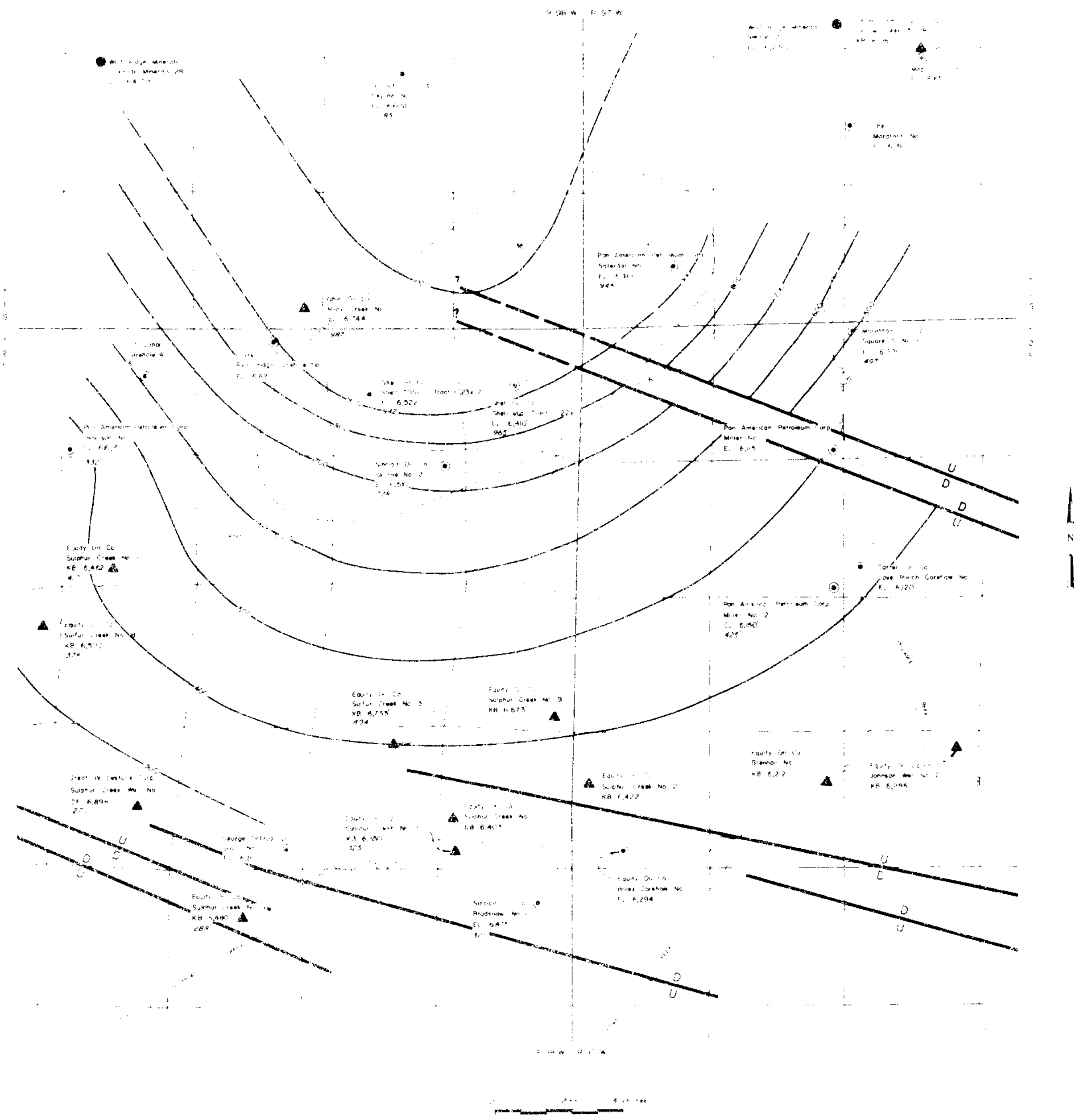


Figure 2.13 Isopach Map of Leached Zone from Top of 'B' Groove to Base of Leached Zone, Site 2



LEGEND

- Corehole - Information publicly available
- Corehole - Information not publicly available
- Assayed Well - Information publicly available
- Assayed Well - Information not publicly available
- Isopach Line - Contour Interval 100 feet
- U D — Fault
- 423' — Thickness of Unleached Zone

Figure 2.14 Isopach Map of Unleached Zone from Base of Leached Zone to Top of Blue Marker, Site 2

Figure 2.15 is a contour map of the base of the leached zone. The anomalous domal feature may be a result of an extension of the Piceance Creek Dome graben. The fault plane has been reported to contain travertine deposits which may effectively reduce or block water flow in the leached zone, resulting in the accumulation of supersaturated solutions. This would account for a relatively low rate of leaching compared with the surrounding area and produce a domal feature as illustrated in Figure 2.15.

The general resource figure for this area was calculated using data from the Shell 23x drill hole. Approximately 2.4 million barrels of oil per acre are contained in a 1452-foot oil shale interval. Other resources figures modified from Beard et al (1974) indicate about 625,000 tons of nahcolite per acre in a 950-foot interval. The total resource figure for site 2 (not the expanded site 2) is 7.7 billion barrels of shale oil, 2 billion tons of nahcolite and 550 million tons of dawsonite. Insufficient data is available to sufficiently estimate the total resources of expanded site 2.

2.6 HYDROLOGY OF PICEANCE CREEK BASIN

The preliminary design of an underground mining system in the Piceance Creek Basin is in part affected by the hydrologic conditions. The amount of water inflow into the mine could affect total mining costs enough to make mining too costly. In order to provide a reasonable estimate of water conditions and pumping costs, the geohydrology of the central Piceance Creek Basin is investigated. The following discussion covers a review of the available groundwater literature in the Piceance Basin area, a description of the general hydrogeologic system and a hydrologic evaluation of proposed mine site 2.

2.6.1 Review of Literature

A summary of hydrologic information of the Piceance Creek structural basin between the White and Colorado Rivers (Coffin, Welder, and Glanzman, 1971) is presented in the form of two large maps printed by the USGS. The maps show the Piceance Creek Basin area and include a potentiometric map and a structure contour map on the base of the Mahogany Zone. Also

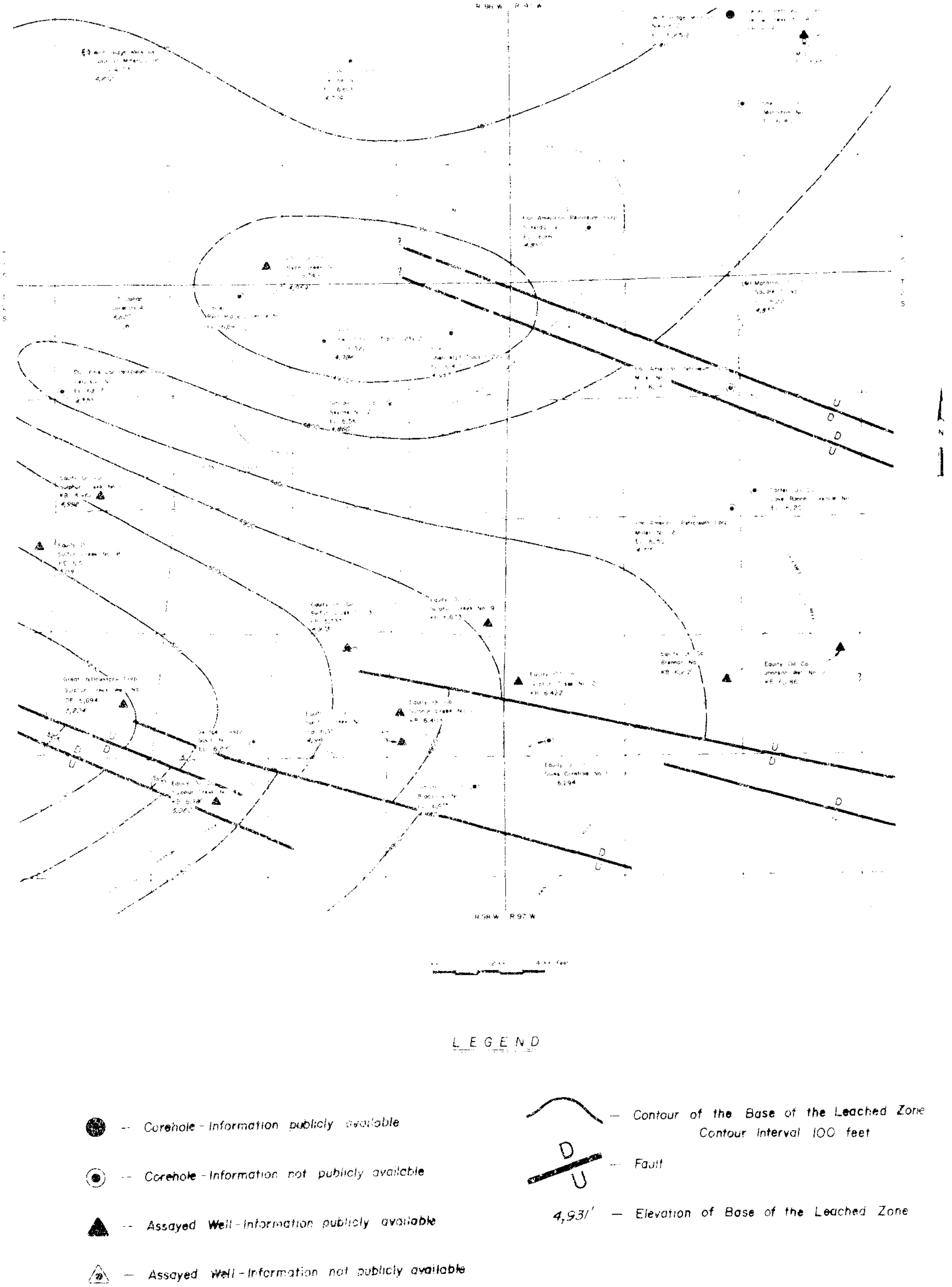


Figure 2.15 Structure Contour on Base of Leached Zone (Approximate), Site 2

included are a stratigraphic section, a diagrammatic section across the Basin, geohydraulic cross sections, chemical character graphs of selected streams, and water quality bar graphs.

In the Rocky Mountain Association of Geologists (RMAG) 1974 guidebook. Weeks (1974) presents a review of the water resources in the Piceance Creek Basin. Annual runoff from the Piceance Creek watershed is estimated at 13,980 acre-feet with its principal use being irrigation. Water quality was found to be poor during periods of low runoff, probably reflecting the poor quality of the groundwater. The groundwater resource is estimated as large with some wells producing as much as 1,000 gpm.

USGS Professional Paper 908 (Weeks, Leavelsey, Welder, and Saulnier, 1974) is a comprehensive evaluation of the hydrologic effects of oil shale mining on Colorado leased tracts C-a and C-b. In the paper, a mathematical simulation of an open area four miles square on each tract was made to estimate mine discharge. Results indicate that on tract C-a the total discharge ranges from 4,000 gpm after one year to 3,000 gpm after 30 years. The majority of the discharge is supplied by a lower aquifer. On tract C-b the discharge varies from 13,000 gpm after one year to 9,000 gpm after 30 years with about two-thirds supplied by an upper aquifer.

A report written by the USGS (1974) for the USBM evaluates the hydrogeology of four proposed oil shale mine sites. The evaluation of the sites was done using the digital model presented by Weeks et al (1974) and resulted in the selection of site 2 as previously described in the resource evaluations.

Wymore (1974) computed a water balance for Yellow and Piceance Creek drainages. Precipitation, evapotranspiration, and runoff plus deep percolation amounts were computed for various elevation-vegetation type combinations.

Striffler (1972) presents some excellent guidelines for surface disposal of spent shale. Characteristics of mountain hydrology and the erosion potential of spent shale disposal areas were discussed in considerable detail. Revegetation techniques were also reviewed.

Both Colorado oil shale leases are administered by the USGS and as a result all data are made public. Currently a hydrologic drilling program is underway to provide more comprehensive data on groundwater conditions in localized areas. It will be possible to estimate more accurately the anticipated mine discharge as these data become public.

2.6.2 Physical Setting

2.6.2.1 Physiography

The Piceance Creek watershed drains a 629 square mile area within the larger White River drainage, a major tributary of the Colorado River. Elevations within the Piceance Creek drainage range from near 5,000 feet to over 9,000 feet, with 86% of the watershed lying between 6,000 and 8,000 feet (Wymore, 1974). The landform is well developed ridge and valley topography with local relief from 200 to 600 feet (Weeks et al, 1974). Drainage development is well advanced and appears to be structurally controlled with drainage patterns ranging from trellis to parallel. The major physiographic feature of the region is a rolling, dissected plateau which forms the south and west portions of the drainage divide and averages over 8,000 feet in elevation. This plateau is a very important feature from a hydrologic standpoint since it receives over 20 inches of precipitation annually and contains the Basin's major groundwater recharge areas.

2.6.2.2 Climate

The Piceance Creek Basin lies in a region generally classified as semi-arid. However, the higher elevation areas, with their attendant higher precipitation and lower evaporative demands, could be classified as sub-humid. This region also lies in the path of several major storm tracks. Storm systems originating in the Gulf of Mexico, in the Pacific coastal region, and in the Gulf of Alaska all have the potential for delivering moisture to western Colorado. Unfortunately, several major mountain systems lie between the Piceance Creek Basin and the storm source areas, with these barriers acting to reduce the amount of moisture available for precipitation. In addition, the relatively low elevation of the Piceance Creek watershed results in only minor orographic lifting as compared to that which occurs on the western slope of the Continental Divide. Annual precipitation averages 17.4 inches and ranges from 12 inches in the valley bottoms to over 25 inches on top of the plateau. Selected sites on the plateau may receive in excess of 30 inches of effective precipitation annually because of snow redistribution by wind. Precipitation is very evenly distributed over the year with month-to-month variations generally less than one inch for the long-term norm. Rainfall intensities are generally light, although intensities exceeding two inches per hour for durations less than 30 minutes occasionally occur at the lower elevations.

Evaporative demand is high at all elevations within the Piceance Creek Basin and probably exceeds precipitation throughout most of the year. Average evapotranspiration is estimated at 17 inches.

2.6.3 Hydrology

2.6.3.1 Surface-Water Hydrology

Streamflow in the Piceance Creek Basin is derived principally from spring snowmelt. Baseflow, however, is derived almost exclusively from groundwater discharge. The mean annual flood event for most drainages within the basin is also derived from snowmelt with the extreme flood events caused by summer convective rainstorms. Although very little peak flow data exist for this region, estimates have been made using computational procedures based upon design storms and watershed characteristics for small drainages (Weeks et al, 1974). Based upon these estimates, it is likely that small drainages of less than five square miles would experience 100-year peak flows exceeding 30 ft^3 per second per square mile (cfsm). Peak flows having recurrence intervals of 100 years or less probably do not exceed 10 cfsm for drainages greater than five square miles.

Snow is perhaps the most important aspect of the hydrologic regime of the Piceance Creek Basin. Nearly all the streamflow and groundwater recharge originates as snowmelt. Snowfall contributes slightly less than half the average annual precipitation at the lower elevations and perhaps as much as 70% at the higher elevations. On top of the plateau, snow is subjected to much redistribution by wind with large accumulations forming behind topographic and vegetative windbreaks. It is highly likely that significant groundwater recharge occurs in these areas. Also, much snow is lost to snowpack evaporation and interbasin transfer of blowing snow.

2.6.3.2 Geohydrology

The Green River Formation was deposited during the Eocene Epoch in a large lake that covered much of northwestern Colorado and North-eastern Utah, forming the Piceance and Uinta Basins of today. Coarser material was deposited near the shoreline with the finer material and organic ooze deposited farther from shore. Deposition included sandy-siltstone and a limey, organic mud that later solidified, forming a marlstone. The organic material gradually converted to the form of kerogen. In some parts of the Basin concentrated salt deposits of sodium chloride and nahcolite were formed; in addition, nahcolite and dawsonite are generally disseminated throughout the Basin in varying quantities.

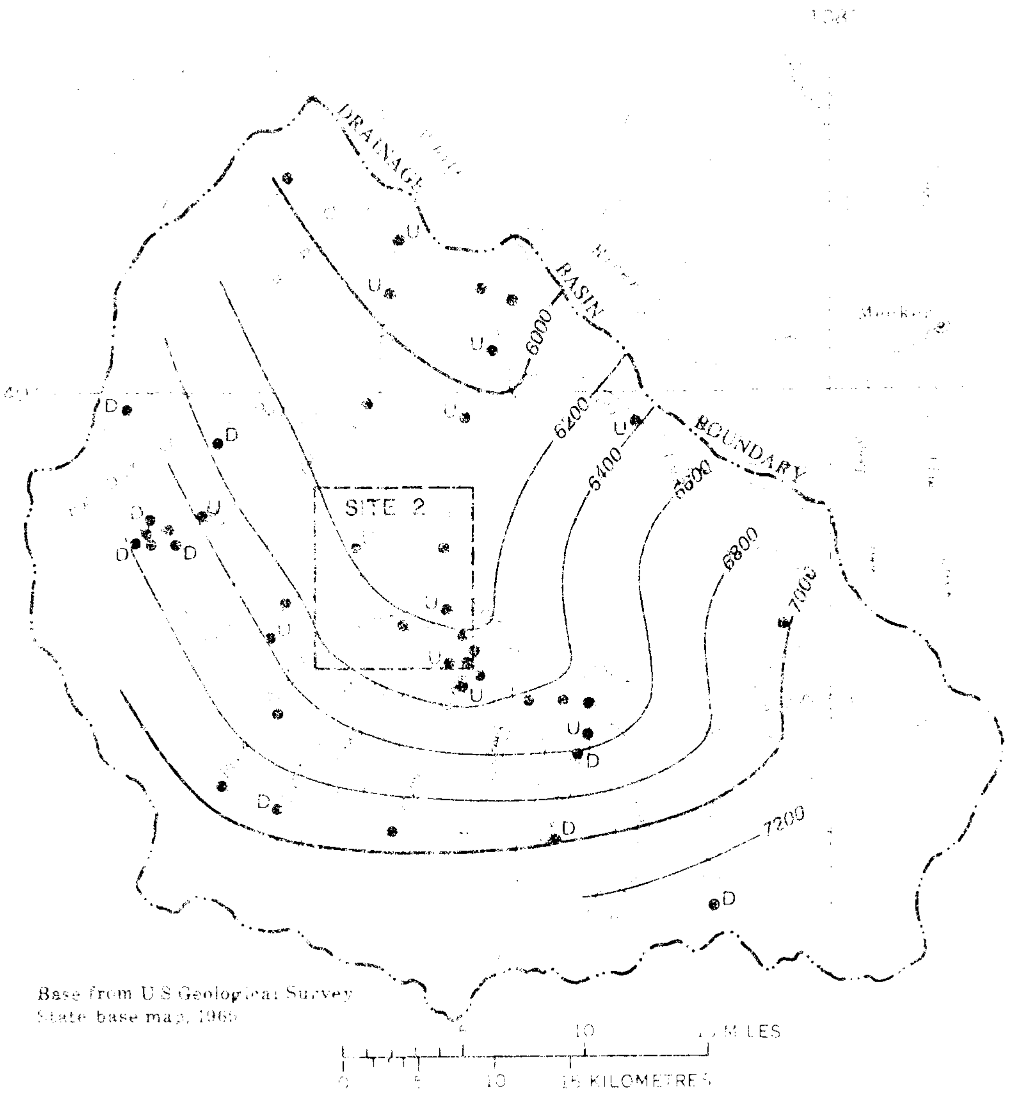
There is no primary permeability in oil shale or marlstone. Marlstone is dense and is formed from calcium carbonate and clay. Organic material was deposited with the mud during periods of great organic abundance followed by periods of organic scarcity. This resulted in what are presently termed rich and lean zones. Since its deposition, the Green River Formation has been subjected to uplifting and tensional forces resulting in the formation of aquifers of structural complexity and secondary permeability. The organic material in the oil rich zones tend to make the oil shale less stiff and able to move plastically during gentle tectonic warping. The lean zones, which are more brittle, fractured during the tectonic warping. Some of the tectonic pressures were so great that the formation was literally torn in two, resulting in faults, grabens, and associated fracturing.

Fracturing, therefore, is a major cause of permeability in the aquifers as they are today. This also holds true for the overlying Uinta Formation that serves as a mantle over most of the Piceance Creek Basin. The Uinta Formation consists of silty, dirty sandstone, clays, and marlstones that are also barren of primary permeability. As a result of the tectonic activity and surface deposition of alluvium the present aquifer system is composed of three systems: (1) alluvial aquifer, (2) upper aquifer, and (3) lower aquifer.

The alluvial aquifers, ranging in thickness from zero to 140 feet, are confined to stream bottoms and restricted in width to generally less than 1/2 mile. The permeability of these aquifers depends upon the source material and increases downstream, being greatest in the main streams approaching the White River. The upper aquifer consists of saturated material from the bottom of the alluvium to the top of the Mahogany Zone. Weeks et al (1974), using data from 26 wells, reports that the transmissivity of the upper aquifer varies from eight to 1000 ft^2 per day.

The Mahogany Zone is generally considered to be an impermeable zone separating the upper and lower aquifers. However, vertical permeability between the two aquifers exists due to fracturing and faulting. Extensive leaching has occurred in the lower aquifer, with its limits extending from the base of the Mahogany Zone to the base of the leached zone. The base of the leached zone is also referred to as the dissolution surface. Below this surface, referred to as the unleached zone, very little permeability exists except that due to minor faulting and fracturing. The transmissivity of the lower aquifer, using data from 20 wells, is approximately 1,940 ft^2 per day.

The potentiometric map, Figure 2.16 (Weeks et al, 1974) indicates that the general water movement is from recharge around the margins of the basin to discharge in the Piceance Creek valley and finally out into the White River. The contours shown are the combined heads of the upper and lower aquifers. There are insufficient data to contour the potentiometric surfaces of each aquifer.



EXPLANATION

● OBSERVATION WELL - D indicates downward and U indicates upward flow in well

---6200--- POTENTIOMETRIC CONTOUR - Shows altitude of water levels. Contour interval 200 feet (61 metres). Datum is mean sea level

Figure 2.16 Potentiometric Map of Piceance Creek Basin, Colorado -- from Weeks et al (1974)

2.6.3.3 Water Quality

Surface water quality, in general, varies with volume and source of flow. Spring runoff provides the best quality of water in terms of dissolved solids, but in this same period the sediment load is the greatest. In late summer and fall, the streamflow is primarily due to groundwater discharge, which causes the water to be high in dissolved solids, but with very little sediment load. Concentrations of dissolved solids in the surface water vary from that contained in rain water and snowmelt in the upper elevations of the basin, to over 5000 milligrams per liter (mg/l) in Piceance Creek at the White River. For the 1973 water year, the greatest sediment load occurred during the peak snowmelt period in May. During this period the sediment load varied from one to four grams per liter.

The concentration of dissolved solids in the aquifers varies with its nearness to recharge, discharge, and the dissolution surface. Table 2.3 gives examples of the minimum, mean, and maximum concentrations of dissolved solids in a number of samples taken from each aquifer. The alluvial and upper aquifers are primarily sodium-bicarbonate waters, with the lower aquifer containing sodium-bicarbonate-chloride water.

2.6.4 Hydrologic Evaluation of Site 2

2.6.4.1 Surface-Water Hydrology

The location of proposed mine site 2, approximately two miles above the mouth of Ryan Gulch, is in the general area of groundwater discharge. About 41 square miles are drained by Ryan Gulch above the site. All of the streams near the site flow only intermittently except for Piceance Creek, which flows perennially. Climate of the area is semi-arid, with precipitation varying from 12 to 15.5 inches. From November through March, precipitation generally occurs as snow and during the remainder of the year as rain. Potential evapotranspiration for a horizontal surface in the area is approximately 43 to 46 inches annually, exceeding precipitation nearly every month of the year. On the average, most of the annual precipitation goes toward satisfying soil water storage deficiencies and eventually is lost through evapotranspiration. Occasionally,

Table 2.3 Dissolved Solids in Water Samples from the Alluvial, Upper, and Lower Aquifers*-- from Weeks et al, 1974

Aquifer	Minimum	Mean	Maximum
Alluvial[1]	469	1,750	6,720
Upper[2]	345	960	2,180
Lower[3]	491	9,400	38,900

[1] Based on 27 samples.
[2] Based on 17 samples.
[3] Based on 27 samples.

* in mg/l

under summer convective storm conditions, the rainfall intensity will exceed the soil water infiltration rate and surface runoff will occur. For some years where winter snowfall is well above normal, some groundwater recharge may occur. At the general elevation of the site, 6,000 to 6,800 feet, snowcover is not persistent and mid-winter melt is a common occurrence. Such conditions reduce the probability of significant runoff from snowmelt.

The USGS (1974) computed the 100-year, one hour runoff event as 0.7 inches total runoff for the drainages within and adjacent to all of the proposed mine sites. The USGS also reported that the Soil Conservation Service runoff estimate for a probable maximum six-hour storm was 6.5 inches for the same general area.

2.6.4.2 Groundwater Hydrology

The USGS (1974) reports that the upper aquifer at site 2 is 700 feet thick and has a transmissivity of about 140 ft^2 per day. The lower aquifer is 550 feet thick and has a transmissivity near 400 ft^2 per day. The storage coefficients are estimated to be 10^{-3} and 10^{-4} for the upper and lower aquifers, respectively. Under dewatering conditions the storage coefficient for the upper aquifer was assumed to be 10^{-1} (USGS, 1974).

2.6.4.3 Water Quality

Based upon water quality data from various locations within the Piceance Creek Basin, it can be stated in general terms that surface water quality is expected to be high, with low dissolved solids concentrations. During periods of storm runoff, however, suspended sediment concentrations will be high, particularly where contributing areas have experienced surface disturbance. Groundwater discharge on proposed mine site 2 is likely to contain relatively low quality water in terms of dissolved solids concentration.

All wastewater and storm runoff produced on site 2 would have to be contained in well sealed detention ponds to prevent degradation of surface waters below the site. Spent shale disposal sites should be selected and designed to minimize surface and subsurface runoff. The concentration of dissolved solids in the water produced from an oil shale mine on site 2 is estimated by the USGS (1974) to be 1,000 mg/l for the upper aquifer and 5,000 to 10,000 mg/l from the lower aquifer.

2.6.4.4 Estimated Water Inflow

The USGS used a digital model of the groundwater system of the Piceance Creek Basin (Weeks et al, 1974) to estimate the dewatering requirements of the federal lease tracts C-a and C-b. The model assumed the groundwater flow between the upper and lower aquifers was unimpeded and that the upper aquifer was drained. From these assumptions, the USGS simulation model computed the discharge from C-a and C-b to be approximately 9 ft^3 per second (4,000 gpm) and 20 ft^3 per second (9,000 gpm), respectively, after a period of one year. Due to the large amount of assumptions made in predicting these values, a figure of 10,000 gpm is assumed for the representative mine site (site 2).

SECTION 3

ENGINEERING PROPERTIES OF GREEN RIVER OIL SHALE

Green River Formation oil shale, an organic-bearing marlstone, is a relatively complex rock with highly variable strength characteristics. The two factors that affect the elastic and dynamic properties most significantly are Fischer assay grade and bedding. In common with all bedded rocks, the physical properties of oil shale measured in the plane of bedding differ from those measured perpendicular to bedding. The relationship of engineering properties to Fischer assay is peculiar to the three oil rich members of the Green River Formation: the Parachute Creek, the Garden Gulch, and the Douglas Creek.

The design of underground openings in oil shale must consider the ability of the rock mass to resist the stress concentrations resulting from the interaction of the original stress state and excavation geometry. Parameters essential to rock mechanics engineering, such as deformation modulus, Poisson's ratio, compressive strength, tensile strength, fracture frequency, internal frictional resistance, and cohesion, must be obtained prior to any realistic engineering design program. Any mine design in oil shale must consider the variations in kerogen content and, consequently, the engineering properties associated with the oil shale deposit.

In the discussion that follows, a review of pertinent published data on engineering physical properties is presented followed by a review of design methods and parameters needed for the design of underground excavations in oil shale. A synthesis of all physical property data on oil shale is presented and is followed by the conclusions.

3.1 REVIEW OF LITERATURE

Published research and physical testing data on engineering properties for oil shale are quite limited. The USBM started an experimental oil shale mining project in 1945 on a naval oil shale reserve 10 miles west of Rifle, Colorado. The majority of physical testing of oil shale has been done on samples from this mine (Anvil Points) which is now operated by Development Engineering, Incorporated, a subsidiary of the Paraho Development Corporation.

Merrill (1954) in RI 5089 reported the results of an extensive testing program performed for the Anvil Points oil shale experimental mine. Unconfined compressive strength, Young's modulus, Poisson's ratio, modulus of rigidity, and sonic velocities were determined at three separate laboratories. In general, it was found oil shale could be classed as a moderately strong rock depending on oil content. As oil content increased, the rock became more plastic and generally weaker.

Tesch (1961), at the Denver Mining Research Center, performed physical property determinations for Young's modulus, unconfined compressive strength, Poisson's ratio, and modulus of rupture on six bulk oil shale samples from Anvil Points. The samples tested ranged in Fischer assay from five to 60 gpt with testing performed both parallel with and perpendicular to the bedding. As expected, the results of the testing program showed the effect of kerogen content on sample strength and an increase in compressive strength when the principal stress was parallel to bedding.

Raju (1961) examined the effects of layering on the elastic, static, and dynamic behavior of oil shale. Tests performed on specimens with a cross section of one by one inch by two inches long showed higher compressive and tensile strengths with loads applied along the bedding. Raju also described a rough relationship between projectile impact energy, depth of penetration, and excavated volume for three degrees of oil shale richness.

At a test mine operated by Mobil Oil Company, adjacent to the Anvil Points operation, Sellers, Haworth, and Zambas (1972) performed physical testing for uniaxial compressive strengths, elastic moduli, and specific gravity with respect to Fischer assay. In addition, measurements of the in situ stress field indicated that the maximum principal stress was in the horizontal plane. Finite element analysis, photo-elastic stress meter monitoring, and displacement and convergence monitoring were performed to examine the effects of excavation in oil shale.

A doctoral thesis by Agapito (1972) reports the results of physical testing and displacement monitoring at the Colony Development Corporation mine at Parachute Creek near Rifle, Colorado to determine parameters for pillar design and failure modes. Essentially all data obtained were used in a finite element program to numerically predict failure mechanisms in layered media and provide more accurate design criteria.

Additional data have been derived from published and unpublished reports by Chamberlain (1970), Hjelmstad (1974), Podio, Gregory, and Gray (1968), and the Colorado School of Mines.

3.2 REVIEW OF DESIGN METHODS FOR UNDERGROUND OPENINGS

Numerous methods and concepts are presently available for designing stable underground openings in rock. All of these methods generally require some common basic engineering properties as input. Pillar designs, for example, require knowledge of the maximum theoretical load carrying capacity of the rock. In roof beam design, the tensile and compressive strength is needed; and in subsidence the elastic moduli must generally be known.

Most design methods have been developed for application in specific localities or types of rock. The following review of analytic methods in pillar design, roof beam analysis, and subsidence estimation is concerned only with those methods applicable to bedded formations. The purpose of this review is to isolate and identify those parameters most important to the design of safe underground openings in oil shale.

The current methods for designing openings in underground excavations are numerous and generally apply only to special local conditions. In the following sections, some of the basic methods are analyzed and the important engineering parameters noted.

3.2.1 Pillar Design Parameters

The simplest approach to pillar design is presented in USBM Bulletin 587 (Obert, Duvall, and Merrill, 1960). This method involves setting up a physical testing program to determine the unconfined compressive strength of competent specimens. The maximum design pillar stress is then determined by dividing the specimen strength by a factor between two to four. However, the shape of the specimens tested must be identical to the shape of the design pillar or a modification of the specimen compressive strength must be made (equation 3.1).

$$\frac{C_1}{C_2} = \frac{0.778 + 0.222(\frac{D_1}{H_1})}{0.778 - 0.222(\frac{D_2}{H_2})} \quad \ldots\ldots\ldots (3.1)$$

The limitations for the above equation are diameter-to-height ratios from 1:4 to 4:1. Many of the planned oil shale pillars exceed this limitation. In addition, the planned pillars may not be cylindrical; however, a conservative assumption is that a square pillar is equal to a circular pillar if the diameter equals the side length of the square pillar. It can also be conservatively assumed that a rectangular strip pillar has an average strength equal to that of a circular pillar with a diameter equal to the narrow dimension of the strip pillar.

An assumption fundamental to this method is that the strength of the rock mass be roughly identical to the strength of the rock specimen measured in the laboratory. Every size effect study performed on rock has shown this assumption to be false.

Several empirical equations for pillar strength have been developed for particular coal seams. Salamon and Munro (1967), Holland (1973), and Greenwald, Howarth, and Hartmann (1941) developed some of the more useful ones. These equations take the form:

$$\text{Strength} = C_o\left(\frac{h^{\zeta}}{W^{\xi}}\right) \quad \ldots\ldots\ldots\ldots\ldots (3.2)$$

The derivation of the constants ζ and ξ is possible only after acquiring extensive experience in a particular geologic environment. No attempt has been made (nor should it be) to transfer these equations directly to another district, another seam, or another material without an in situ testing program. Extensive experience in oil shale will be needed before this method can be used. It would also be hazardous to extrapolate the limited experience of room and pillar mining in oil shale near the Anvil Points outcrop to the greater depths of the central Piceance Creek Basin.

The design of pillars based on the assumption that an average stress acts across an entire pillar has been demonstrated to be unrealistic. Measurements of stress on pillars have repeatedly shown that the stress level increases from the edge toward the center of the pillar, Figure 3.1. Wide pillars reach a maximum (peak) stress ($\hat{\sigma}v$) at some distance ($\hat{Y}$) in from the edge of the pillar. The central core of a wide pillar is subject to a stress less than or equal to the peak stress.

Wilson (1972) presents a rationale for determining the magnitude and location of the peak stress in a pillar. The peak stress is defined as that stress which can be carried by the rock mass under the confinement offered by the horizontal stress acting in the roof and floor. This in situ horizontal stress can be measured; however, at this point in the design of pillars in oil shale it must be estimated.

Some reported in situ stress measurements in oil shale indicate a ratio of horizontal to vertical in situ stresses that vary from 0.16 to 1.16. Available in situ stress measurements at Anvil Points are probably not representative of the major portion of oil shale which occurs at greater depth and is remote from any lateral stress-releasing outcrop. The best current estimate is to equate the horizontal confining stress to a value near that of the overburden stress.

The maximum failure stress, σ_p, in a pillar is determined by summing the uniaxial compressive strength of the pillar rock and the effect of confining stress on compressive strength, $\sigma_h \tan\beta$:

$$\sigma_p = C_o + \sigma_h \tan\beta \qquad (3.3)$$

The passive pressure coefficient, $\tan\beta$, is determined by two methods. The first involves assuming a value for the internal frictional

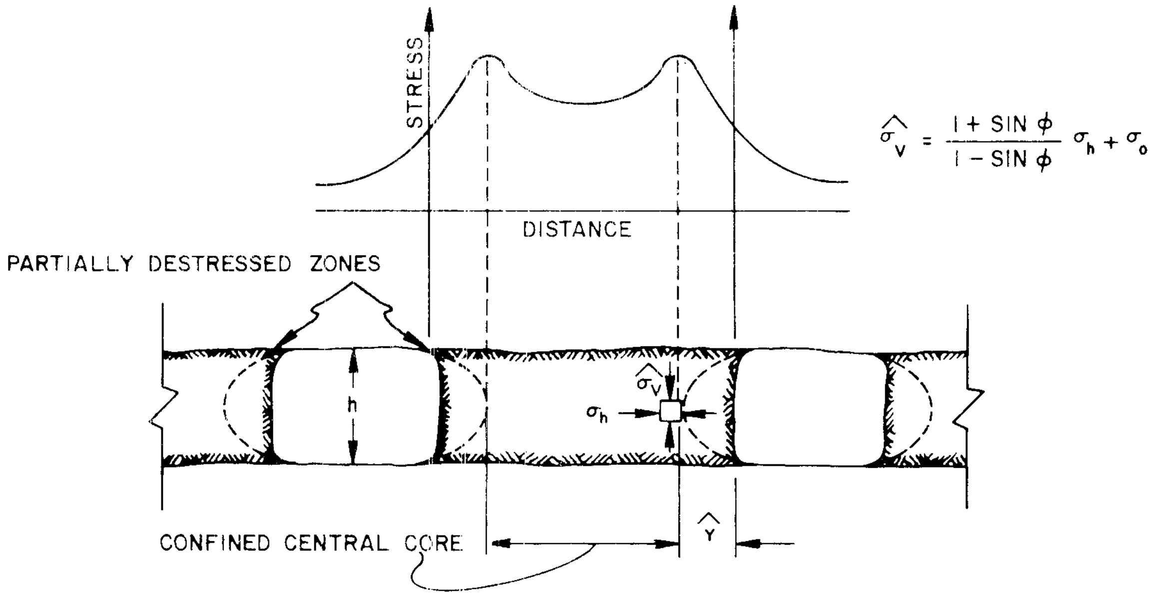


Figure 3.1 Passive Pressure Potential for Confined Central Core of a Pillar

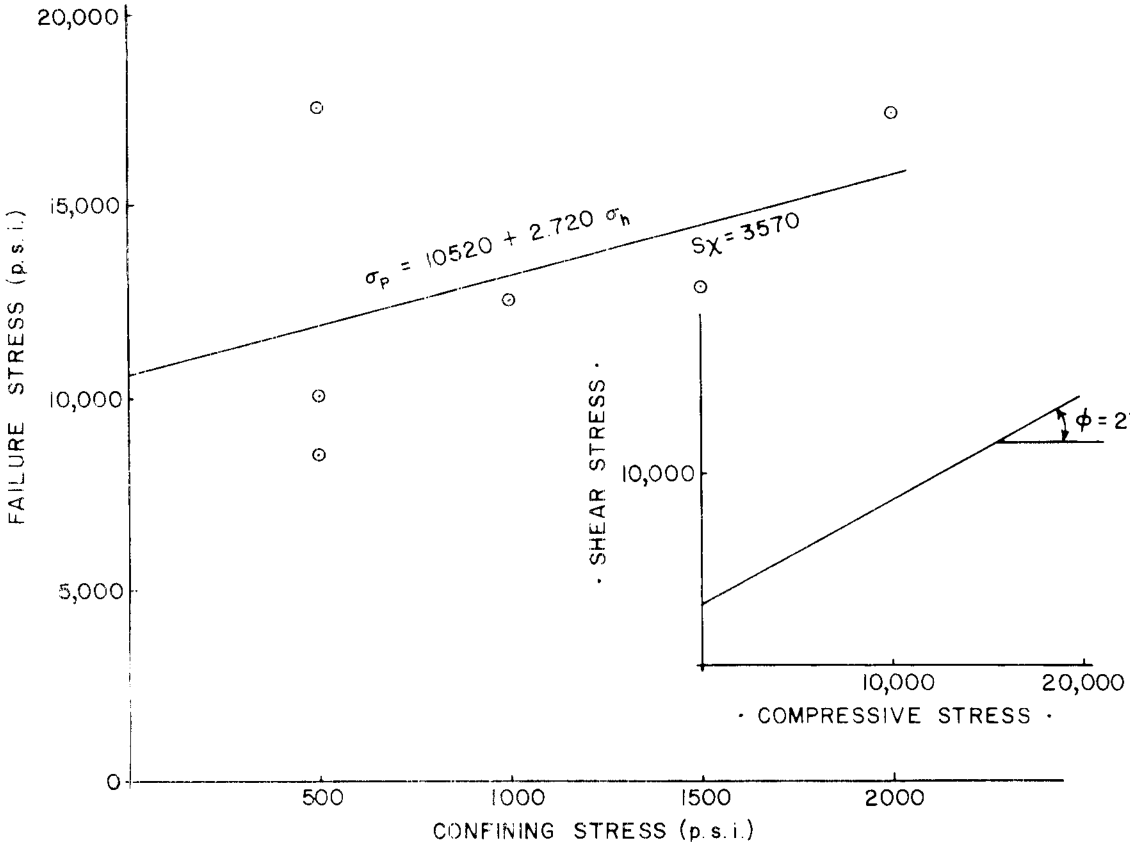


Figure 3.2 Triaxial Testing to Determine Passive Pressure Coefficient and Friction Angle

resistance (ϕ) of the rock and calculating tanβ from the following equation developed by Terzaghi and Peck (1948):

$$\tan\beta = \frac{1 + \sin\phi}{1 - \sin\phi} \quad \ldots\ldots\ldots\ldots (3.4)$$

The second involves performing triaxial tests on the rock, plotting the data with respect to failure stress and confining stress, and finding the equation of the best fit line through the data points. Figure 3.2 presents a set of triaxial test results on oil shale samples from the Green River Formation. The passive pressure coefficient in this case is 2.720. When triaxial test results are available this method provides more reliable values of tanβ.

Wilson has derived an equation for the location of the peak pillar stress:

$$\hat{Y} = \frac{h}{\sqrt{\tan\beta}\,(\tan\beta - 1)} \ln \frac{\hat{\sigma}_v}{\sigma_o} \quad \ldots (3.5)$$

As load is applied to the pillar, the stress in the confined central core increases until the entire central core carries the peak stress. At this time the pillar begins to fail, either from the outside or from the weakest part of the pillar.

Before using equation (3.5) the cohesion of the rock mass must be determined (derived from triaxial testing):

$$\sigma_o = \frac{C_o}{2\sqrt{\tan\beta}} \quad \ldots\ldots\ldots\ldots (3.6)$$

A narrow pillar, that is, a pillar whose height is so much greater than its width that no confined central core develops, has no reserve load carrying capacity. This pillar would be prone to sudden failure as soon as its load carrying capacity is reached. Such a narrow pillar never reaches the peak stress ($\hat{\sigma}_v$) because the confinement offered by the horizontal stresses in roof and floor are never fully effective at the center of the pillar.

Wilson's method can also be used to evaluate the increased strength of the pillar when the lateral confinement is increased. Backfilling around the pillar, cabling, or bolting the pillar increases the lateral confinement.

3.2.2 Roof Beam Design Parameters

The design of safe and efficient widths for rooms in oil shale may require the use of roof bolts or dowels. The high probability of thin beds, with minimal bed-to-bed strength being present in the immediate roof above a room, will require a positive method of defining the thickness of the roof beam. Roof bolting can provide the needed positive assurance of adequate roof beam thickness.

Coates and Cochrane (1970) present a method derived from laboratory tests and field observations for roof bolt design in Canadian coal and metal mines. The required input consists of rock bolt tensile strength, anchorage capacity of the bolt, joint or fracture spacing, and the unit weight of roof rock. This method is used in areas of highly jointed and fractured rock where the roof is not considered to have a significant horizontal stress component or any load carrying capacity. The relatively large average joint spacing in oil shale, approximately seven feet, indicates a spacing of 20 feet is possible with 21-foot long bolts. The weight of such a massive block, approximately 1,180,000 pounds, would require five-inch diameter bolts to carry the design load, which is unrealistic. Bolts of such length are unnecessary because the roof beam need be only thick enough to carry the loosened rock above the room to the pillars. The height or thickness of this zone cannot elastically exceed one-third of the width of the opening below.

The strength of an oil shale roof beam, due to the presence of joints, should not be calculated from the tensile strength of oil shale specimens. The strength of the roof beam is exceeded when the tension developed from bending, at the center of the roof span, exceeds the in situ horizontal compression in the roof. A minimum safe value of tensile strength, σ_t, is selected which in this case is actually minimum allowable compression:

$$\sigma_t = \sigma_h - \sigma_b \quad \ldots\ldots\ldots\ldots\ldots \quad (3.7)$$

The calculated bolting length is then: (maximum beam load is $\frac{\gamma \ell}{3}$)

$$b = \sqrt{\frac{\gamma \ell^3}{12(\sigma_h - \sigma_t)}} \quad \ldots\ldots\ldots\ldots \quad (3.8)$$

It is then necessary to verify that the compressive strength of the rock subjected to an upper fiber compressive stress is not exceeded. The maximum compressive stress, σ_c, is equal to: (σ_c must be less than C_o)

$$\sigma_c = \sigma_h + \sigma_f \quad \ldots\ldots\ldots\ldots\ldots \quad (3.9)$$

The compressive strength of oil shale is the only engineering property that effects the design of the roof member.

3.2.3 Subsidence

Surface subsidence above underground room and pillar mining is a function of the percentage extraction, the elastic constants of the pillars, and the width of the mining area with respect to the depth below surface. When the minimum mining width exceeds the depth by approximately 1.4 times, maximum vertical subsidence occurs in the center of the mining area. The shape of the subsidence curve is independent of the overlying rock. The magnitude of the vertical subsidence is dependent on the average vertical stress added to the pillars, the horizontal stress removed from the pillars, and the elastic constants of the pillar. A compatability equation will yield an estimate of the vertical strain in the pillar. The vertical shortening of the pillar is equal to the pillar strain times the pillar height.

The theoretical determination of stresses in a pillar due to mining are calculated from the following equations:

$$\Delta\sigma_v = \frac{\gamma H}{144}\left(\frac{R}{100-R}\right) \quad \text{.........(3.10)}$$

$$\Delta\sigma_h = \frac{\gamma H}{144} \quad \text{...............(3.11)}$$

The vertical strain, ε_v, is then:

$$\varepsilon_v = \frac{\Delta\sigma_v - \nu(2\Delta\sigma_h)}{E} \quad \text{.........(3.12)}$$

The shortening of the pillar, not considering time effects, due to mining then becomes:

$$\Delta\ell = h\,\varepsilon_v \quad \text{...............(3.13)}$$

Provided that the depth of mining and roof span are adequate to ensure excessive deflection does not occur between pillars, equation (3.13) will give a reasonable estimation of subsidence. The subsidence that will appear at the surface, in the case of pillar failure or full extraction and backfilling, is independent of the rock properties but dependent on the properties of any backfill and the thickness of the mining horizon.

3.2.4 Summary of Design Parameters

From the preceeding discussion, the physical properties necessary to design safe underground openings in oil shale have been described. In summary, the physical parameters needed for an adequate initial design of a deep underground oil shale mine are: (1) unconfined compressive and tensile strength, (2) passive pressure coefficient, (3) deformation or Young's modulus, (4) Poisson's ratio, and (5) cohesion. All of the above properties can be obtained from uniaxial and triaxial testing.

3.3 DATA SUMMARY OF OIL SHALE ENGINEERING PROPERTIES

Early studies on the physical properties of oil shale conducted in the 1930's reported only those chemical properties necessary for processing development. More recent tests focus on those physical properties of oil shale which are needed for engineering design.

The majority of studies reporting engineering properties also include density. With Sellers' work correlating density to Fischer assay, it becomes possible to relate engineering properties to Fischer assay through the density. Most investigators have stated, or implied, a qualitative relationship between a given physical property and Fischer assay; however, there is little or no published quantitative evaluation of these relationships. By correlating all the studies to reported values for density, an attempt is made to show the dependence of engineering properties on Fischer assay.

Figure 3.3 (Sellers et al, 1972) presents the correlation of specific gravity to Fischer assay. On the basis of this relationship, the Fischer assay can be estimated where specific gravity was reported, but the actual assay was not obtained.

3.3.1 Strength Properties

The unconfined compressive strength perpendicular to the bedding of oil shale specimens has been measured by several investigators. The results of an extensive compression testing program were presented graphically by Sellers. Figure 3.4 presents those data, plus additional data related to Fischer assay, illustrating both the natural variability of oil shale compressive strength along with the effect of increasing Fischer assay. Notice that the compressive strength reaches a minimum at about 50 gpt.

The relationship of tensile strength to Fischer assay is presented in Figure 3.5. Notice that the tensile strength is greater parallel to bedding. Three methods of measuring tensile strength were reported: axial load (pure) tension, modulus of rupture, and induced tension. The measured values were converted to approximate axial load values by the following relationships: modulus of rupture divided by 3.5 equals pure tension, and induced tension divided by 1.6 equals pure tension.

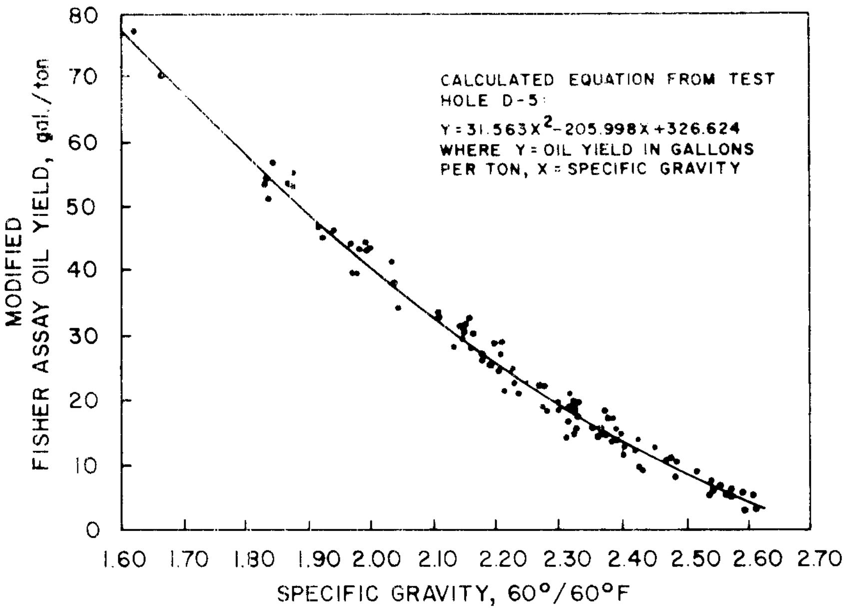


Figure 3.3 Specific Gravity vs. Oil Yield for Core D-5 --from Sellers et al, 1972

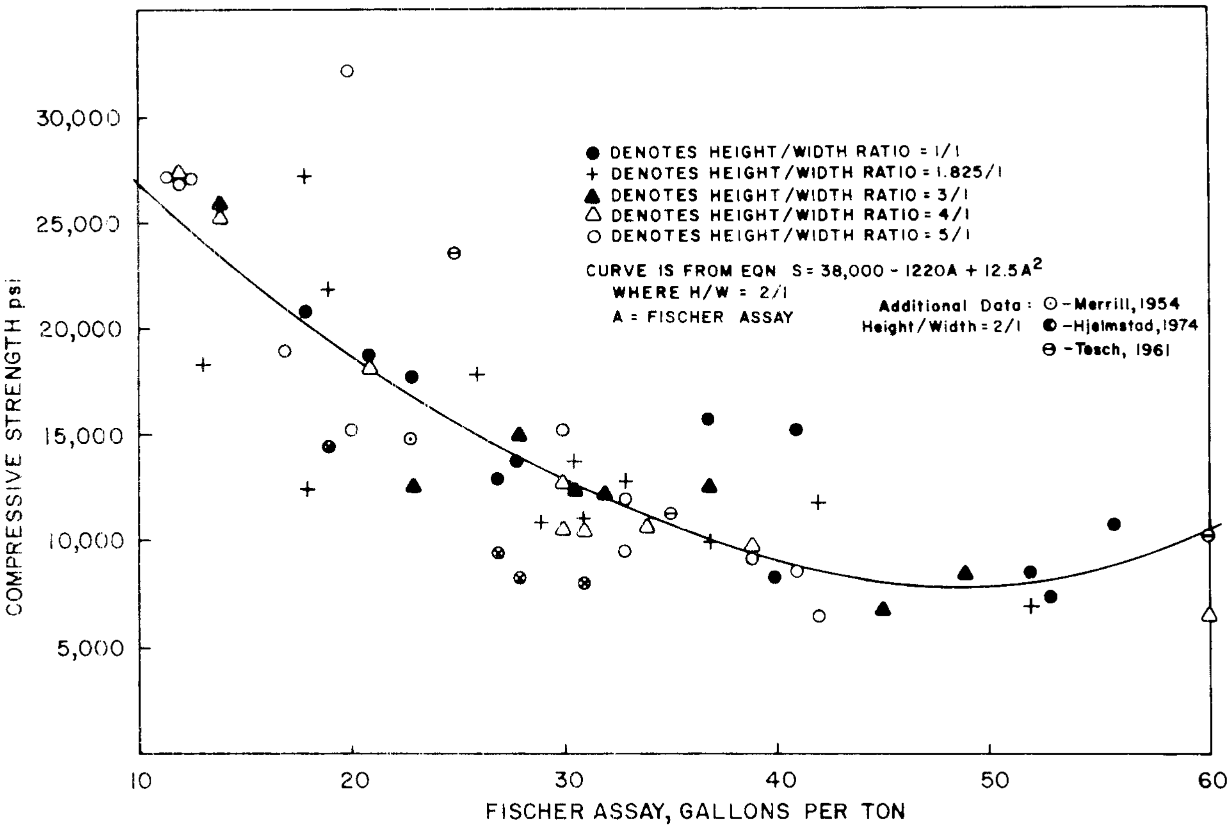


Figure 3.4 Compressive Strength vs. Fischer Assay for Anvil Points Oil Shale -- after Sellers et al, 1972

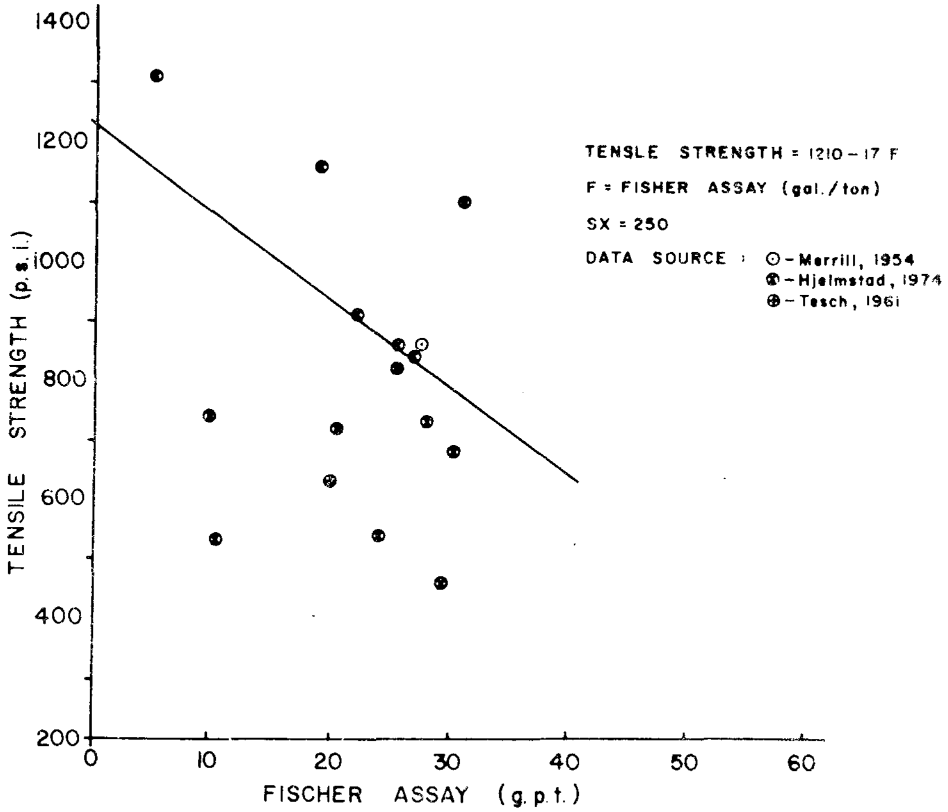


Figure 3.5 Tensile Strength vs. Fischer Assay (Parallel to Bedding)

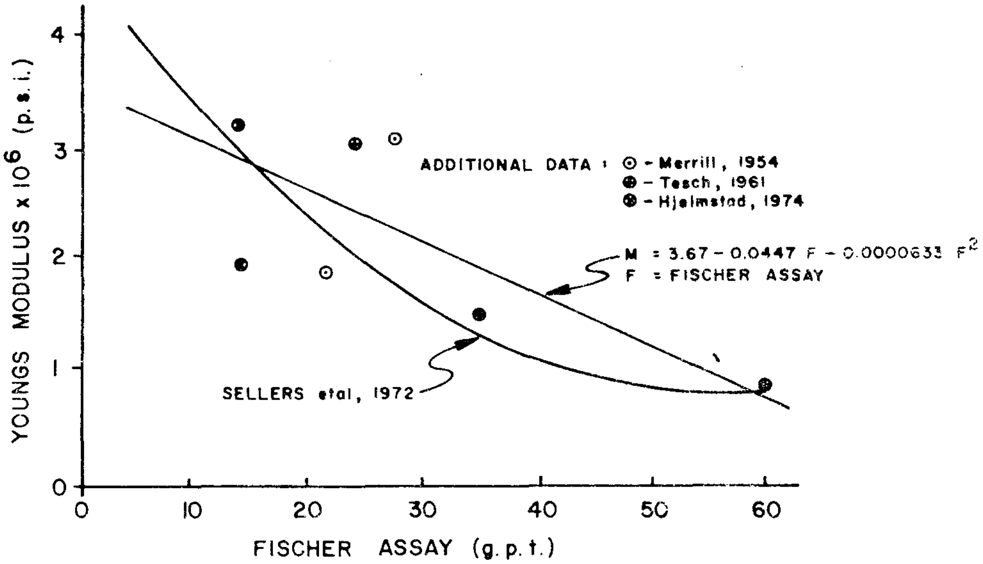


Figure 3.6 Elastic Modulus vs. Fischer Assay (Parallel to Bedding)

There is a close correlation between tensile strength and Fischer assay. This is obvious despite the inherent inaccuracy of estimating both Fischer assay from density and pure tension from other cheaper and faster tension testing methods. The correlation coefficient, r, obtained from the data is 0.796 for the 36 sets of reported values. The calculated "Student t" statistic is 7.660 which exceeds that needed for a 99 percent confidence level for interrelationship.

3.3.2 Elastic Properties

The elastic deformation of pillars depends on the stress applied, the stiffness of the pillar (elastic modulus), and deformability (Poisson's ratio). Both of these properties are related to the Fischer assay of the specimen being tested. Sellers et al, (1972) developed approximate relationships which are presented in Figures 3.6 and 3.7. Data from other investigators have also been added to Sellers' curves. These results also have obvious application to the design of permanent pillars and to the prediction of subsidence at the surface.

The degree of elastic anisotropy perpendicular and parallel to bedding appears to be minor. The elastic properties of a single oil shale layer are approximately the same in all directions.

3.3.3 Stability Parameters

The design of pillars using the load carrying capacity of a confined central core requires the determination of the angle of internal friction (ϕ) and cohesion (c). Figure 3.8 presents the limited data available on the relationship between angle of internal friction of intact oil shale and Fischer assay. Figure 3.9 presents the available data on cohesion of intact oil shale with respect to Fischer assay.

The correlation between the intact ϕ for oil shale and Fischer assay is phenomenally close, r = 0.998. The confidence level exceeds 99 percent for this interrelationship. This high confidence is unusual for so few sets of samples, five in this case.

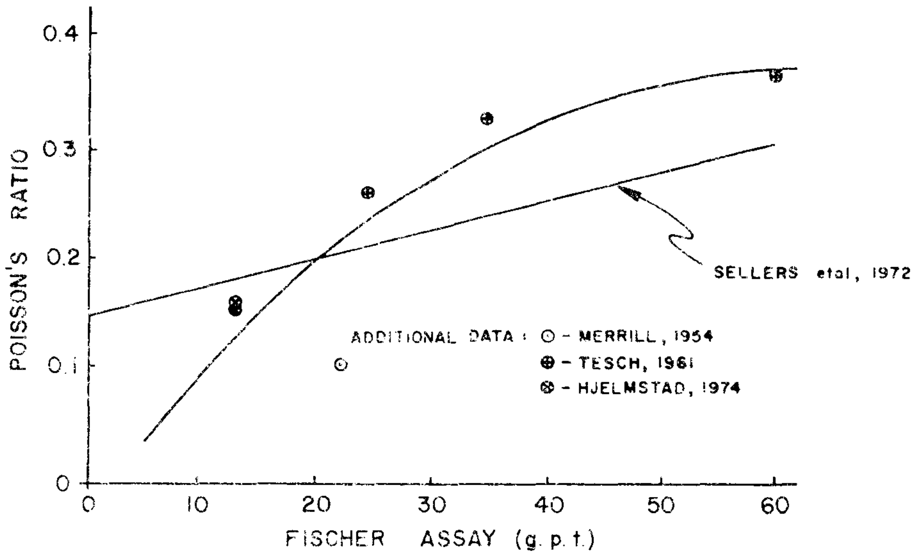


Figure 3.7 Poisson's Ratio vs. Fischer Assay -- after Sellers et al, 1972

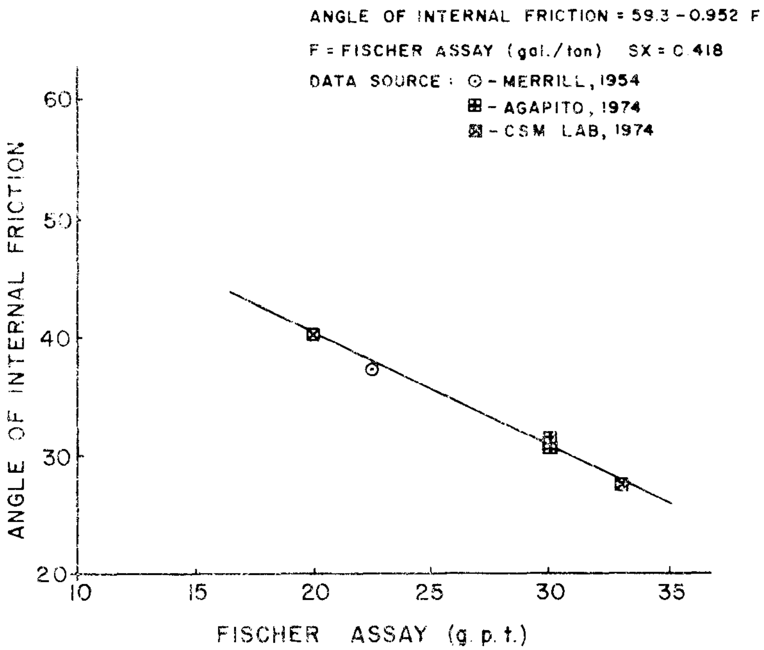


Figure 3.8 Angle of Internal Friction vs. Fischer Assay (Intact Rock, Perpendicular to Bedding)

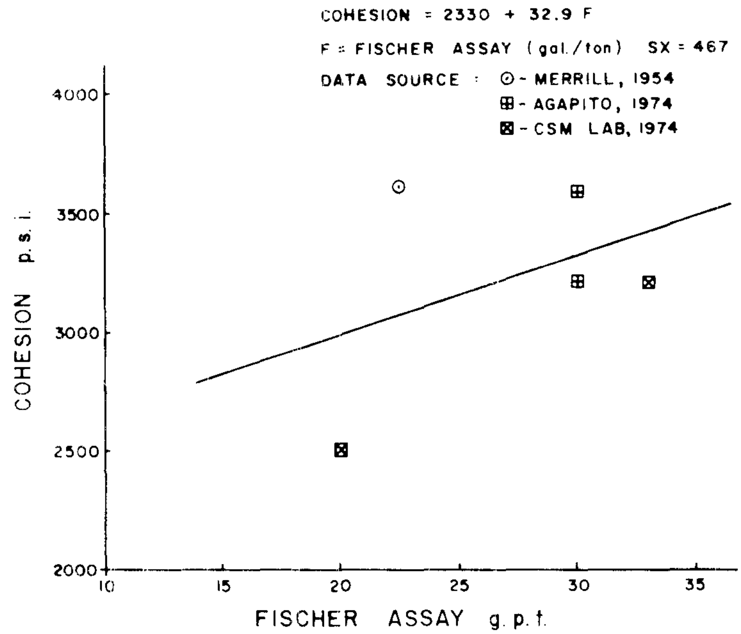


Figure 3.9 Cohesion vs. Fischer Assay (Intact Rock, Perpendicular to Bedding)

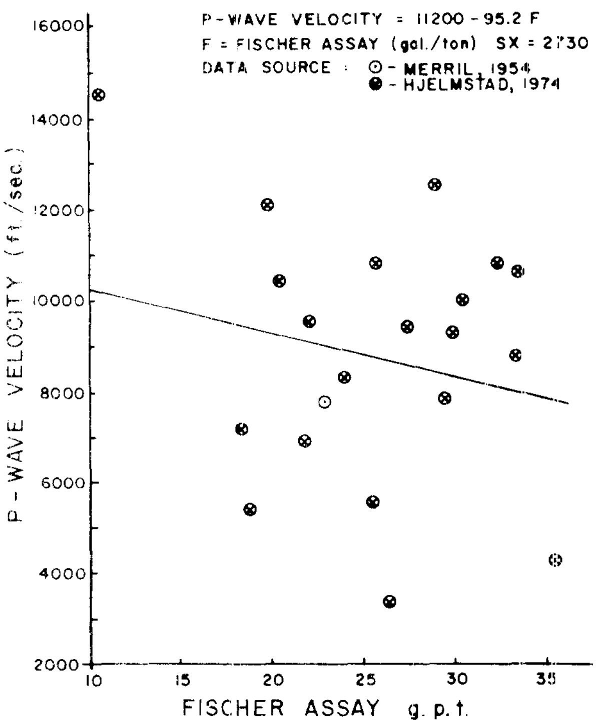


Figure 3.10 P-Wave Velocity vs. Fischer Assay (Perpendicular to Bedding)

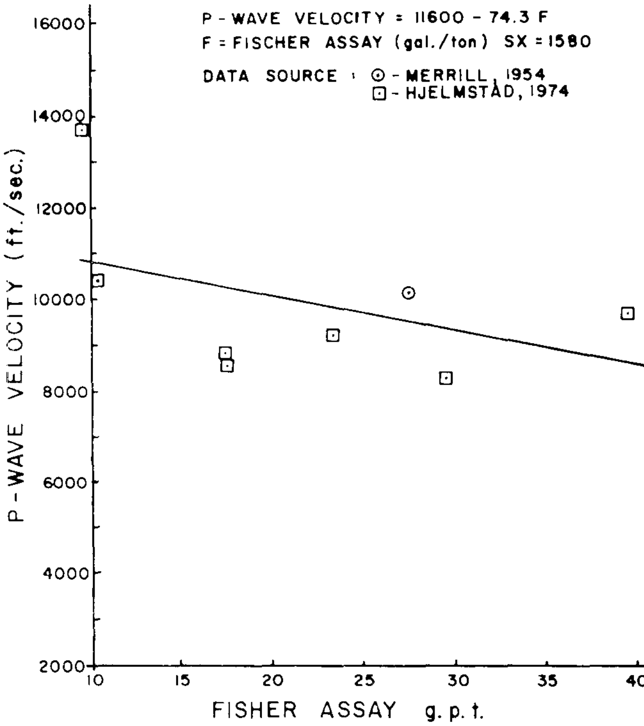


Figure 3.11 P-Wave Velocity vs. Fischer Assay (Parallel to Bedding)

The correlation between cohesion and Fischer assay is low, r = + 0.412. The confidence level is only 65 percent that a relationship actually exists between these two properties of oil shale. This probably results from cohesion being calculated from the angle of internal friction as well as from triaxial tests. There is no direct method for the measurement of cohesion.

The angle of surface friction and surface cohesion variation with respect to Fischer assay cannot be estimated from available data. These properties are needed to evaluate the active pressure of raw broken shale or spent shale on the walls of an active stope. Spent shale would be a natural backfill for exhausted stopes. The active pressure of spent shale would load and restrain any permanent pillars. Active pressure against the pillars will increase their load carrying capacity. The only measurements of the angle of surface friction and surface cohesion are from Agapito (1974) for 30 gpt oil shale.

3.3.4 Compressional (P) Wave Velocity

The P-wave velocity of oil shale has been investigated by several researchers. Their results demonstrate a wide scatter in P-wave velocities, which nearly masks the effect of Fischer assay. Figure 3-10 shows the P-wave velocity perpendicular to bedding with respect to Fischer assay; and Figure 3.11 shows P-wave velocity parallel to bedding. These figures demonstrate the greater consistency of P-wave velocity as measured parallel to bedding against those measured perpendicular to bedding.

The correlation coefficient for the relationship between the P-wave velocities measured perpendicular to bedding and Fischer assay is -0.210. The correlation coefficient between P-wave velocities measured parallel to bedding and the Fischer assay is -0.414. The 22 measurements perpendicular to bedding yield only an 85 percent confidence that a Fischer assay relationship exists; where as, the 11 measurements parallel to bedding yield an 88 percent confidence in the Fischer assay relationship.

Figures 3.10 and 3.11 also demonstrate the relationship reported by Merrill (1954), Podio et al (1968), and Raju (1961) that the P-wave velocity parallel to the bedding exceeds the P-wave velocity perpendicular to bedding. These figures also imply that this velocity difference increases with increasing Fischer assay. The compressional stress wave generated by blasting or crushing will probably be more effective in inducing fracturing parallel to bedding because compressional wave energy will be trapped between beds, or layers. This factor will become slightly more important to rock breakage by blasting at higher oil yields.

Raju's (1961) projectile impact studies on oil shale demonstrated that high grade oil shale is more readily removed by projectile impact than low grade oil shale. The maximum volume of shale removed by any given projectile impact occurred when the oil shale was thin bedded. Rapid alterations in the grade of oil between thin beds will produce the most efficient blasting results.

3.3.5 Creep Characteristics

The study of long term creep (plastic) deformation of oil shale by sellers et al (1972), indicated that no creep occurred below 2000 psi uniaxial compression. Fischer assay became a factor in creep above 2000 psi uniaxial compression. Specimens with Fischer assay above 30 gpt underwent permanent deformation when subjected to uniaxial compressive stress above 2000 psi. At stress levels below 6000 psi, the creep rate decreased to zero in a few days. Above 8000 psi, 30 gpt shale creeped to failure.

Triaxially confined compression test specimens of oil shale undergo massive plastic deformation while still carrying load. Confining pressures of 1500 psi permit at least 25% shortening to occur without brittle failure. High grade layers of oil shale extrude out of the specimen. The limiting strain of confined oil shale is unknown because the triaxial test chambers did not permit carrying such an

Experiment to an end point. The load carrying capacity of a triaxially confined oil shale specimen decreases progressively as creep deformation proceeds.

This phenomenon is important to the response of the confined central core of any overloaded pillar. Such an overloaded pillar would be expected to undergo measurable progressive deformation until it either shed its overload to adjacent pillars or failed.

3.4 CONCLUSIONS

The engineering properties of Green River oil shale near the edges of the Piceance Creek Basin, are directly related to organic content or Fischer assay. The strength and competency of the oil shale decreases with increasing Fischer assay. This study of the engineering properties of oil shale is incomplete. Additional triaxial and direct shear tests on oil shale are necessary to quantify the relationship between Fischer assay and those physical properties needed for rational design of an oil shale mine. The effects of disseminated saline minerals, present in the thicker, central Green River oil shale deposits, must also be evaluated to determine their effects on total strength.

SECTION 4
CONTRACT STUDIES AND INVESTIGATIONS

The objective of the phase I study is to evaluate all feasible systems for mining the thicker oil shale deposits in the deeper central part of the Piceance Creek Basin. Several basic criteria were used to select the most promising systems for evaluation. These included the extent of technical development of each system as well as its potential for high volume mechanized production at relatively low labor intensity and cost. Overall resource recovery, inherent safety, and environmental acceptance also were pertinent factors. Only demonstrated systems with reliable performance data could be effectively appraised.

The systems best meeting the basic criteria and selected for more detailed engineering and cost evaluation include block caving mining, two variations of room and pillar mining, and two variations of sublevel stoping. Several other possible candidate systems were investigated but rejected as not sufficiently meeting the basic criteria.

This section briefly reviews past and present oil shale mining as well as those candidate systems initially investigated, but rejected for further evaluation. It includes the design and analysis of common mine access and underground crushing and concludes with the design, costing, environmental, and safety evaluation of each selected mining system.

4.1 REVIEW OF LITERATURE ON UNDERGROUND OIL SHALE MINING

The principal period of development of underground oil shale mining was from 1945 to 1956, East and Gardner (1964). During these years the practicality of mining the 72-foot thick Mahogany Zone by a room and pillar method was demonstrated by the Bureau of Mines at Anvil Points, Colorado. A demonstration mine was originally planned for three levels, a 27-foot top heading and two 23-foot benches. However, mining of the lower bench was never attempted. A two level operation with a 39-foot top heading and 34-foot bench was also planned but not implemented. The mine was laid out with pillars 60-foot square in staggered rows 60 feet apart. Most of the equipment for drilling, blasting, and scaling the top heading and the benches was unique and was developed and built specifically for the project and later adopted by the mining industry. An electric shovel with a three yd^3 bucket was used to load the blasted oil shale into 18 yd^3 diesel trucks. The Bureau made three test runs at the Anvil Points mine to determine direct operating costs. The direct mining costs ranged from $0.46 per ton during August 1948 to $0.27 per ton during September 1949. Reductions in cost per ton for the later test runs reflected equipment changes and improvements in mining techniques. The average labor cost in those days was $1.81 per hour so the above costs are now obsolete. However, the costs were based on percussion drilling and with dynamite as the blasting agent, both of which were high cost items.

Experiments on rotary drilling were started early during the program because it was realized that lower mining costs could not be attained if the cost of drilling was not drastically reduced. The drilling rate with the percussion drills averaged 20 inches per minute with the cost of drill rods due to breakage at two cents per foot of hole. The early experiments with rotary drills proved that penetration rates of at least 60 inches per minute could be attained and that the cost of drill rods per foot drilled would be negligible. During the later years of the Bureau operation, great progress was made in developing better bits and in determining optimum operating conditions for rotary drilling.

Much rock mechanics research was done by the Bureau particularly during the early days of the program. The room spans and pillar dimensions of the demonstration mine were based on rock mechanics studies which indicated that 60-foot wide rooms and 60-foot square pillars on a staggered pattern should be safe. Shorter spans were used within 200 feet of the closest surface exposed to weathering because weakening of the rock by weathering had been measured within this distance. Two roof falls later occurred in the demonstration mine and both falls were in areas where 60-foot spans had been mined within 200 feet of an exposed cliff face. The first fall occurred more than 4-1/2 years after the room had been mined. The entire mine roof was bolted thereafter. The second fall occurred two years after mining a room 60 feet wide without staggering the pillars. It is believed that weathering, plus jointing, which is much more open and intense close to the cliff face in this area, were the major causes of the roof falls. There have been no more falls in the mine to the present time. East and Gardner concluded that the working areas of a commercial mine probably would not require roof bolting. However, areas such as haulageways that would remain in use for several years should be roof bolted as soon as possible.

From 1967 to April 1968 the Anvil Points Oil Shale Research Program was conducted by the Colorado School of Mines as research contractor for a consortium of six participating petroleum companies. The Mobil Oil Corporation was the project manager. A number of techniques and various types of mining equipment were tested, and an extensive rock mechanics study was undertaken (Zambas, Haworth, Brakenbusch, and Sellers, 1972, and Sellers, Haworth, and Zambas, 1972). A new mine close to the old Bureau of Mines mine was developed with rooms 60 feet wide and pillars 40 feet square. The top heading was 40 feet high with the benches 38 feet high. The pillars, as first mined, were 40 feet wide and very long with the extraction sequence planned so that most of the crosscuts connecting the rooms could be mined on the retreat. Drilling research was done with a rotary percussion drill that drilled holes a maximum of 30 feet long and up to five inches in diameter. A comprehensive experimental drilling program showed that the best results could be obtained with four to 4-1/2-inch drag bits that had a clearance angle of 17 to 18 degrees using the maximum thrust available to the drill jumbo (20,000 pounds). Penetration rates for a 4-1/2-inch bit were found to range from slightly over five feet per minute in 20 gpt shale, seven feet per minute in 30 gpt shale, and ten feet per minute in 50 gpt shale. A standard 28-hole heading round was developed for a face 40 feet high and 60 feet wide with holes 27 and 29 feet long. The average round pulled 25 feet and broke 4000 tons of oil shale per blast. The drill holes were pneumatically loaded with ammonium nitrate/fuel oil prills (ANFO) with a powder factor between 0.6 and 0.7 pounds per ton. It was later found during benching operations that excessive spalling occurred on the ribs of pillars mined with the standard heading rounds. Another heading round was designed incorporating 25 blast holes and 20 pre-spliting holes. Experimentation with benching rounds also indicated that presplitting would be desirable to minimize spalling. On bench rounds the powder factor was 0.30 pounds of explosive per ton of rock broken. The project was terminated before complete data on presplitting was collected, however, it was felt that the beneficial effects of presplitting would outweigh its higher cost. As a result of tests on a three yd^3 Caterpillar front end loader and a Koehring Scooper with a 6.7 and a 3.7 yd^3 bucket, it was concluded that a rubber tired front end loader would be more suitable for oil shale mining. Other recommendations resulting from the study were that drilling jumbos and roof bolting rigs could well be adapted to electric operation reducing ventilation problems.

During 1971 and 1972, considerable mining research was conducted at the experimental mine of the Colony Development Operation, Agapito (1974). In the mining method a top heading 29 feet high was mined first and then benched to a full height of 60 feet. Pillars were planned to be 60 feet square on a regular pattern with room spans of 60 feet. Plan maps of the mined areas show that some of the pillars were much less than 60 feet square and that some of the roof spans were considerably in excess of 60 feet. The irregular outlines of the pillars indicate some rather heavy blasting. One pillar, and later another, partially failed and subsequent stress measurements by overcoring showed a ten-foot deep blast damage zone in the pillars.

A paper presented by P. W. Marshall (1974) describes the mining plan of the Colony Development Operation for a commercial mine producing 66,000 tons of oil shale per day. Entry into the mine is by 30 by 30-foot adits, which are then widened to 50-foot haulageways inside the mine. The 60-foot thick section of oil shale is mined by a top heading 30 feet wide and not more than 60 feet wide. Drilling is with two boom rotary drill jumbos and rounds are blasted with ANFO primed with dynamite and non-electric blasting caps. Broken rock is loaded by 10 to 15 yd^3, front-end loaders into 80-ton trucks, which haul to a primary crusher on the portal bench.

4.2 CANDIDATE UNDERGROUND MINING SYSTEMS

The following discussions are reviews of candidate mining systems that could be used to mine oil shale underground. In each discussion, the mining method is investigated as to its capability for large scale production, mechanization, low production cost, and impact on the local environment. The mining systems that were selected for further study are presented in succeeding sections.

4.2.1 Square Set Stoping

Square setting, Haffner and Hoskins (1973), Dravo Corporation (1974), and Woodruff (1966), is a method of mine timbering in which heavy timbers are framed together in rectangular sets, six or seven feet high and four to six feet square, to hold heavy ground. In square set stoping, a small block of ore is blasted, extracted, and timbered with a timber set before the next block is mined. The method is flexible, selective, and is a safe method of extracting weak ore confined by weak walls. It is most commonly employed as an auxiliary method, or in combination with some other method, to mine remnants of ore such as pillars between filled stopes. Productivity with this method is low, and cost per ton is very high. In addition, consumption of timber is extremely high, this fact in itself precluding the adoption of square set mining for the underground mining of oil shale.

4.2.2 Shrinkage Stoping

Shrinkage stoping is a modification of overhand stoping in which broken ore is used as a working platform and to support the walls of a stope, Hoover (1973), Dravo Corporation (1974), and Woodruff (1966). Mining proceeds by breasting or back stoping to the level above, with the miners standing on the broken ore in the stope. As broken rock occupies more volume than solid rock, about 40% of the ore is drawn off during the stoping operation. Shrinkage stoping is applicable to regularly shaped, tabular deposits that dip more than 50 degrees, and to wide ore bodies which can be mined by transverse shrinkage stopes separated by pillars. The ore must be strong because the men work under an unsupported back or roof extending the full width of the stope. To minimize dilution of the ore and blockage of draw holes, the walls must be sufficiently strong to stand with very little support. A shrinkage stoping method in oil shale would have to use stopes separated by oil shale pillars because there is no physical difference between wall rock and "ore." Thus shrinkage stoping would be applicable only to relatively strong, competent oil shale.

The principal advantages of shrinkage stoping as it might be applied in oil shale mining are:

1. Stope backs and walls are readily accessible for inspection and scaling,
2. Stopes can be well ventilated without much difficulty,
3. Development costs are relatively low.

Disadvantages of shrinkage stoping are:

1. Only the "swell" rock can be removed while stoping is in progress (this amounts to about 40% of the ore, so the remaining 60% cannot be drawn until stoping is completed),
2. Draw control to provide an even working floor is difficult,
3. The method does not lend itself readily to mechanization.

The last disadvantage is critical and in itself overrides the advantages because there are other methods for mining strong, competent oil shale, such as sublevel stoping or room and pillar mining, that do lend themselves to mechanization, high productivity, and low production costs.

4.2.3 Cut and Fill Stoping

Horizontal cut and fill stopes are overhand excavations in which the ore is mined by a series of horizontal cuts. After all the ore from one cut is removed, an equal depth of fill is introduced for support of the walls and as a base for mining the next cut (Rausch and Stitzer, 1973

Dravo Corporation, 1974, and Woodruff, 1966). Miners work under an overhanging back; therefore, the ore must be fairly strong and competent although loose sections of back can be supported by timber stulls if necessary. Wall rock may be weak because it is soon supported by fill material

In metal mines, fill material may be waste rock or classified mill tailings, sometimes combined with alluvial sand. Hydraulic fill is taken underground and distributed through a pipeline system that may become a major expense item. Fill for the top few inches of each pour is usually mixed with cement to form a good working floor. For oil shale mining, the fill material would most likely be spent shale from the surface retorting process. Spent shale performance as a fill material, or the amount of classification or other treatment necessary to make it a suitable fill material, must be investigated further and is dependent upon the retorting process.

One very important advantage of cut and fill stoping is that it lends itself to mechanization. With ramp access to the stopes, loading and transportation can be accomplished by more efficient LHD units with drill jumbos used for stope drilling.

The disadvantages of the method are: (1) the cost of operation and maintenance of the fill system, (2) the fact that stopes are nonproductive during the fill cycle, and, (3) the large number of stopes required to maintain production.

Cut and fill stoping could be used to mine oil shale that is relatively strong and competent, but as a filled stope method in relatively competent shale, it would be more expensive than sublevel stoping with filling. Consequently it has not been investigated further as a viable method for mining oil shale.

4.2.4 Sublevel Caving

In sublevel caving the ore is mined downward from a series of sublevels, using fan blasts to break the ore (Haycocks, 1973, Dravo Corporation, 1974, and Woodruff, 1966). Sublevels have a vertical spacing from 30 to 50 feet and are driven from ramps between haulage levels. The method is represented diagramatically in Figures 4.1 and 4.2 (Haycocks, 1973).

Sublevel caving is applicable to massive and medium width ore bodies. It can be used to mine ore that ranges from weak to strong. Capping should cave easily but should not break fine in comparison to the ore or excessive dilution will result. Under optimum conditions a recovery of 85 to 90% of the ore with 15 to 20% dilution can be expected (Just, 1972, Cox, 1967, and Janelid and Kvapil, 1966). As a method for mining oil shale, it would appear to be suited to mining zones of shale that range from weak to moderately strong.

The advantages of sublevel caving in oil shale are:

1. The method lends itself to a high degree of mechanization with LHD's used for loading and transportation in the stopes and jumbos used for drilling,
2. The method is safe because miners work in drifts rather than under large spans of unsupported back,
3. Sublevel caving ranks among the lower cost, large scale mining methods.

The disadvantages of sublevel caving in oil shale are:

1. If the capping does not cave readily a potential for air blasts results,
2. The large number of dead-end sublevels must be ventilated with tubing, presenting a major ventilation problem when considering gassy conditions,
3. A relatively high consumption of explosives and a low tonnage per foot of drill hole.

Sublevel caving is a possible method of mining weak, jointed zones of oil shale because of the high productivity attainable through mechanization. However, for oil shale mining it must be compared to a caving or an induced caving method of mining. In view of the relatively high cost per ton for explosives, drilling supplies, and the problems

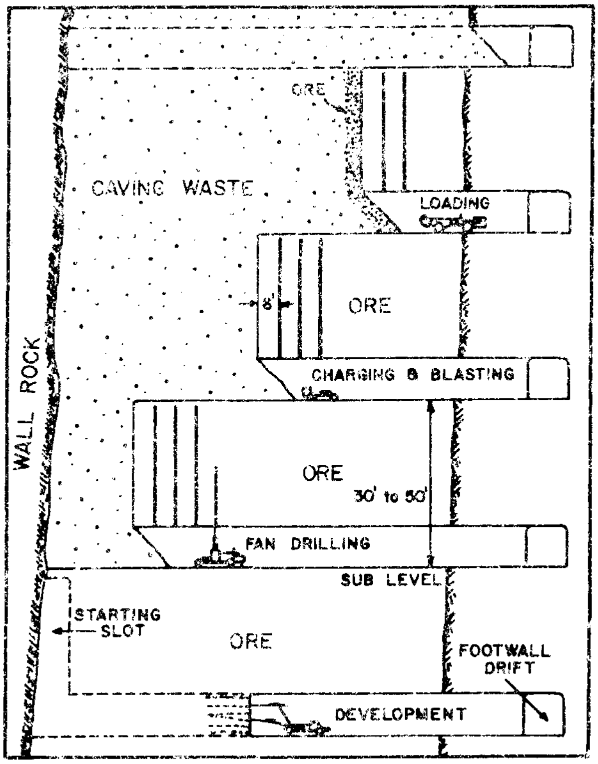


Figure 4.1 Side View of a Typical Sublevel Caving Operation --from Haycocks (1973)

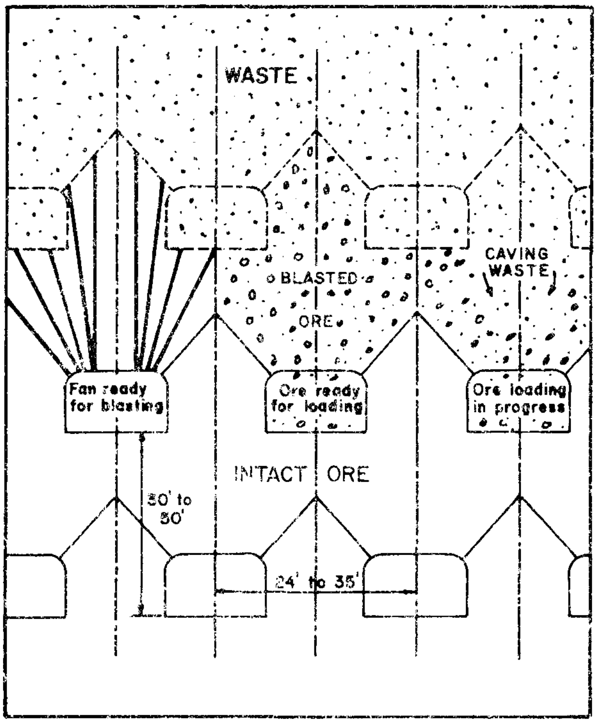


Figure 4.2 Cross Section of a Typical Sublevel Caving Operation--from Haycocks (1973)

associated with ventilating sublevel caving stopes, a caving or induced caving method is a better first choice for detailed study of a method suitable for mining weak zones of oil shale.

4.2.5 Longwall Mining

The longwall mining method is most successful in mining extensive, fairly flat-lying, thin to moderately thick sedimentary deposits, principally coal seams (Laird, 1973, and Woodruff, 1966). It is applied most readily to regular deposits that are not subject to sudden changes in dip, thickness, or elevation. The deposits must also be overlain by a series of beds several times the thickness of the deposit that cave readily without cantilevering too far beyond the face supports. In this system, large areas or panels of coal are completely extracted in a single, continuous operation. Panel dimensions vary considerably from mine to mine. A panel length of 5000 feet is common in the U.S. with the width of the panels at least 300 feet and commonly about 600 feet. The longwall mining method consists essentially of three systems that interact with each other: (1) the roof support system, (2) the extraction-loading system, and (3) the transportation system.

The transportation system consists of a chain face conveyor, a "motherline" or stage loader, and a panel entry belt conveyor. Coal is extracted or broken from the face and loaded onto the chain conveyor by either a plow or a shearer. A plow is a cutter loader with picks or blades that are mounted on the face conveyor and pulled along the face by a chain. A shearer is also mounted on the conveyor but has a rotating drum armed with picks or cutter bits for mining the coal.

The roof support system is probably the most important of the three systems. This system can in turn be divided into three problem areas: the face area, the tailgate area, and the headgate area. The support problems in the three areas are somewhat different and call for different solutions. The principal or critical area of concern is the face area, insofar as the practicality of longwall mining is concerned. Support problems are especially important with regard to applying the longwall method to mining oil shale. The transportation system would present some design problems to achieve the capacity desired for oil shale mining, but could be resolved without much difficulty. The problem of designing an extraction-loading system appears more difficult but also not insurmountable. Oil shale can be drilled with rotary drills using drag bits; therefore, designing a shearer loader or similar machine to mine the richer zones of oil shale seems within the limits of available technology and materials. This is indicated by the fact that a boring-type machine, the Alkirk Oil Shale Miner (Hamilton, 1965, and Carver, 1965) for mining 30-foot diameter drifts in oil shale has already been proposed and preliminary engineering studies indicate possible feasibility.

In modern longwall operations, the face area can be supported by any one of various types of self-advancing, hydraulic jack units. These units hold up a sufficient area of exposed roof rock to protect the men and the plow or shearer, but allow the roof behind the support units to cave. The height of this caved zone is important and can vary considerably depending upon the structure and physical characteristics of the roof rock and on the thickness (height) of the seam that is being extracted. The main roof has to be self-supporting even though the roof is highly jointed as well as bedded. The face supports, therefore, must be capable of supporting a dead load of rock having a column equal to the height of the caved zone, times the length of the face, times the width of the face area supported by the chocks, plus the maximum distance the roof strata will cantilever beyond the support units before caving.

The required capacity for longwall support units thus depends primarily on the following parameters: the face area that each unit is required to support or the distance back from the face that must be protected, the maximum distance that roof strata will cantilever before failing, and the height of the caved zone. The height of the caved zone in turn depends primarily upon the height of material mined and the swell factor for the roof rock. Although the values of these parameters with respect to mining oil shale are not known, indications are that oil shale would have a marked tendency to cantilever and probably would have a low swell factor in the caved zone. Thus very high capacity, and consequently very expensive support units will probably be required for mining relatively thin beds of oil shale five to six feet thick.

Two novel methods for mining oil shale by the longwall method have been suggested. Johns (1974) has a patent on a method and apparatus for mining oil shale and other "friable mineral deposits" that employs an "arch shield" concept for support. The arch shield support resembles one half an arch with its top resting against the top of the face. The proposed method for mining a thick zone of oil shale is to start at the bottom of the zone and mine a cut 10 to 12 feet high over a long distance. A subsequent cut would be made through the caved zone in the same area. Cuts would be repeated until the entire thickness had been mined. One of the problems occasionally encountered in conventional longwall mining is that of a void area forming in the roof so that a fully extended jack cannot reach the roof. This can happen when there has been a roof fall between the face and the cantilever canopy of the support units on a previous cut. The void area presents a dangerous situation and delays production if it is necessary to place timber between the top of the void and the top of the support unit. This situation may never occur on the first cut using this method, but would be a common occurrence on subsequent cuts. Voids would also occur near the top of the face and thereby remove the necessary abutment for the top of the arch shield. For this reason it is felt that this method has not been developed enough to warrant its investigation as a prime method for mining oil shale.

Another longwall method for mining thick beds of oil shale was proposed by Waltch and Rausch (1956). This method is based on a system first introduced at Warwickshire, England for mining coal seams 18 to 24 feet thick at a depth of 2,250 feet. At Warwickshire coal was extracted in panels approximately 900 feet by 2100 feet by first mining the bottom six to eight feet of coal. As the face retreated, pack walls of waste material were built which prevented the roof from caving but allowed it to subside gradually until complete closure took place. The next six to eight feet of coal were then mined under a roof that had subsided but had not caved. The same method is proposed for mining the 72-foot thick Mahogany Zone with panels 1000 feet wide by one mile long and with lifts eight feet thick. The method also requires mining oil shale conventionally by drilling, blasting, loading, and conveying and using a portion of the broken shale to build pack walls. This mining system presents intriguing possibilities and it is not too difficult to visualize how large thicknesses of oil shale may be mined by this method without causing the roof to cave. However, mining must be done over a large enough area to permit reducing the mined thickness at the boundaries. Pack walls could be built with spent shale placed pneumatically and the mining done with longwall units. Under these conditions the support units act more as safety canopies than as actual support units, making possible lighter and less expensive supports.

Longwall mining of oil shale is a distinct possibility; however, a large amount of development work must be completed and it is believed that other methods offer better promise of being economic in the near future.

4.3 MINE ACCESS AND UNDERGROUND CRUSHING FOR CANDIDATE MINE DESIGNS

Analyses of the mine designs in the following sections all have two areas in common: mine access and crushing. The phase I investigation imposes the limitation that all initial concepts must consider an ore body and overburden depth that is average for the deep deposits of the Piceance Creek Basin. The major differences in design among the various systems being investigated occurs only in the mining method. Therefore, it becomes convenient to evaluate the design and cost of various access hoisting, and crushing systems in this section and use the data in individual analyses.

The majority of data used for determining system capabilities and costs were developed from current vendor data and a report by Dravo Corporation (1974).

4.3.1 Mine Access and Hoisting

Estimates of construction time and costs for the various systems investigated are based on the following assumptions: (1) vertical production and service shafts are sunk to the 3000-foot level of the mine before mine development starts, (2) ground conditions are considered to be average, (3) a minimum of 85,000 tpd must be hoisted to the surface, and (4) the systems must be capable of multilevel operation. All conventional shafts have steel reinforced concrete lining.

Mine access is broken into three basic areas: ventilation, service, and production. The requirements for each system are different and, therefore, are investigated separately.

4.3.1.1 Ventilation

The large ventilation requirements of the candidate mining systems due to potentially gassy conditions and underground diesel equipment will probably require separate exhaust shafts. The size of the shafts can be reduced by not including any service or production equipment to impede airflow. Two methods of shaft sinking, conventional and boring, are evaluated for shaft diameters up to 20 feet. Shafts up to 28 feet in diameter are sunk using conventional methods only. Table 4.1 is a breakdown of the estimated construction costs for vertical shafts on a per foot basis.

4.3.1.2 Service

Access for men, materials, supplies, and intake ventilation must not interfere with production hoisting. For this reason, a separate service shaft is provided that has the capacity to lower necessary equipment into the mine. Dimensions and weight of some of the large production items necessitate using a minimum shaft diameter of 28 feet. The hoist system used in this shaft is a double drum because of the multilevel nature of mining. Table 4.2 lists the estimated cost of a double drum service hoist.

Another system of service access to the mining levels is an incline from the surface. The incline, approximately -15°, would serve a dual role as both service and production access. If the incline is to be used for

Table 4.1 Estimated Construction Costs for Vertical Shafts

Description	Cost per Foot			
Conventional Sinking	12-ft Dia.	16-ft Dia.	20-ft Dia.	28-ft Dia.
Labor	$364	$455	$546	$774
Construction material and equipment	$347	$433	$520	$736
Contractor overhead and profit	$156	$195	$234	$332
Total Cost	$867	$1,083	$1,300	$1,842
Excavation time (3000-ft)	74 weeks	82 weeks	89 weeks	125 weeks
Raise Bored	12-ft Dia.	15-ft Dia.	20-ft Dia.	
Labor	$125	$142	$170	
Construction material and equipment	$293	$366	$488	
Contractor overhead	$105	$127	$166	
Total Cost	$523	$635	$824	
Excavation time	50 weeks	63 weeks	84 weeks	

Table 4.2 Estimated Costs of Hoisting Equipment Including Construction Hoist Building and Skip Pockets

Description	Cost	Power (KW/H)
Double drum production hoist (2,200 tph)	$6,700,000	7,460
Double drum service hoist	$3,222,500	3,730
Counterweighted friction hoist (1,100 tph)	$3,866,000	5,600

service and production, care must be taken to ensure the production belt is isolated from the intake air. Table 4.3 lists the estimated costs for a -15 degree incline, 18 by 20 feet in cross section.

4.3.1.3 Production

The design of shafts for hoisting ore depends primarily on the rate of production desired through each shaft. Initial production levels will be 85,000 tpd with capability for expansion up to 170,000 tpd. For this reason it is desirable to consider having two or three shafts devoted entirely to production. Hoisting capacity per shaft would then be 42,500 or 29,000 tpd, respectively. Evaluation of current hoisting systems and designs indicates a 28-foot diameter shaft (Table 4.1) and an automated production hoist (Table 4.2) are capable of handling the total tonnage in either two or three shafts.

An alternative system of hoisting production ore is an inclined shaft and conveyor system. Table 4.3 shows the estimated construction costs, including the conveyor belt installation. An advantage to the conveyor system is in the large production capability. A significant increase in daily production can be accomplished by simply increasing belt speed. Maintenance and power costs for a conveyor system are high and may offset any advantages.

4.3.2 Underground Crusher

Efficient handling and loading of ore on conveyor belts or in hoist skips is enhanced if the ore is crushed prior to hoisting. Underground crushers allow larger fragments of ore to be present in the muck, thereby reducing blasting costs.

Oil shale is most efficiently crushed using gyratory crushers over impact crushers. The costing in Table 4.4 is based on gyratory crushers with variable output rates and maximum lump size. The number of crushers depends on desired output rate and production shaft configuration. It may be possible to use one crusher for 85,000 tpd if all shaft pockets can be fed from one location. The more desirable plan would be two or more crushers. Also included in Table 4.4 are the costs of crusher room excavation and surge pocket construction.

Table 4.3 Estimated Construction Cost for a -15 Degree Production Incline (18-ft X 20-ft)

Description	Cost per Foot
Labor	$235
Construction materials and equipment expense	$224
Contractor overhead and profit	$100
Total	$559
Excavation time (11,600-ft)	186 weeks
Belt and structure cost (5,000 tph)	$3,774,000
Power required	16,200 KW/H

Table 4.4 Estimated Cost for a Primary Crusher Room, Crushers, and Surge Bins

Description	Cost	Power (KW/H)
Crusher room construction (20 ft X 50 ft X 80 ft)		
Labor	$300,000	
Construction materials	67,000	
Contract overhead and profit	128,000	
Total	$495,000	
Excavation, finished construction and crusher installation time	80 days	
Crushers and rotary dump cost		
60" gyratory crusher (2900 tph capacity)	$1,300,000	522
72" gyratory crusher (5500 tph capacity)	2,289,000	746
Rotary dump (80-ton cars)	260,000	
Total	$3,849,000	
Surge bin construction (3,300 tons)		
Labor	$190,000	
Construction materials	29,000	
Contractor overhead and profit	76,000	
Total	$295,000	
Excavation and finished construction time	45 days	

4.4 DESIGN AND ANALYSIS OF SUBLEVEL STOPING IN OIL SHALE

Sublevel stoping is a high-production, low-cost, open stoping method generally used in mines with competent ore and country rock. Ore bodies mined by this technique are usually tabular vein types that are fairly uniform and anywhere from 20 to 200 feet or greater in width. The sublevel stoping technique inherently requires a high development-to-production ratio that is partially offset if the preproduction development is in ore. Open stope production is characterized by longhole fan drilling from pre-developed sublevels from which the ore is blasted down in slices. A high degree of mechanization is required to handle the large tonnages of ore produced by this method.

A major disadvantage to sublevel stoping is the large initial capital expenditure needed before full production is realized. Once full production is reached, however, the cost per ton of ore produced by this method is one of the lowest of underground mining techniques.

In the sections that follow, a review of the pertinent literature on sublevel stoping and the constraints and limitations on the proposed design for sublevel stoping in oil shale is discussed. Two sublevel stoping designs are then presented; the first has full resource recovery with all pillars pulled and the surface allowed to subside, and the second is designed to leave all pillars and fill mined-out stopes with spent shale. Each design is explained in detail from preproduction through production and costing.

4.4.1 Review of Literature

A general description of sublevel stoping techniques is given by Haycocks, 1973, in the current edition of the SME Mining Engineering Handbook. Haycocks describes several sublevel stoping techniques and modifications including the Parallel-hole (Noranda) and Cascade (Mufulira) methods. He reports the average cost per ton of sublevel stoping as \$2.37, compared to \$4.97 for sublevel caving, \$6.69 for cut-and-fill and \$3.92 for shrinkage stoping. Some of the major advantages listed are: (1) increased safety because miners are always working in the cover of sublevel drifts, (2) a high degree of mechanization is possible, (3) ring drilling does not require blasting on shift therefore ventilation is improved, (4) draw control is good, and (5) total recovery can be quite high. Two major disadvantages are: (1) initial capital outlay for preproduction is high, and (2) flat dipping ore bodies are very difficult to mine by this technique.

A USBM research report by Dravo Corporation, 1974, titled, "Analysis of Large Scale, Non-Coal Underground Mining Methods," includes a review of sublevel stoping and descriptions of several systems now in use. Total ore recovery is described as being close to 100% depending on the difficulty of pillar recovery. Several different mining operations are described in the report. Cost distributions for one trackless sublevel stoping operation were divided as follows: development - 30%, load and haul -20%, stoping - 11%, hoisting - 5%, supervision and service - 14%, crushing and conveying - 10%, power - 3%, stopefill - 1%, general - 6%.

Graham (1968) describes the Whalesback Mine in Newfoundland and quotes a mining and milling cost of $3.02 per ton of ore. Figure 4.3 shows the mine and stope layout in use during that time. The author describes this method as longhole stoping, a term commonly synonymous with sublevel stoping. Although the Whalesback ore is relatively soft the sublevel stoping technique works quite successfully. The ore body is about 1200 feet long, varies in width from 10 to 130 feet, and dips at about 70 degrees. Level intervals are about 200 feet and crosscuts are driven on 200-foot centers. Operating costs were broken down as follows: development - 8%, stoping - 14%, mine indirect - 26%, milling - 15%, services - 21%, and general overhead - 16%.

The Algoma Mine in Canada (Beck, 1957) reported nearly 100% extraction of an ore body by mining the pillars in conjunction with sublevel stoping. The ore body was 200 feet wide, 700 feet long, and 300 feet high. Stopes were 60 feet wide going across the ore body and 200 feet high. Pillars were 80 feet wide and 70 feet thick. Sublevels were positioned at 65-foot vertical intervals. Longhole blasting into the

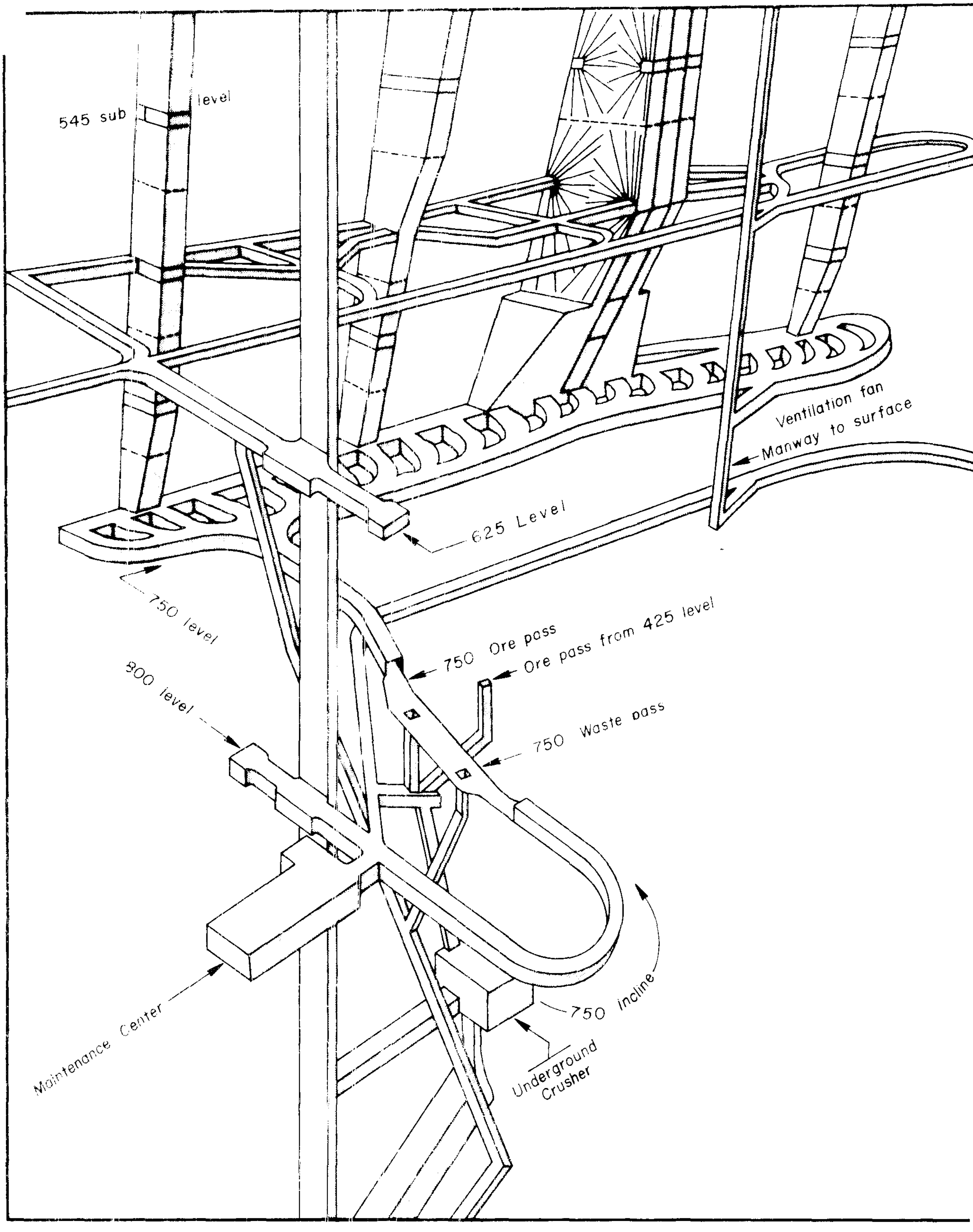


Figure 4.3 Isometric of Whalesback Mine Development and Stope Preparation -- from Graham, 1968

stope was done in a sequence that allowed the waste to subside with the draw. Pillars were blasted into the finished stopes as mining progressed from one end of the ore body to the other.

The Strassa Mine in Sweden (Mamen, 1969, and Dravo Corporation, 1974) also employs pillar mining in conjunction with sublevel stoping. Figure 4.4 is an isometric drawing showing the development, stoping, and haulage network. Ore mucking is by rail bound 21 ft^3 muckers dumping into five yd^3 rail cars. When a stope has been fully mined, the roof and rib pillars are blasted into the open stope with the triangular pillar blasted last. Cost distributions for the Strassa Mine are outlined as follows (Dravo Corporation, 1974): haulage, crushing and hoisting - 13.1%, development and production - 61.4%, and general mine services - 25.5%. Production at the time these figures were calculated was 1.73 million mtpy.

Other mines with stoping techniques similar to the Strassa Mine are the Sullivan Mine, Canada (Staff, 1957) and the Madeline Mine, Canada (Dravo Corporation, 1974).

Direct comparison of cost distributions among the mines described above is difficult because of differences in cost analysis techniques. However, it is apparent that development costs are approximately one-fourth the total cost of mining. It is obvious that if all development is done in ore a more favorable cash flow is realized and development costs are reduced.

4.4.2 Sublevel Stoping Design with Full Subsidence

As previously discussed, underground mining by sublevel stoping is generally limited to steeply dipping, tabular ore bodies that have definite waste cut-off boundaries. The oil shale deposit of the Central Piceance Creek Basin covers approximately 883,000 acres where the ore horizon varies in thickness from 50 to 2000 feet, averaging approximately 1000 feet. As a consequence the lateral extent of the oil shale deposit could be considered infinite in relation to the size of the mining operation.

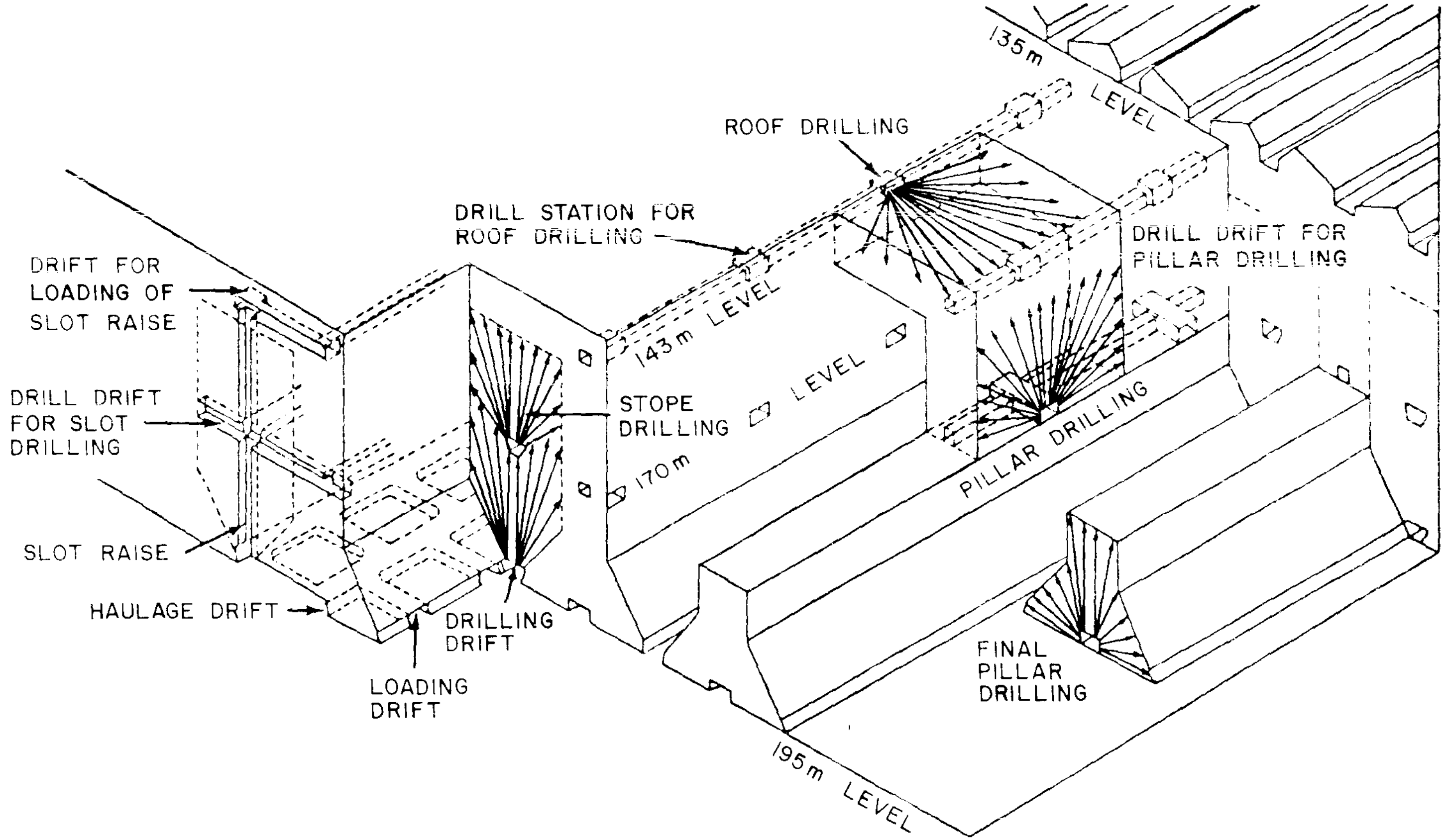


Figure 4.4 Sublevel Stoping at Strassa, Sweden -- from Dravo Corporation, 1974

The average overburden thickness being considered for this design is approximately 1000 feet and is the only waste rock through which development will have to be done. Once level development has started all mining is in oil shale and development costs are significantly reduced.

The possibilities of large inflows of water have been previously discussed in the section on geology and may have a critical impact on sublevel stoping. An overall mining plan with full pillar recovery requires that the overburden cave as oil shale is extracted. Large stopes will open a considerable area for water inflow which must be pumped back to the surface. Insufficient data exist to determine an accurate rate of inflow, consequently, a figure of 10,000 gpm is assumed.

Disposal of spent shale by two methods is considered. The first involves disposing of the spent shale in the surface subsidence depression as it is formed. This method is considered dangerous from the standpoint of possible inflow into working areas underground. However, it is felt that if the spent shale were deposited in a dry or relatively dry condition and water were not allowed to pond in the disposal area the danger of sudden inflow could be eliminated. The second involves disposing of the spent shale in large ponding areas on the surface away from the subsidence zones.

The mine dimensions used in the analysis of sublevel stoping methods are estimates used to determine approximate production costs. A more detailed analysis, not in the scope of this study, is required to determine optimum dimenisons.

4.4.2.1 Preproduction Development

Once the shaft and pocket development has been completed, an undercut level is started above the first stope level. This undercut level is necessary to start the overburden caving above the sublevel stopes. The mining method used on the undercut level will be room and pillar with approximately 75% recovery. Pillars and rooms will be 60 by

60 by 30 feet high. Initially two, 30 by 20-foot main haulage drifts will be driven a total distance of about 7000 feet to two 16-foot exhaust ventilation shaft sites (Figure 4.5). The exhaust ventilation shafts will be raise-bored from the 1000-foot level to the surface. The undercut mining will start at a desired distance from the shaft pillar (3040 feet in this analysis) and an area 5300 feet long by 540 feet wide mined out. A 200-foot barrier pillar is then left and the intermediate pillars are blasted to initiate caving. This procedure is repeated, retreating towards the shaft pillar, as required by mining progress on the level below.

Drilling on the undercut level will be with two-boom jumbos in the main headings and rooms. Mucking will be done using 12 yd^3 front end loaders (FEL's) in the main headings and the rooms. All muck will be dumped into 80-ton rail cars and hauled by 40-ton trolley locomotives to the primary crusher at the shaft or incline station.

Development on the first stoping level (350 feet below first undercut level) will start approximately three months after development has started on the undercut level. The same type of drilling and mucking equipment will be used to drive the 30 by 20-foot main haulage drifts. Primary concern will be to drive the main haulage drifts out to the exhaust ventilation shafts as was done on the undercut level. However, at the same time rail haulage cross-cuts and stope development will start whenever possible. Table 4.5 is a time schedule of preproduction to full production planning, not including shaft development. Estimated construction time for shafts, skip pockets, surge bins, and crusher rooms is 2.5 years. Total time to full production is estimated to be 3.5 to four years.

4.4.2.2 Main Haulage Layout

All development and production ore is loaded by FEL's into 80-ton rail cars and hauled by trolley locomotive to a crushing and ore transfer pocket. Rail haulage drifts are 30 by 20 feet with each haulage loop servicing four stopes (Figure 4.6). The design used in this analysis has five

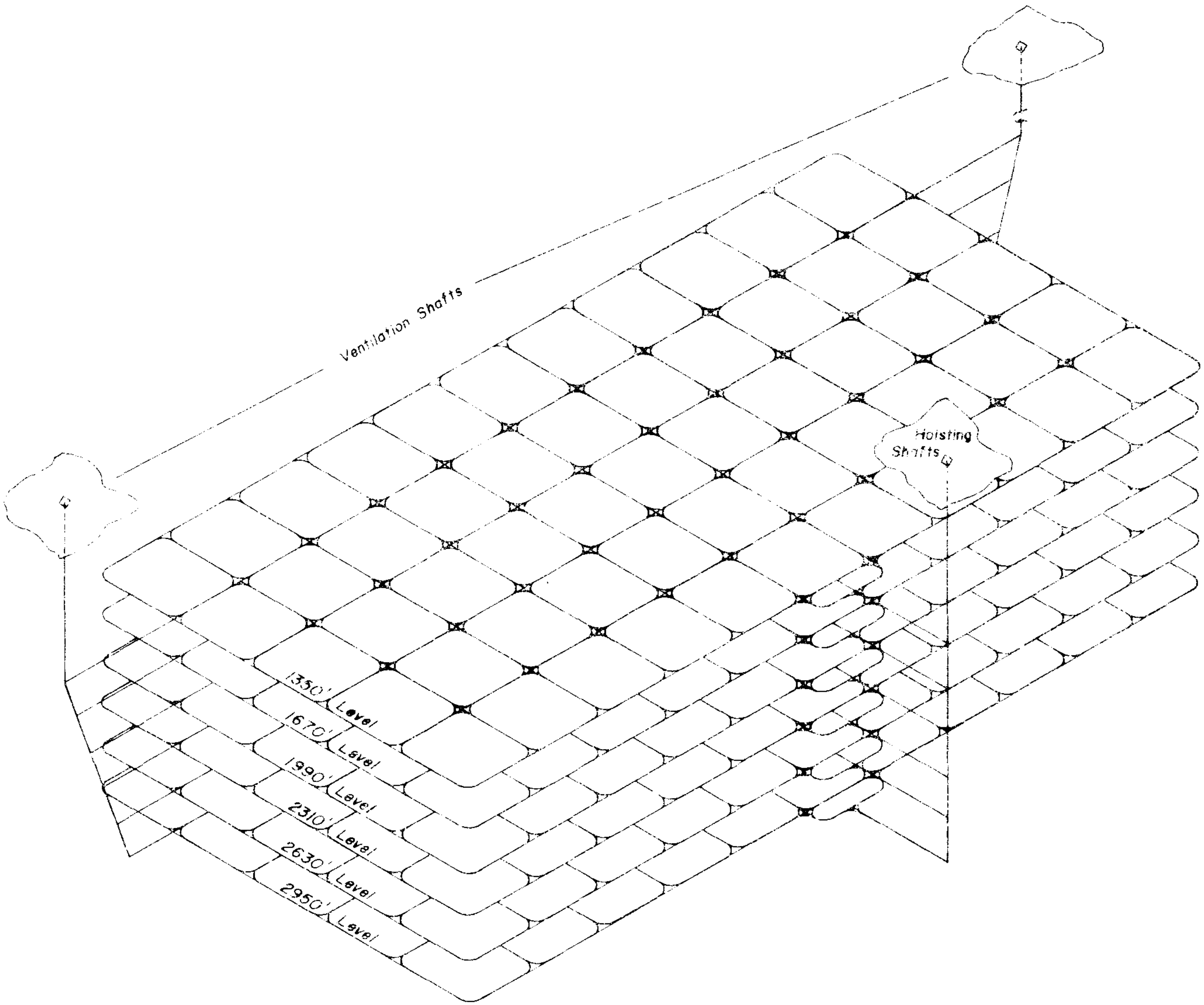


Figure 4.5 Multilevel Layout, Sublevel Stoping with Full Subsidence

Table 4.5 Schedule of Preproduction Work for Sublevel Stoping with Full Subsidence

Description of Task	Days	Ore Production (tons)
Undercut level, main haulage, ventilation drifts	152	320,000
Room and pillar mining of first cut	100	5,900,000
First stoping level; main haulage drifts, ventilation drifts	151	619,000
Raise bore ventilation shafts from surface	60	
Raise bore ventilation shafts from 1000' to 1350' level	30	11,000
First stoping level main haulage development to stopes	82	335,000
Sublevel ramp systems construction	36	65,000
Preparation of stopes 1 & 2	37	148,000
Full production stopes 1 & 2	111	990,000
Preparation of stopes 3 & 4	37	48,000
Full production stopes 3 & 4	111	502,000
Preparation of stopes 5 & 6	37	148,000
Full production stopes 5 & 6		
Room and pillar mining of next cut	87	1,030,000
	Total	10,216,000

Full Production

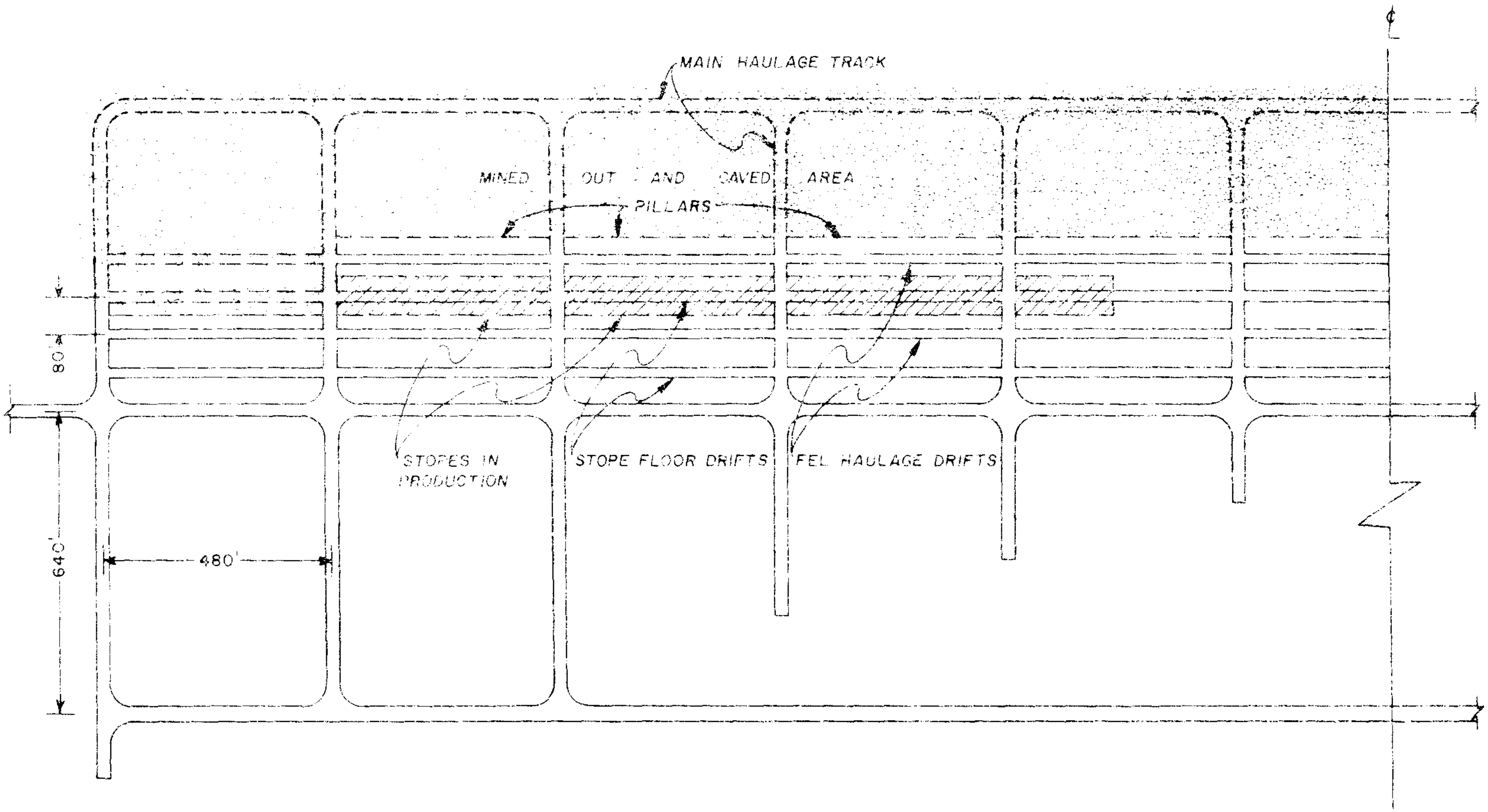


Figure 4.6 Rail Haulage Layout, Sublevel Stoping with Full Subsidence

loops on each side of the main line haulage to the ore transfer pockets. Stope development and production starts at the outer boundaries and proceeds inward. The general mining plan is retreat with the first stopes coming into production located at a predetermined distance from the access area. The actual size of the mine layout will be determined by the desired life of each operating level.

As stopes are mined out (one every 37 days) the rail haulage is developed for the next stope. At any one time, six stopes are operating and producing 12,000+ tpd. One 40-ton locomotive can service three stopes in an hour, hauling ten, 80-ton cars per stope per hour. As 60 cars are being loaded by the FEL's, the locomotives are hauling and dumping 20 cars. An additional 20 cars per shift are required for development ore and can be handled by the available locomotives.

4.4.2.3 Open Stope Production

When full production has been reached on the first level below the undercutting level, six stopes will be producing 12,000+ tpd for a total of 72,000+ tpd. Approximately 13,000 tpd will come from development and provide a total of 85,000+ tpd. Figures 4.7, 4.8, and 4.9 are plan and cross section views showing the proposed stope layout. Initial development of the first stopes would involve driving rail haulage loops, FEL cross-cuts, stope floor drifts, and a ramp up to the sublevel. A raise at the end of the stope (Figure 4.8) is bored and widened to 80 feet, the full stope width. When driving sublevels, these raises are used to drop broken rock to the haulage level. Longhole fan drilling from the 20 by 20-foot stope floor drift and the sublevel drift is done during development and production with automatic rotary drills using three-inch drag bits (Figure 4.9). Full production requires that at least one fan per stope be blasted each shift. A total of six fan drill jumbos are required for all stopes.

When a stope has been fully mined and the broken shale removed, the crown pillar, one end pillar, and one rib pillar are blasted into the open stope (Appendix A). The broken shale from the blasted pillars is

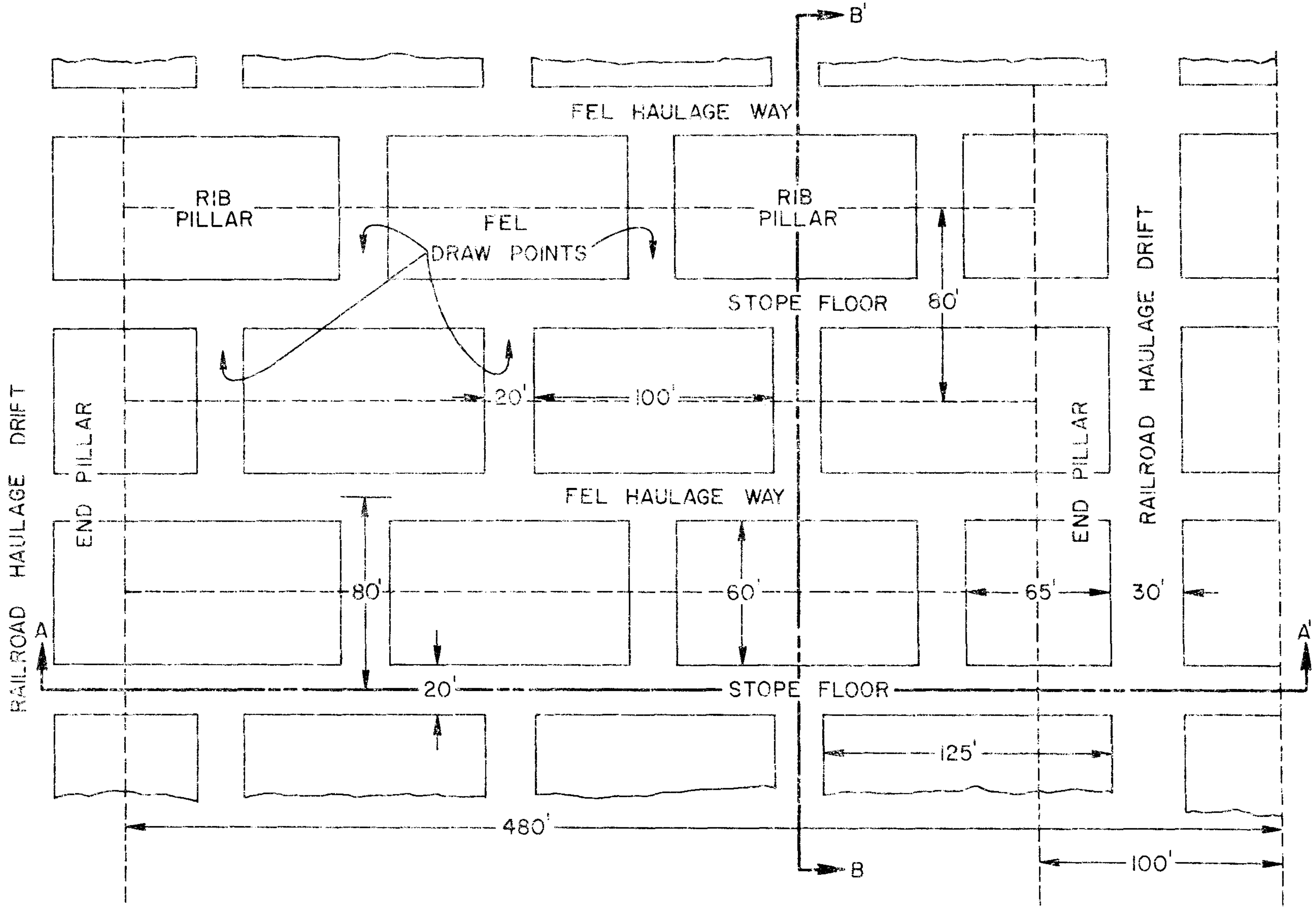


Figure 4.7 Plan View of Haulage Level, Sublevel Stoping with Full Subsidence

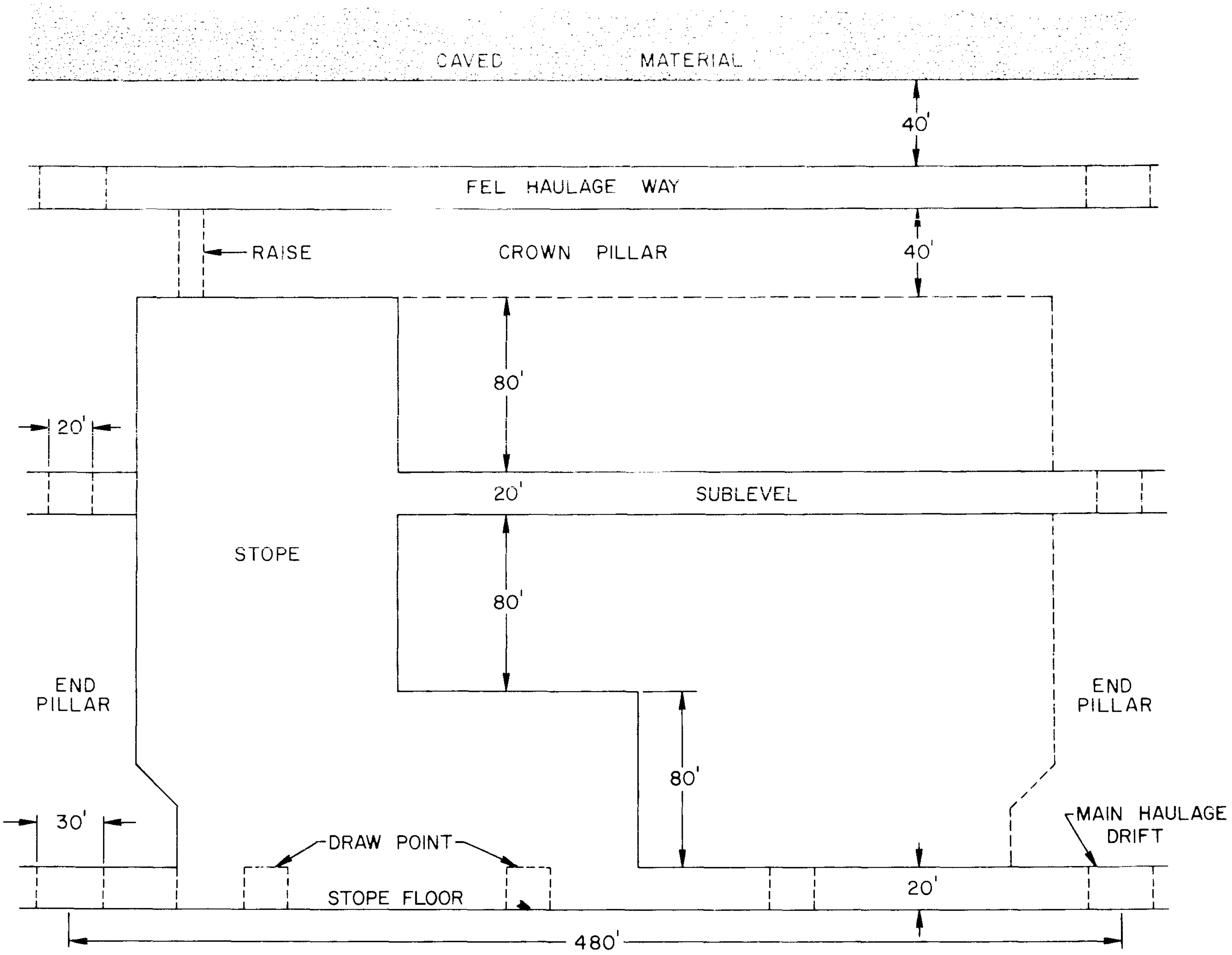


Figure 4.8 Longitudinal Section Through A-A', Sublevel Stoping with Full Subsidence

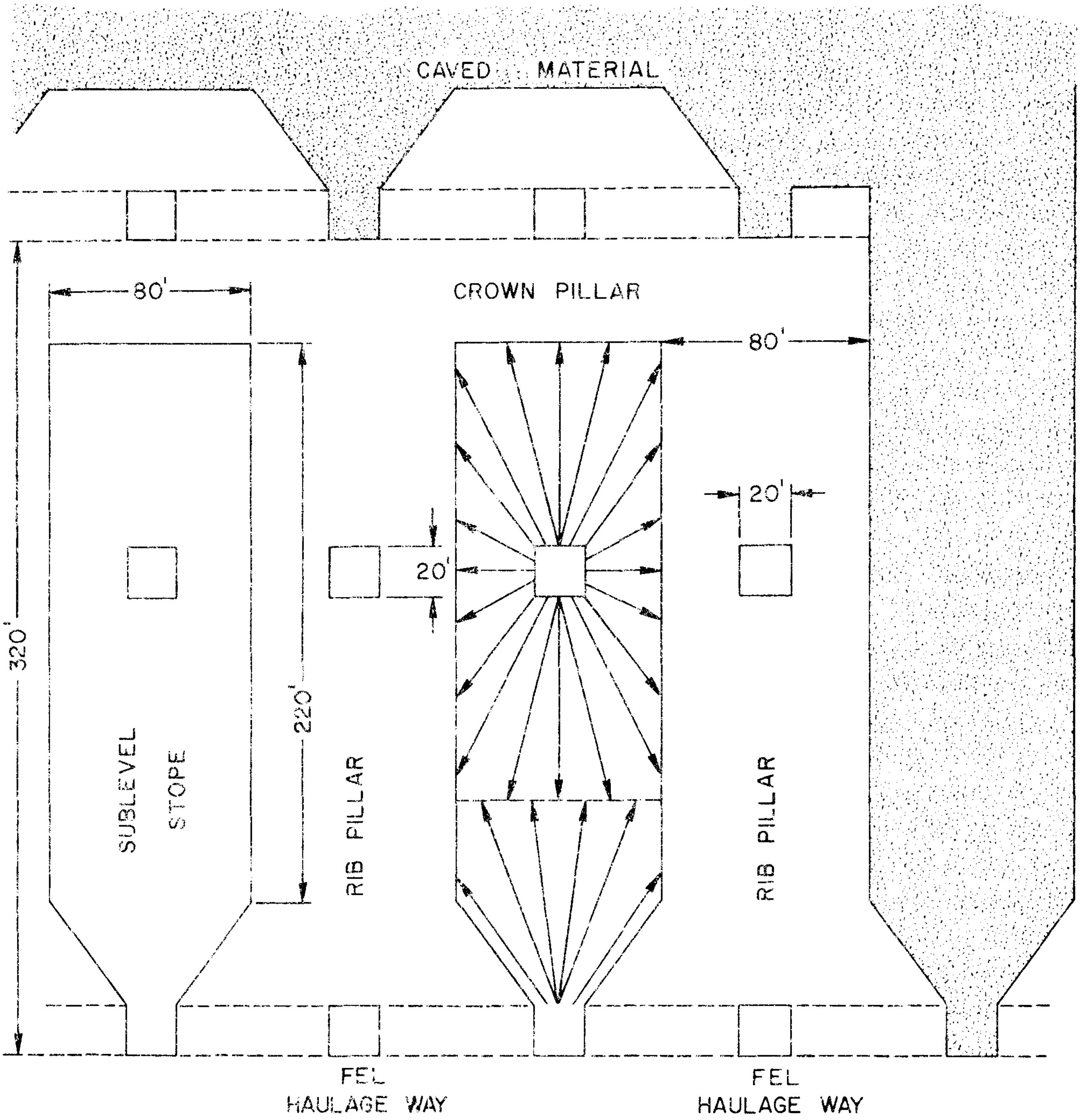


Figure 4.9 Vertical Section Through B-B', Sublevel Stoping with Full Subsidence

then recovered using the same stope drawpoints. The rail haulage drifts and FEL cross-cuts remain open during the entire life of the level and become the crown pillars for the stopes on the succeeding level (Figure 4.10).

4.4.2.4 Ventilation

The following discussion is based on rough estimates assuming gassy mining conditions and approximating the diesel horsepower output of mechanized equipment. Air circuit quantities and velocities will be investigated at a later phase if this method of mining is selected for more detailed analysis.

Colorado mining laws (Colorado Bureau of Mines, 1971 and State of Colorado, 1966) require 75 cfm of free air for each brake horsepower of diesel equipment used underground. The calculations for estimated mine ventilation requirements are shown in Appendix A. Results of the calculations indicate a need for about 2,000,000 cfm of free air, which includes an estimated 45 % leakage loss. A proposed fan arrangement would be to have axial-vane fans at the top of the raise-bored exhaust shafts sucking air. The intake shafts will be the production hoisting and auxiliary shafts.

Air flow will be controlled by conventional air doors and stoppings. Once a stope has been mined to completion, a permanent stopping is erected to reduce leakage into the caved area. Ventilation on the caving level will be limited only to the immediate working heading once full production is reached.

Auxiliary fans are needed on a temporary basis for development work on main haulage and sublevel stope drifts. Two 100,000 cfm axial-vane fans will be installed near the base of each sublevel ramp system to provide full sublevel ventilation. Additional 36,000 cfm fans will be used to provide ventilation for main line haulage development drifts.

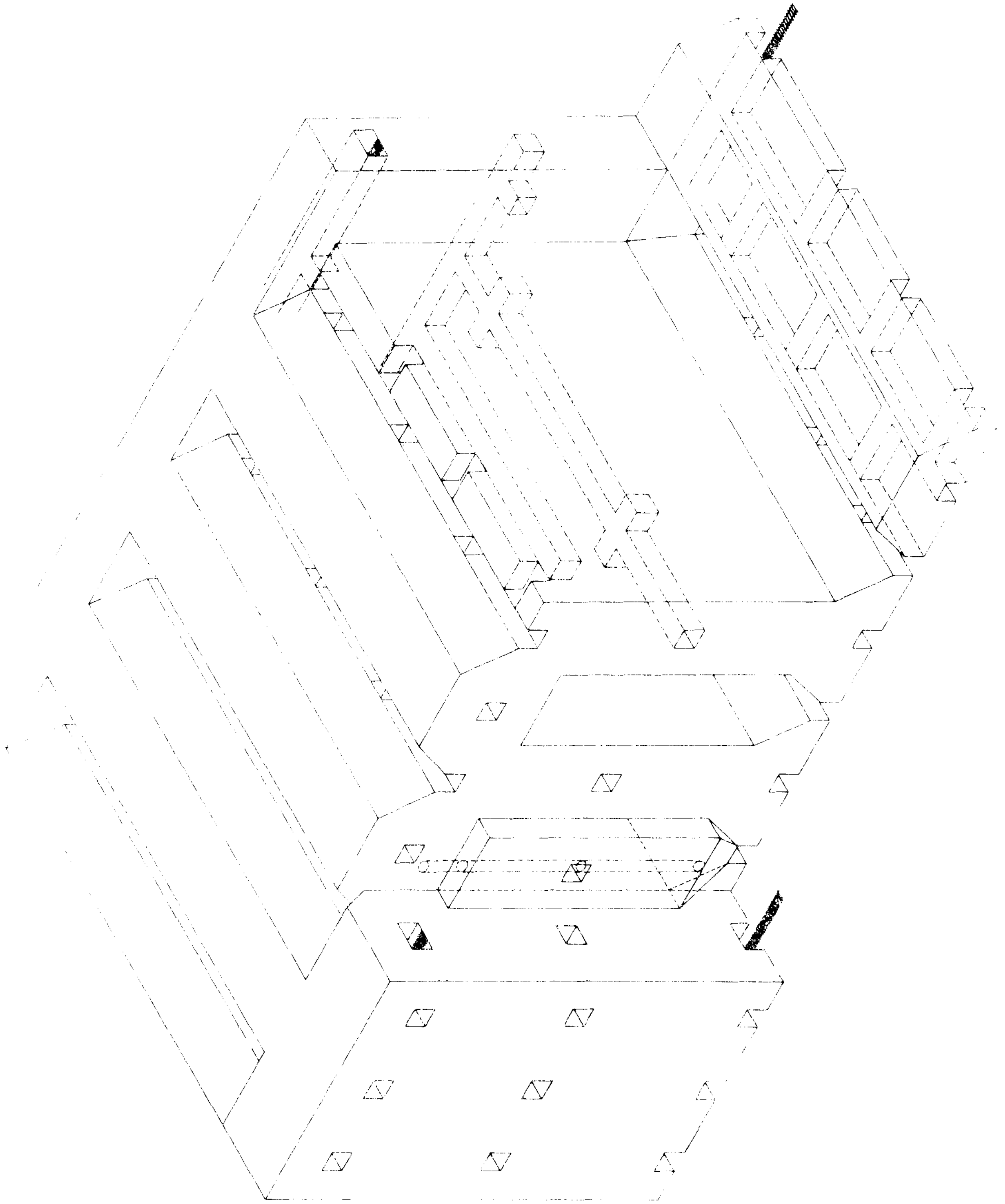


Figure 4.10 Isometric View, Sublevel Stoping with Full Subsidence

4.4.2.5 Production Equipment Selection

Appendix A contains the calculations that were used to estimate equipment size, quantities, and cycle times for major equipment items. The capital expenditure tables in the following section on cost analysis summarize the equipment selections.

Evaluation of capital equipment for this large tonnage operation has been done relying almost exclusively on current vendor data. Wherever possible, only that equipment currently manufactured, whether as a full production item or as a prototype, was selected for analysis.

For this initial evaluation operator efficiencies and minor equipment set up times were approximated by considering only five hours of working time per eight-hour shift. The mine is designed on a three-shift, seven-days per week work schedule. For the purpose of including holidays and unexpected work stoppages, a work-year was considered to be 355 days. Equipment availabilities are estimated and range from 65 to 90%, depending on use and past performance of similar models.

4.4.2.6 Production Cost

The method of production costing for sublevel stoping with full subsidence is similar to the format used by the USBM (Staff, 1972, Katell and Hemingway, 1974). This costing method necessarily assumes that the oil shale mine is the only income source to the corporation; therefore, development costs (including mine access and hoisting) are capitilized. In reality, these costs would most likely be charged to production and deducted as negative cash flow in the overall corporate cash flow determination. Excluded in this analysis are royalty payments, welfare payments, and surface transportation costs. All costs were collected during first quarter, 1975.

Table 4.6 lists the supervisory and hourly personnel needed for the operation of an 85,000 tpd mine. An additional 12% of the hourly manpower requirements are included to account for absenteeism. Figure

Table 4.6 Manning Table, 30.175 MM tpy
(Sublevel Stoping with Full Subsidence)

Personnel	Total	Annual Cost per Employee	Annual Cost (260 Workdays)
Salary			
Superintendent	1	$33,000	$33,000
Technical superintendent	1	25,000	25,000
General mine foreman	1	27,000	27,000
Mine foreman	1	22,000	22,000
Shift foreman	3	20,000	60,000
Section foreman	6	18,000	108,000
Chief mechanical foreman	1	21,000	21,000
Shift mechanic	3	19,000	57,000
Section mechanic	6	18,000	108,000
Chief electrical foreman	1	21,000	21,000
Shift electrician	3	19,000	57,000
Chief engineer	1	25,000	25,000
Mining engineer	5	20,000	100,000
Surveyor	3	10,800	32,400
Surveyor helper	3	9,000	27,000
Draftsman	2	9,600	19,200
Mine geologist	1	18,000	18,000
Safety director	1	19,000	19,000
Safety man	3	16,500	49,500
Accountant	3	9,600	28,800
Bookkeeper	1	7,500	7,500
Purchasing agent	1	14,400	14,400
Warehouse supervisor	1	13,200	13,200
Warehouseman	3	7,200	21,600
Watchman	3	6,000	18,000
Secretary	3	7,800	23,400
Subtotal	61		$956,000

Table 4.6 Manning Table, 30.175 MM tpy
(Sublevel Stoping with Full Subsidence) con't

Personnel	Total	Wages per Day	Annual Cost (260 Workdays)
Underground			
Drilling operator	43	$51.81	$579,200
Drilling operator helper	43	49.96	558,600
FEL operator	61	50.88	807,000
Scaling and rock bolt operator	86	52.81	1,180,800
Powderman	58	52.81	796,400
Truck driver	22	50.88	291,000
Trackman	29	50.42	380,200
Laborer	22	50.88	291,000
Locomotive operator	14	51.81	188,600
Mechanic, first class	11	56.58	161,800
Mechanic, second class	29	52.90	398,900
Electrician, first class	4	56.58	58,800
Electrician, second class	7	52.90	96,300
Master machinist	4	56.58	58,800
Machinist	14	51.80	188,600
Welder	7	51.80	94,300
Subtotal	454		$6,130,300
Outside			
Hoistman	4	$51.70	$53,800
Cage tender	4	49.77	51,800
Lampman	4	47.15	49,000
Subtotal	12		$154,600
Contingency for absenteeism (12%)	56		$754,200
Total labor and supervision	583		$7,995,100

4.11 is an example of the anticipated management flow chart for the underground mine only. Table 4.7 is the capital investment summary that includes contingencies, mobilization of capitalized equipment to mine site, consulting, environmental impact statement, mine access, and preproduction development. The total capital investment is estimated to be $113,500,000, excluding the value of oil in the development ore (estimated to be worth $53,500,000).

Table 4.8 is the straight-line depreciation schedule for the equipment listed in Table 4.7. Table 4.9 is the estimated yearly interest cost on the money borrowed to finance the capital equipment. In this table, interest rates from eight to ten percent are assumed; however a rate of nine percent is used in the cost summary. Notice that the mine development costs are capitalized over a period of 30 years. Power and water consumption, Table 4.10, is estimated from vendor data, and the electric rate as quoted by a local Colorado public utility for the Central Piceance Basin. Water cost is neglected because it is assumed more than enough water will be available from the mine itself. Table 4.11 is the estimated cost of preproduction, including the cost of interest on capital. The cost per ton figure is determined by dividing the total cost by the amount of preproduction development ore (10,216,000 tons).

Table 4.12 is a summary of the estimated annual production cost for an 85,000 tpd mining operation. It must be understood that this cost only includes mining costs and no processing costs. Also included are yearly interest, reclamation, exploration, and environmental monitoring costs. The total estimated cost per ton is $1.15.

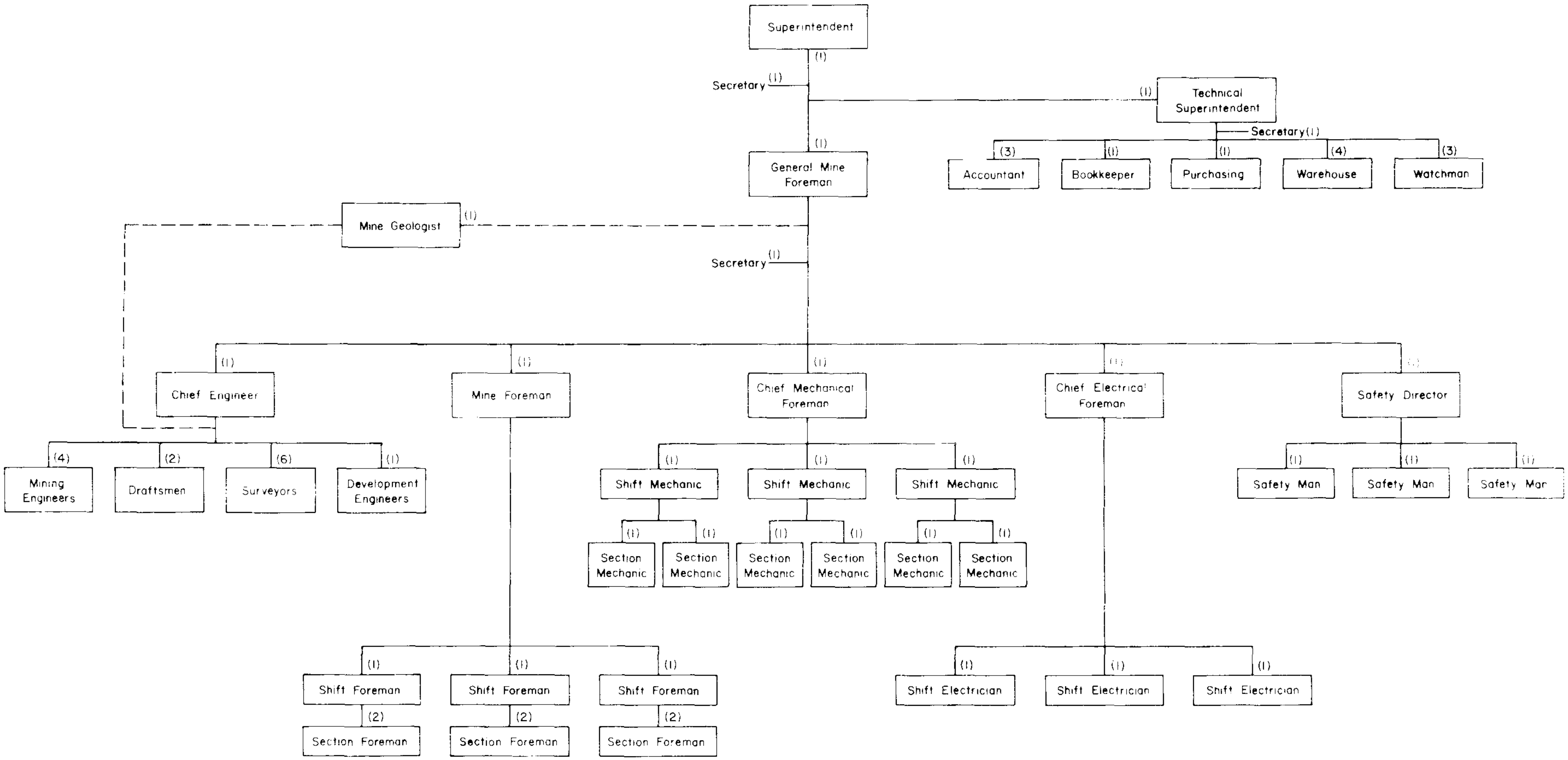


Figure 4.11 Management Flow Chart, 30.175 MM tpy, Sublevel Stoping with Full Subsidence

Table 4.7 Capital Investment Summary, 30.175 MM tpy (Sublevel Stoping with Full Subsidence)

ITEM	Quantity	Cost per Unit	Total Cost
Two boom hydraulic drill jumbo	8	$310,000	$2,480,000
Two boom hydraulic fan drill	7	300,000	2,100,000
40-ton trolley locomotive	3	228,500	685,500
30-ton trolley locomotive	2	189,500	379,000
FEL (12 yd^3)	24	280,000	6,720,000
80-ton rail car (air brakes)	120	23,000	2,760,000
Two drill rock bolt machine	14	150,000	2,100,000
Primary ventilation fan	4	115,000	460,000
Secondary (100,000 cfm) fan	2	8,600	17,200
Secondary (36,000 cfm) fan	4	7,500	30,000
Powder and ANFO loading truck	8	28,000	224,000
Water truck	2	30,000	60,000
Lubrication and fuel truck	4	30,000	120,000
Personnel carrier	6	41,500	249,000
Fire chemical truck	3	25,000	75,000
First aid kit	20	30	600
Lamp (including accessories)	430	85	36,600
Self rescuer	450	43	19,400
Stretcher set	9	235	2,100
Telephone (page phones)	9	450	4,100
Underground machine shop	1	1,000,000	1,000,000
Bathhouse, office and lamphouse	1	2,000,000	2,000,000
Warehouse and supply yard	1	130,000	130,000
Substation and distribution	1	900,000	900,000
Forklift	1	25,000	25,000
Exhaust fan (machine shop)	1	8,000	8,000
Rectifier (250 KW)	15	6,112	91,700
Water pumps (10,000 gpm total)	16	33,000	528,000
Track (115-lb)	70,000 ft.	$4.8/foot	336,000
Trolley wire	35,000 ft.	$0.68/foot	23,800
Track plates	1,800	$2.15	3,900
Treated ties	12,000	$6.64	79,800
Methane detector	20	235	5,000
Port. fire extinguisher (20-lb.)	20	45	900
High voltage cable (8KV-4°)	10,000 ft.	$7.2/foot	72,000
Track maintenance machine	1	50,000	50,000
Underground crusher	2	2,700,000	5,400,000
4-inch diameter pipe	20,000 ft.	$2/foot	40,000
Total direct cost.			$29,216,600

Table 4.7 Capital Investment Summary, 30.175 MM tpy (Sublevel Stoping with Full Subsidence) con't

ITEM	Total Cost
Contingencies (9% of total direct cost).	$2,629,500
Subtotal	31,846,100
Mobilization charge (2% of line above)	637,000
Total Construction	32,438,100
Consulting engineering fees, overhead and administration (6% of line above).	1,949,000
Subtotal	34,432,100
Environmental impact statement	2,000,000
*Mine Access (included are ventilation, production, and service shafts, three double drum production hoists, and one double drum service hoist)	47,000,000
*Preproduction cost.	30,067,000
Net capital investment	$113,501,000
Value of oil in 10,216,000 tons of 20 gpt oil shale stockpiled or retorted during preproduction development (assumes $11/bbl.)	$53,512,000

* Assumes capitalization

Table 4.2 Depreciation Schedule, 30.175 MM tpy
(Sublevel Stoping with Full Subsidence)

ITEM	Straight-line Depreciation (Years)	Yearly Charge
Two boom hydraulic drill jumbo	8	$ 310,000
Two boom hydraulic fan drill	8	262,500
40-ton trolley locomotive	15	45,700
30-ton trolley locomotive	15	25,300
FEL (12 yd^3)	8	840,000
80-ton rail car (air brakes)	10	276,000
Two drill rock bolt machine	10	210,000
Primary ventilation fan	30	15,400
Secondary (100,000 cfm) fan	10	1,700
Secondary (36,000 cfm) fan	10	3,000
Powder and AMFO loading truck	10	22,400
Water truck	10	6,000
Lubrication and fuel truck	10	12,000
Mine safety equipment	5	28,800
Personnel carrier	10	24,900
Underground machine shop	30	33,300
Bathhouse, office and lamphouse	30	66,700
Warehouse and supply yard	30	4,300
Substation and distribution	30	30,000
Forklift	10	2,500
Exhaust fan (machine shop)	30	300
Rectifier (250 KW)	15	6,100
Track and trolley material	30	14,800
High voltage cable (8KV-4")	30	2,400
Track maintenance machine	15	3,300
Underground crusher	30	180,000
Shaft access and ventilation	30	1,567,000
Water pumps (10,000 gpm total)	15	35,200
Preproduction development	30	1,000,000
Environmental impact statement	30	66,700
Contingencies, mobilization, and engineering	30	174,000
Total		$5,270,300

Table 4.9 Estimated Yearly Interest Cost on Capitalized Items, 30.175 MM tpy (Sublevel Stoping with Full Subsidence)

Depreciable Life (yrs)	5	8	10	15	30	Total
Capital Required	144,000	11,300,000	5,585,000	1,734,000	94,638,000	113,401,000
Interest Charge						
10%	8,640	635,600	307,200	92,500	4,889,600	5,993,540
9.5%	8,200	603,800	291,800	87,900	4,645,100	5,636,800
9%	7,780	572,000	276,500	83,200	4,400,700	5,340,180
8.5%	7,340	540,500	261,700	78,600	4,156,200	5,044,340
8%	6,900	508,500	245,700	74,000	3,911,700	4,746,800

Table 4.10 Power and Water Cost, 30.175 MM tpy (Sublevel Stoping with Full Subsidence)

Number of Units	Operation	H.P. per Unit	H.P. total Load	Hr. per day, full Load	KW total Load	Total KWH Requirement	Total* Water (gpd)
6	Two boom hydraulic drill jumbo	135	810	15	604	9,060	81,000
6	Two boom hydraulic fan drill	135	810	15	604	9,060	81,000
12	Rock bolting machine	150	1,800	18	1,343	24,170	65,000
4	Primary ventilation fan	1,350	5,400	24	4,030	96,680	--
2	Secondary ventilation fan	155	310	24	231	5,550	--
4	Secondary ventilation fan	45	180	24	134	3,220	--
1	Auxiliary fan	105	105	24	78	1,880	--
2	40-ton trolley locomotive	600	1,200	15	895	13,430	--
2	30-ton trolley locomotive	380	760	15	567	8,500	--
12	Water pumps	425	5,100	24	3,800	91,300	--
3	Double drum hoists, production	10,000	30,000	20	22,380	447,600	--
1	Double drum hoist, service	5,000	5,000	20	3,730	74,600	--
2	Underground crusher	700	1,400	20	1,050	21,000	18,000
	Miscellaneous underground	--	--	--	1,000	12,000	12,000
	Miscellaneous surface	--	--	--	1,000	12,000	6,000
						830,050	263,000

Electric power cost/year = \$0.013 X 830,050 X 355 = \$3,831,400

* Water assumed to be provided by site wells.

Table 4.11 Estimated Preproduction Cost
(Sublevel Stoping with Full Subsidence)

ITEM	Total Cost	Cost/Ton*
Total labor and supervision.	$8,395,000	$0.82
Operating supplies		
Drill bits.	141,000	
Rockbolts, shells, and plates	645,000	
Fuel and lubrication.	1,009,000	
ANFC, wire, and caps.	710,000	
Drill steel and couplings	150,000	
Machine parts	800,000	
Rail ties, plates, etc.	75,000	
Operating contingencies (8%).	275,000	
Interest on capitalized equipment (380 days).	3,957,000	
Subtotal	7,762,000	0.76
Power	3,700,000	0.36
Payroll overhead (35%).	2,938,000	0.29
Indirect costs (10% of labor, supervision, and operating supplies.	1,220,000	0.12
Fixed Costs		
Depreciation (380 days)	4,411,000	0.43
Taxes and insurance (3% of 52,539,00 for 380 days).	1,641,000	0.16
Total	$30,067,000	$2.94

* Total tons = 10,216,000

Table 4.12 Estimated Annual Production Cost, 30.175 MM tpy
(Sublevel Stoping with Full Subsidence)

ITEM	Annual Cost	Cost/Ton
Direct Costs		
Labor and supervision.	$7,995,000	$0.26
Operating supplies		
Machine parts.	860,000	
Lubrication and fuel	1,057,000	
Rockbolts, shells, and plates. .	159,000	
Drill bits	127,000	
ANFO, caps, and wire	1,622,000	
Drill steel.	116,000	
Ventilation tubing, bulkheads. .	500,000	
Rail, ties, plates, etc.	111,000	
Subtotal.	4,552,000	
Operating contingencies (5% of line above)	228,000	
Yearly interest cost (9%). . . .	5,340,000	
Subtotal	10,120,000	0.33
Power.	3,831,000	0.13
Reclamation.	1,000,000	0.03
Payroll overhead (35%)	2,798,000	0.09
Exploration	500,000	0.02
Environmental monitoring	300,000	0.01
Indirect Costs (10% of labor, supervision, and operating supplies, not including yearly interest cost) . .	1,278,000	0.04
Fixed Costs		
Taxes and insurance (3% of mine cost)	1,575,000	0.05
Depreciation	5,270,000	0.18
Total.	$34,668,000	$1.15

4.4.3 Sublevel Stoping Design with Spent Shale Backfill

A sublevel stoping design with spent shale backfill is being investigated because of three major advantages over a full subsidence design. These advantages are: (1) minimization or prevention of fracturing of overlying strata above the stopes, (2) minimization of surface subsidence, and (3) a significant reduction in the amount of spent shale disposal on the surface. The major disadvantage of sublevel stoping with spent shale backfill is the reduced total resource recovery due to leaving pillars.

The design limitations for this are the same as those described in the previous section. Mine access and hoisting has been discussed in a separate section, therefore, the discussions that follow will deal only with the design of the mining method. Initially, the preproduction development, rail haulage layout and stope production are described, followed by subsections on ventilation, stope filling, equipment selection, and costing. A later section will cover conclusions and recommendations of sublevel stoping with spent shale backfill.

4.4.3.1 Preproduction Development

After the hoisting and access installations are completed, development is started simultaneously on the first stoping level and on a top level from which the first level stopes are filled.

On the first stoping level, two 30 by 20-foot main haulage drifts are driven from the shaft through a 920-foot shaft pillar. Then two 30 by 20-foot crosscuts are driven to two ventilation shafts on each side of the mining area. While these crosscuts to the ventilation shafts are being driven, development of stopes adjacent to the shaft pillar is started.

A 20 by 20-foot drift (on the top fill level) is driven through the shaft pillar, then single 20 by 20-foot crosscuts are driven to the two exhaust ventilation shafts. During this development, a ramp is driven from the first stoping level to the upper fill level. This ramp is driven in the stope pillar and provides access to a sublevel which is required for the stoping operations. Table 4.13 is an approximate schedule of the preproduction to full production period. Estimated construction time for shafts, skip pockets, surge bins, and crusher rooms is 2.5 years. Total time to full production is estimated to be three to 3.5 years.

4.4.3.2 Main Haulage Layout

The same general rail haulage layout and equipment is used as is proposed for "Sublevel Stoping with Full Subsidence." However, the general mining plan is advance (mining away from the shaft pillar area), with the first stopes to come into production located next to the shaft pillar, as shown in Figure 4.12.

4.4.3.3 Open Stope Production

When full production is reached, six stopes produce 12,000± tpd for a total of 72,000± tpd. Approximately 13,000 tpd is from development, providing a total of 85,000± tpd. Figure 4.13 is an isometric of the proposed stope layout representing the overall system layout. Dimensions of rib pillars, crown pillars, and end pillars will depend heavily on the physical characteristics of the fill material, including compressibility, which are not known at the present time. The actual dimensions of the pillars should not materially influence the overall mining plan nor the direct cost per ton for mining, but it will affect the percentage of oil shale recovered from the deposit.

With pillars and stopes having the dimensions shown in Figures 4.14, 4.15, and 4.16, almost 55% of the oil shale can be recovered. This should possibly be considered an absolute top limit for stopes on the upper levels using good fill material. A low limit for percent recovery would be about 35%. This is the percentage of resource mined by the method of sublevel stoping with full subsidence, leaving the remainder of the oil shale as unrecovered pillars. In sublevel stoping with full subsidence, the pillar sizes were designed for pressures that might be

Table 4.13 Schedule of Preproduction Work for Sublevel Stoping with Spent Shale Backfill

Description of Task	Days	Ore Production (tons)
First level haulage drifts thru shaft pillar	21	84,000
30' x 20' drifts and ventilation shafts	88	364,000
Top level 20' x 20' drift thru shaft pillar	21	56,000
Top level 20' x 20' drift to ventilation shaft	88	242,000
Raise bore ventilation shafts from surface	60	
Raise bore ventilation shafts from 1000' to 1350' level	30	
Ramp system construction	36	100,000
Preparation of stopes 1 & 2	37	308,000
Preparation of stopes 3 & 4	37	308,000
Preparation of stopes 5 & 6	37	308,000
Production from stopes 1 & 2	71	990,000
Production from stopes 3 & 4	71	502,000
Production from stopes 5 & 6	71	
Total		3,262,000

Full Production

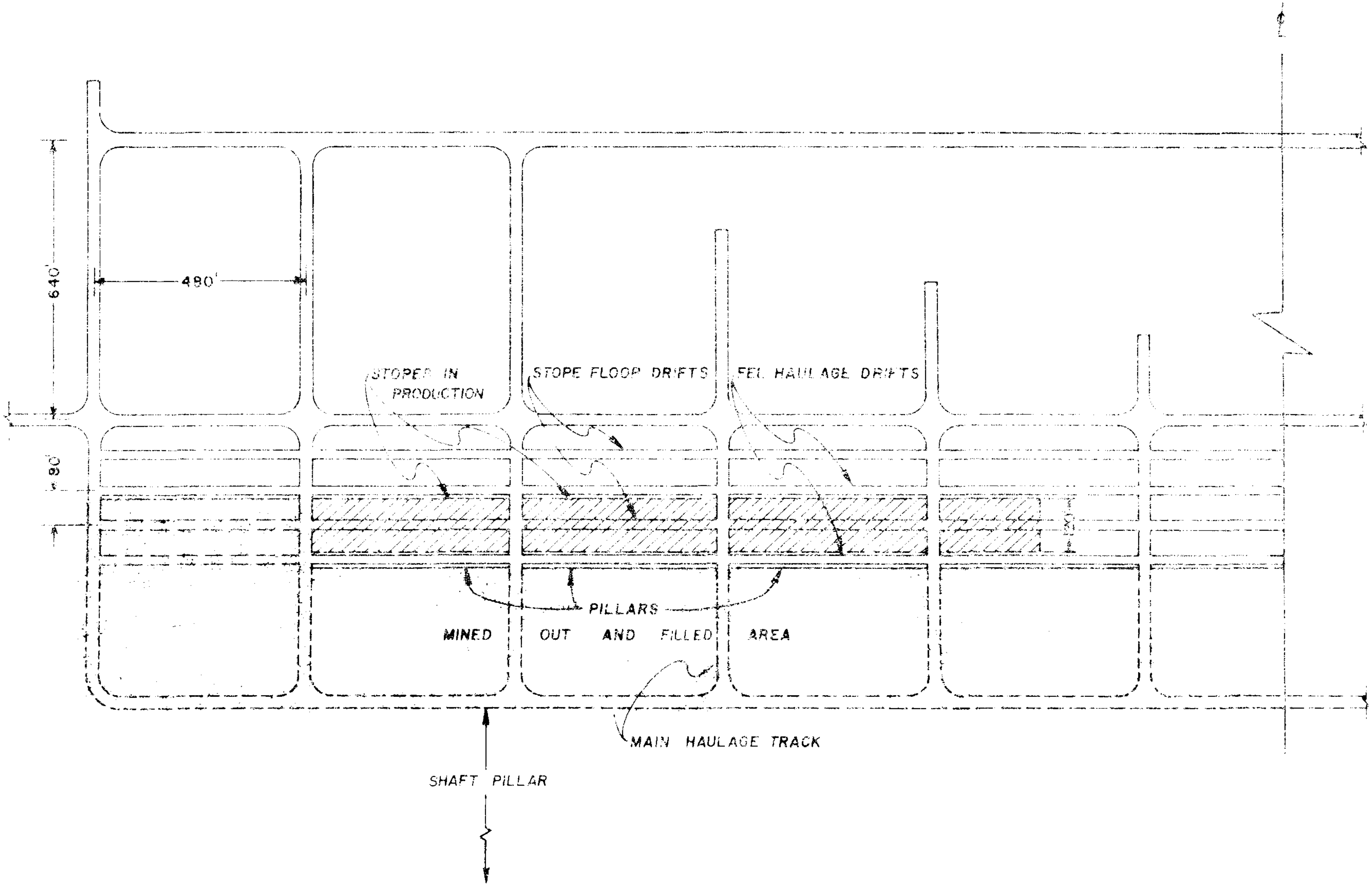


Figure 4.12 Rail Haulage Layout, Sublevel Stoping with Spent Shale Backfill

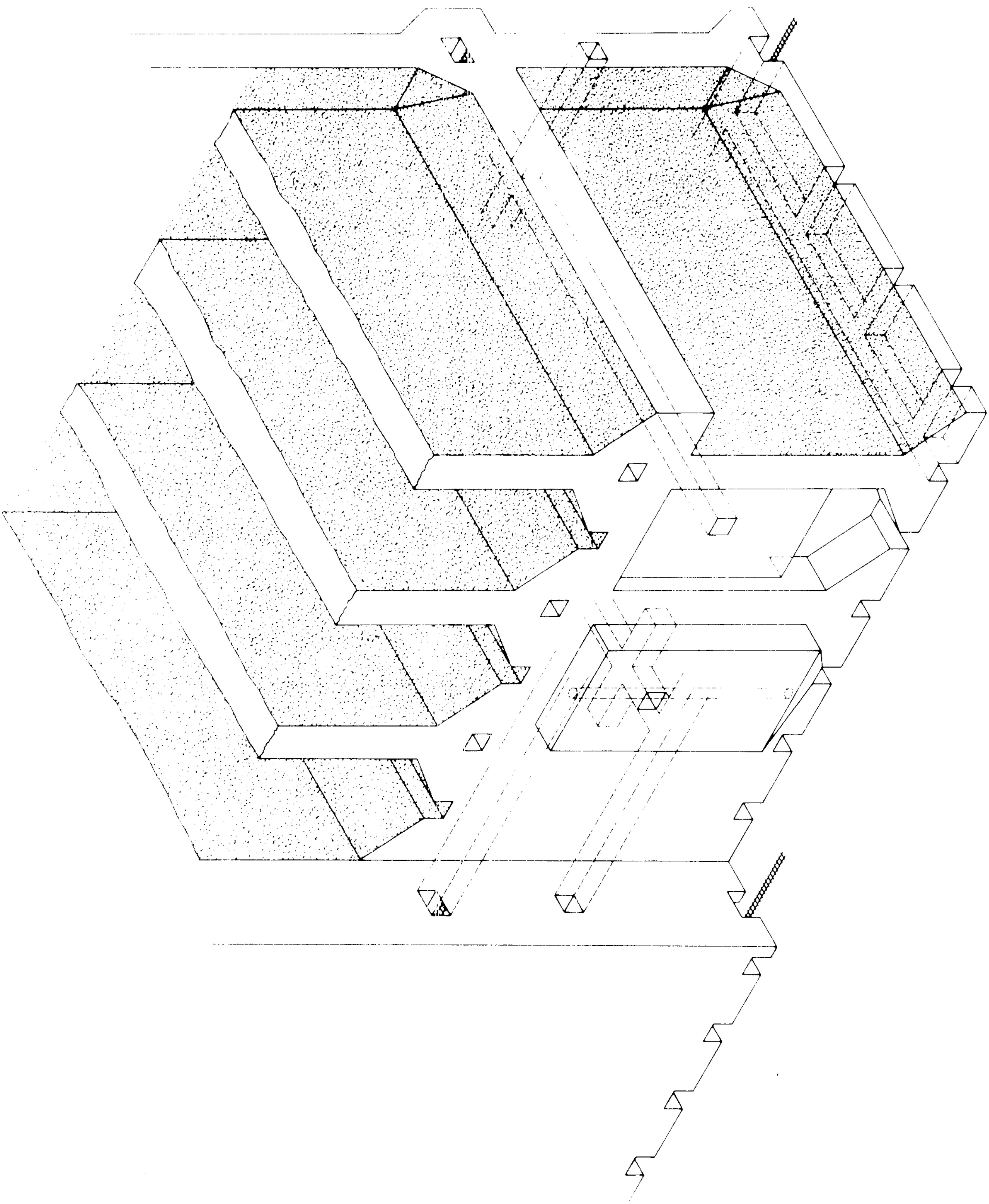


Figure 4.13 Isometric View Sublevel Stoping with Spent Shale Backfill

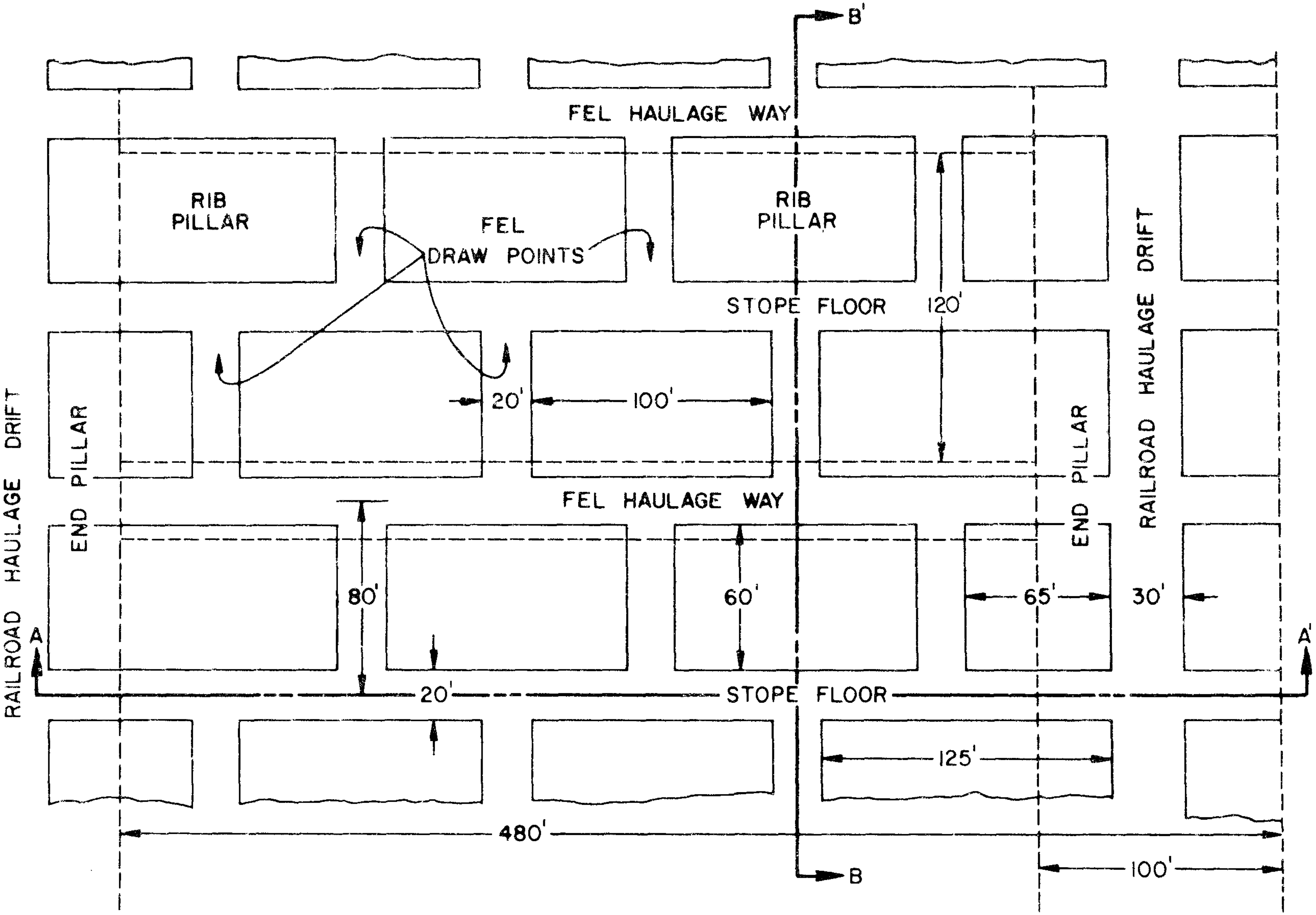


Figure 4.14 Plan View of Haulage Level, Sublevel Stoping with Spent Shale Backfill

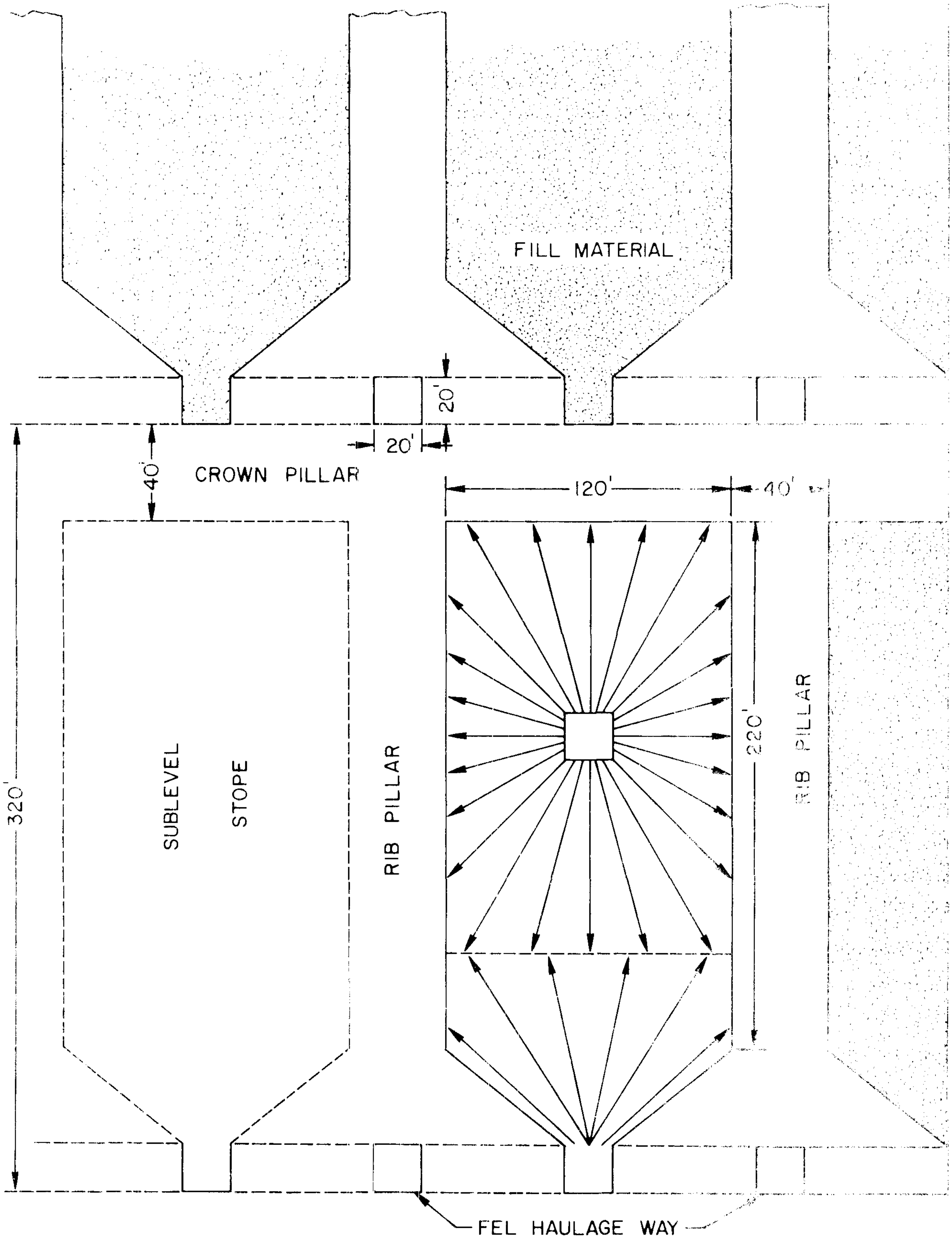


Figure 4.15 Vertical Section Through B-B', Sublevel Stoping with Spent Shale Backfill

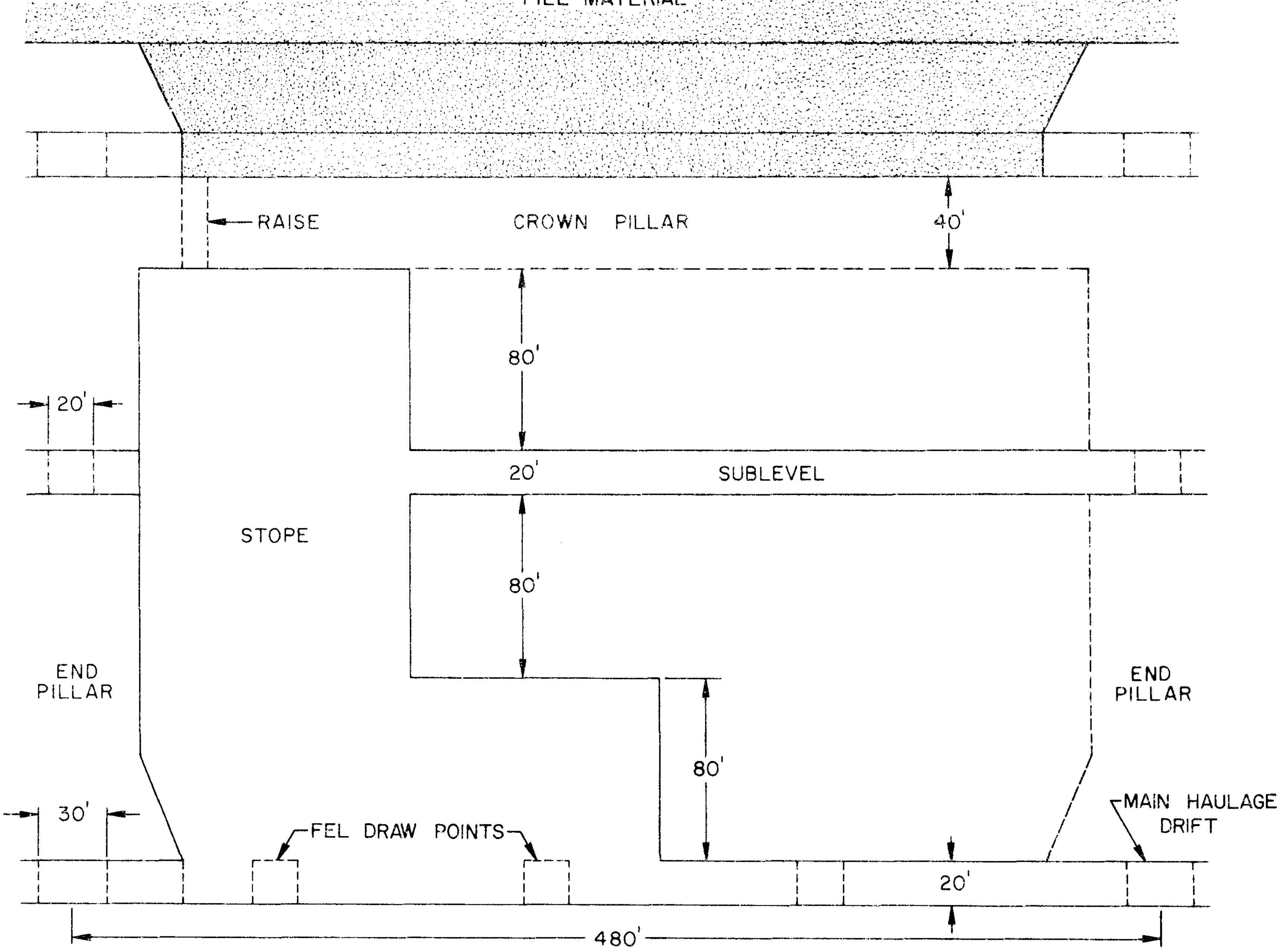


Figure 4.16 Longitudinal Section Through A-A', Sublevel Stoping with Spent Shale Backfill

expected at a depth of 3,000 feet considering the caved overburden and fill above and adjacent to the stopes.

4.4.3.4 Stope Filling

Except for the first level, stopes will be filled from the 20 by 20-foot drifts previously used as FEL haulageways for the mined out stopes above. One drift can be used for filling two lower stopes by drilling eight-inch diameter holes, inclined downward from the drift, through the crown pillar to the stope below. Three holes for each stope are sufficient. Stopes are prepared for filling by lowering slotted, six-inch PVC drainage tubes, covered with burlap or similar filtering medium, through the inclined holes to the floor of the stope. From there the pipe goes out through a crosscut to a box sump in the FEL haulageway. The number of drainage tubes required in each stope depends upon the drainage characteristics of the fill material.

Three concrete bulkheads are installed in every opening leading into the stope on the haulage level and on the sublevel. The drainage tubes go through the bulkheads and, in addition, each bulkhead has a bleed-off pipe and valve to reduce high hydrostatic pressures. Each stope requires eight bulkheads on the haulage level designed to withstand 280 feet of head of fill material and four bulkheads on the sublevel to withstand 85 feet of head.

4.4.3.5 Ventilation

The ventilation circuitry and air requirements are almost identical to those described in the previous sublevel stoping design. The primary exception is that lower initial total air volumes are needed in the first stages of production. Exhaust ventilation shafts are located immediately adjacent to the shaft pillar.

4.4.3.6 Production Cost

The method of production costing for sublevel stoping with spent shale backfill is similar to the format used by the USBM (Staff, 1972, Katell and Hemingway, 1974). This costing method necessarily assumes that the oil shale mine is the only income source to the corporation; therefore, development costs (including mine access and hoisting) are capitilized. In reality, these costs would most likely be charged to production and deducted as negative cash flow in the overall corporate cash flow determination. Excluded in this analysis are royalty payments, and surface transprotation costs. All costs were collected during first quarter, 1975.

Table 4.14 lists the supervisory and hourly personnel needed for the operation of an 85,000 tpd mine. An additional 12% of the hourly manpower requirements are included to account for absenteeism. Figure 4.17 is an example of the anticipated management flow chart for the underground mine only. Table 4.15 is the capital investment summary that includes contingencies, mobilization of capitalized equipment to mine site, consulting, environmental impact statement, mine access, and preproduction development. The total capital investment is estimated to be $96,500,000, not including the value of oil in development ore (estimated to be worth $17,100,000).

Table 4.16 is the straight-line depreciation schedule for the equipment listed in Table 4.15. Table 4.17 is the estimated yearly interest cost on the money borrowed to finance the capital equipment. In this table, interest rates from eight to ten percent are assumed; however a rate of nine percent is used in the cost summary. Notice that the mine development costs are capitalized over a period of 30 years. Power and water consumption, Table 4.18, are estimated from vendor data, and the electric rate as quoted by a local Colorado public utility for the Central Piceance Basin. Water cost is neglected because it is assumed more than enough water will be available from the mine itself. Table 4.19 is the estimated cost of preproduction, including the cost of interest on capital. The cost per ton figure is determined by dividing the total cost by the amount of preproduction development ore (3,262,000 tons).

Table 4.14 Manning Table, 30.175 MM tpy
(Sublevel Stoping with Spent Shale Backfill)

Personnel Salary	Total	Annual Cost per Employee	Annual Cost (260 Workdays)
Superintendent	1	$33,000	$33,000
Technical superintendent	1	25,000	25,000
General mine foreman	1	27,000	27,000
Mine foreman	1	22,000	22,000
Shift foreman	3	20,000	60,000
Fill foreman	1	22,000	22,000
Section foreman	8	18,000	144,000
Chief mechanical foreman	1	21,000	21,000
Shift mechanic	3	19,000	57,000
Section mechanic	6	18,000	108,000
Chief electrical foreman	1	21,000	21,000
Shift electrician	3	19,000	57,000
Chief engineer	1	25,000	25,000
Mining engineer	5	20,000	100,000
Surveyor	3	10,800	32,400
Surveyor helper	3	9,000	27,000
Draftsman	2	9,600	19,200
Mine geologist	1	18,000	18,000
Safety director	1	19,000	19,000
Safety man	3	16,500	49,500
Accountant	3	9,600	28,800
Bookkeeper	1	7,500	7,500
Purchasing agent	1	14,400	14,400
Warehouse supervisor	1	13,200	13,200
Warehouseman	3	7,200	21,600
Watchman	3	6,000	18,000
Secretary	3	7,800	23,400
Subtotal	64		$1,014,000

Table 4.14 Manning Table, 10.175 MM tpy
(Sublevel Stoping with Spent Shale Backfill) con't

Personnel	Total	Wage per Day	Annual Cost (260 Workdays)
Underground			
Drilling operator	43	$51.81	$579,200
Drilling operator helper	43	49.96	558,600
FEL operator	61	50.88	807,000
Scaling and rock bolt operator	86	52.81	1,180,800
Fill crewman	14	50.88	185,200
Powderman	58	52.81	796,400
Truck driver	22	50.88	291,000
Trackman	29	50.42	380,200
Laborer	22	50.88	291,000
Locomotive operator	14	51.81	188,600
Mechanic, first class	11	56.58	161,800
Mechanic, second class	29	52.90	398,900
Electrician, first class	4	56.58	58,800
Electrician, second class	7	52.90	96,300
Master machinist	4	56.58	58,800
Machinist	14	51.80	188,600
Welder	7	51.80	94,300
Subtotal	468		$6,315,500
Outside			
Hoistman	4	51.70	$53,800
Cage tender	4	49.77	51,800
Lampman	4	47.15	49,000
Subtotal	12		$154,600
Contingencies for absenteeism (12%)	58		$776,400
Total labor and supervision	602		$8,260,500

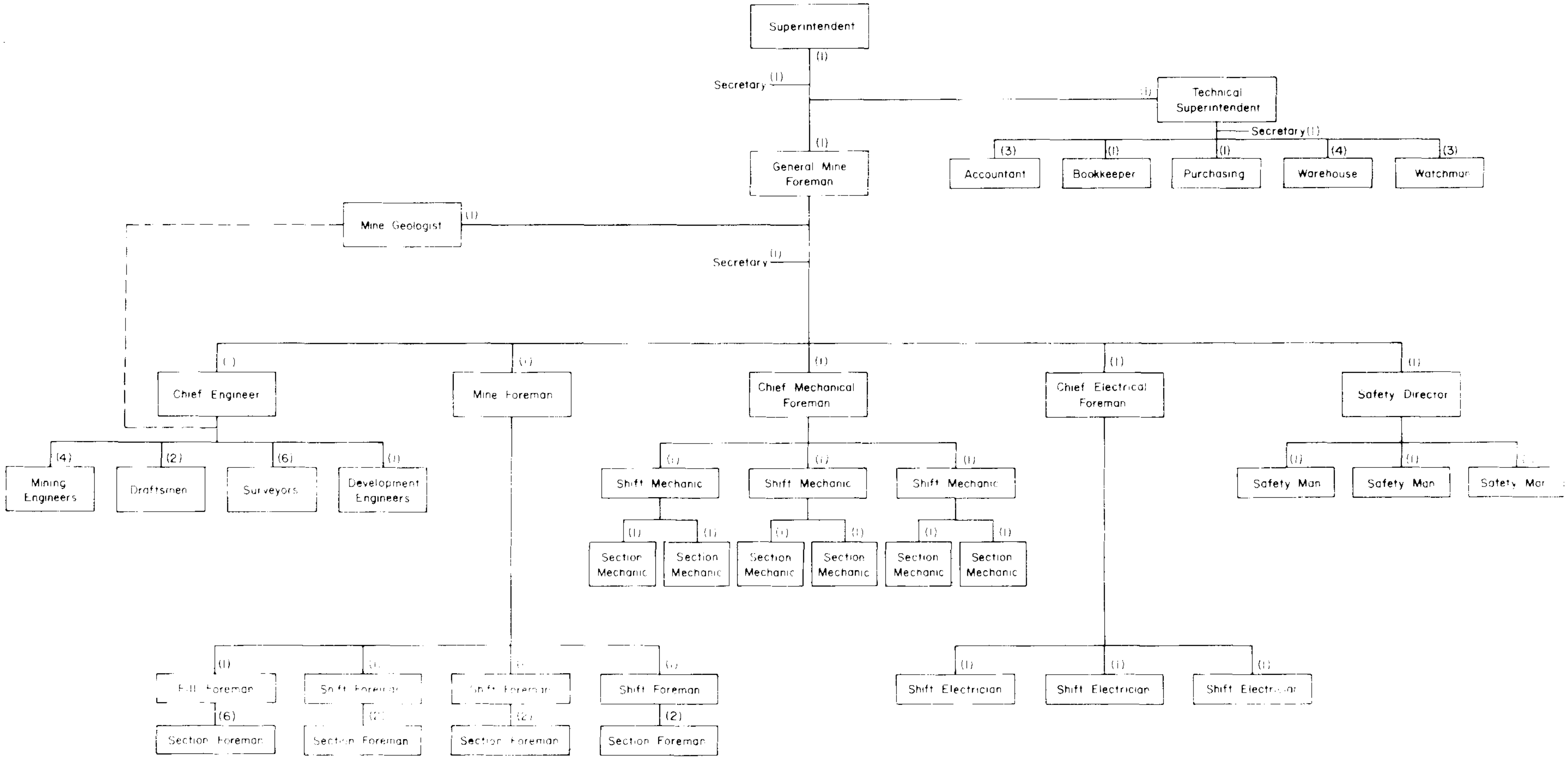


Figure 4.17 Management Flow Chart, 30.175 MM tpy, Sublevel Stoping with Spent Shale Backfill

Table 4.15 Capital Investment Summary, 30.175 MM tpy (Sublevel Stoping with Spent Shale Backfill)

ITEM	Quantity	Cost per Unit	Total Cost
Two boom hydraulic drill jumbo	8	$310,000	$2,480,000
Two boom hydraulic fan drill	7	300,000	2,100,000
40-ton trolley locomotive	3	228,500	685,500
30-ton trolley locomotive	2	189,500	379,000
FEL (12 yd^3)	24	280,000	6,720,000
80-ton rail car (air brakes)	120	23,000	2,760,000
Two drill rock bolt machine	14	150,000	2,100,000
Primary ventilation fan	4	115,000	460,000
Secondary fan (100,000 cfm)	2	8,600	17,200
Secondary fan (36,000 cfm)	4	7,500	30,000
Powder and ANFO loading truck	8	28,000	224,000
Water truck	2	30,000	60,000
Lubrication and fuel truck	4	30,000	120,000
Personnel carrier	6	41,500	249,000
Fire chemical truck	3	25,000	75,000
First aid kit	20	30	600
Lamp (including accessories)	430	85	36,600
Self rescuer	450	43	19,400
Stretcher set	9	235	2,200
Telephone (page phones)	9	450	4,100
Underground machine shop	1	1,000,000	1,000,000
Bathhouse, office, and lamphouse	1	2,000,000	2,000,000
Warehouse and supply yard	1	130,000	130,000
Substation and distribution	1	900,000	900,000
Forklift	1	25,000	25,000
Exhaust fan (machine shop)	1	8,000	8,000
Rectifier (250 KW)	15	6,120	91,800
Water pumps (10,000 gpm total)	16	33,000	528,000
Track (115-lb)	50,000 ft.	$4.8/ft.	240,000
Trolley wire	25,000 ft.	$0.68/ft.	17,000
Track plates and ties	13,000	$8.80/unit	114,400
Methane detector	20	235	4,700
Portable fire extinguishers	20	41.48	800
High voltage cable (8KV-4°)	10,000 ft.	$7.20/ft.	72,000
Track maintenance machine	1	50,000	50,000
Underground crusher	3	750,000	2,250,000
Cement mixers	3	20,000	60,000
4-inch diameter pipe	20,000 ft.	$2/ft.	40,000
8-inch diameter pipe	15,000 ft.	$4/ft.	60,000
Spent shale disposal equipment	1	500,000	500,000
Total direct cost			$26,614,300

Table 4.15 Capital Investment Summary, 30.175 MM tpy (Sublevel Stoping with Spent Shale Backfill) con't

ITEM	Total Cost
Contingencies (9% of total direct cost).	$ 2,396,000
Subtotal.	29,010,300
Mobilization charge (2% of line above)	581,000
Total construction	29,591,300
Consulting engineering fees, overhead, and administration (6% of line above).	1,776,000
Subtotal.	31,367,300
Environmental Impact Statement	2,000,000
*Mine access (included are ventilation, production, and service shafts, three double drum production and service hoists, and one double drum service hoist)	47,000,000
*Preproduction cost.	15,430,000
Net capital investment	$95,797,300
Value of oil in 3,262,000 tons of 20 gpt oil shale stockpiled or retorted during preproduction development (assumes $11/bbl).	$17,087,000

* Assumes capitalization

Table 4.16 Depreciation Schedule, 30.175 MM tpy
(Sublevel Stoping with Spent Shale Backfill)

ITEM	Straight-Line Depreciation (years)	Yearly Charge
Two boom hydraulic drill jumbo	8	$ 310,000
Two boom hydraulic fan drill	8	262,000
[illegible]-ton trolley locomotive	15	45,700
[illegible]-ton trolley locomotive	15	25,300
[illegible] (12 yd³)	8	240,000
[illegible]-ton rail car (air brakes)	10	[illegible]
[illegible] drill rock bolt machine	10	[illegible]
Primary ventilation fans	[illegible]	[illegible]
Auxiliary ventilation fans	10	5,[illegible]
[illegible] and AMFO loading trucks	10	[illegible]
[illegible] truck	10	[illegible]
Lubrication and fuel truck	10	12,[illegible]
Personnel carrier	8	40,[illegible]
[illegible] chemical truck	8	[illegible],400
Mine safety equipment	5	13,700
Underground machine shop	30	[illegible]
Bathhouse, office, and lamphouse	30	[illegible],700
Warehouse and supply yard	30	4,400
Substation and distribution	30	[illegible],000
Forklift	10	2,500
Rectifier (250 KW)	15	6,[illegible]00
Water pumps (10,000 gpm total)	15	35,200
Track and trolley material	30	14,800
Track maintenance machine	15	3,400
Underground crusher	30	75,000
Cement mixers	10	6,000
Pipe	30	3,400
Spent shale disposal equipment	10	50,000
Shaft access and ventilation	30	[illegible],567,000
Preproduction development	30	514,000
Environmental impact statement	30	56,700
Contingencies, mobilization, and engineering	30	158,400
Total.		$4,740,600

Table 4.17 Estimated Yearly Interest Cost on Capitalized Items, 30.175 MM tpy (Sublevel Stoping with Spent Shale Backfill)

Depreciable Life (yrs)	5	8	10	15	30	Total
Capital Required	317,500	11,375,200	6,004,000	1,735,500	76,394,400	95,826,000
Interest Charge 10%	19,000	639,900	330,200	92,600	3,978,900	5,060,600
9.5%	18,100	607,900	313,700	87,900	3,779,900	4,807,500
9%	17,100	575,900	297,200	83,300	3,581,000	4,554,500
8.5%	16,200	543,900	280,700	78,700	3,382,000	4,301,500
8%	15,200	511,900	264,200	74,000	3,183,100	4,048.

Table 4.1[illegible] Power and Water Cost, 30.175 MM tpy (Sublevel Stoping with Spent Shale Backfill)

Number of Units	Operation	H.P. per unit	H.P. total load	Hr/Day full load	KW total load	Total KWH requirement	Total* Water (gpd)
6	Two boom hydraulic drill jumbo	135	810	15	604	9,060	81,000
6	Two boom hydraulic fan drill	135	810	15	604	9,060	81,000
12	Rock bolting machine	150	1,800	15	1,343	24,170	65,000
4	Primary ventilation fan	1,350	5,400	24	4,030	96,680	--
2	Secondary ventilation fan	155	310	24	231	5,550	--
4	Secondary ventilation fan	45	180	24	134	3,220	--
1	Auxillary fan	105	105	24	78	1,880	--
2	40-ton trolley locomotive	600	1,200	15	895	13,430	--
2	30-ton trolley locomotive	380	760	15	567	8,500	--
12	Water pumps	425	5,100	24	3,800	91,300	--
3	Double drum hoist, production	10,000	30,000	20	22,380	447,600	--
1	Double drum hoist, service	5,000	5,000	20	3,730	74,600	--
2	Underground crusher	700	1,400	20	1,050	21,000	18,000
	Miscellaneous underground	--	--	--	1,000	12,000	12,000
	Miscellaneous surface	--	--	--	1,000	12,000	6,000
4	Spent shale pumps	150	600	20	448	8,950	360,000
					Total	839,000	623,000

Electric power cost/year = \$0.013 X 839,000 X 355 = \$3,872,000

* Water assumed to be provided by site wells.

Table 4.19 Estimated Preproduction Cost (Sublevel Stoping with Spent Shale Backfill)

ITEM	Total Cost	Cost/Ton*
Total labor and supervision.	$3,729,000	$1.14
Operating supplies		
Drill bits	51,000	
Rock bolts, shells, plates	187,000	
Fuel and lubrication	515,000	
ANFO, wire, and caps	228,000	
Drill steel and couplings.	49,000	
Machine parts.	440,000	
Operating contingencies (8%) . . .	118,000	
Interest on capitalized equipment (240 days)	2,463,000	
Subtotal	4,051,000	1.24
Power.	2,013,000	0.62
Payroll overhead (35%)	1,305,000	0.40
Indirect costs (10% of labor, supervision, and operating supplies).	532,000	0.16
Fixed costs		
Depreciation (240 days).	2,768,000	0.85
Taxes and insurance (3% of 52,333,000 for 240 days) . . .	1,032,000	0.32
Total	$15,430,000	$4.73

* Total tons = 3,262,000

Table 4.20 is a summary of the estimated annual production cost for an 85,000 tpd mining operation. It must be understood that this cost includes only mining costs and does not include processing costs. Also included are yearly interest, reclamation, exploration, and environmental monitoring costs. The total estimated cost per ton is $1.12.

Table 4.20 Estimated Annual Production Cost, 30.175 MM tpy (Sublevel Stoping with Spent Shale Backfill)

ITEM	Annual Cost	Cost/Ton
Direct Cost		
Labor and supervision.	$8,261,000	$0.27
Operating supplies		
Machine parts.	870,000	
Lubrication and fuel	1,057,000	
Rock bolts , shells, and plates.	159,000	
Drill bits	127,000	
ANFO, caps, and wire	1,622,000	
Drill steel.	116,000	
Ventilation tubing, bulkheads. .	500,000	
Cement, aggregate, lumber. . . .	200,000	
Rail, ties, plates, etc.	111,000	
Subtotal	4,762,000	
Operating contingencies(5% of line above).	238,000	
Yearly interest cost	4,555,000	
Subtotal.	$9,555,000	0.32
Power	3,872,000	0.13
Reclamation.	1,000,000	0.03
Payroll overhead (35%)	2,891,000	0.10
Exploration.	500,000	0.02
Environmental monitoring	300,000	0.01
Indirect costs (10% of labor, supervision, and operating supplies, not including yearly interest cost).	1,326,000	0.04
Fixed costs		
Taxes and insurance (3% of mine cost, $52,333,000).	1,570,000	0.05
Depreciation	4,759,000	0.15
Total	$34,034,000	$1.12

4.5 DESIGN AND ANALYSIS OF ROOM AND PILLAR MINING IN OIL SHALE

Room and pillar mining is being evaluated in this contract study because of its high degree of flexibility in conforming to changing mining conditions and equipment modifications. The stratiform character and inherent strength of oil shale also make room and pillar mining attractive. To date, all underground mining of oil shale has been done using room and pillar methods.

Although room and pillar methods are highly flexible, they do have some disadvantages when compared to other methods. The cost of room and pillar mining is generally higher because of the larger number of headings needed for production. Ventilation and roof control become significant factors because large areas are open for a much longer period.

Two room and pillar variations for mining oil shale are investigated: (1) advance entry and pillar and (2) chamber and pillar. These two systems are quite similar to room and pillar mining in that large underground openings are excavated and intermediate pillars left for support. Systems of mining on the advance and retreat were investigated for both methods. Preliminary results showed that retreat mining is not a favorable approach, and consequently, all designs are for advance mining (mining away from the shaft area). The mine dimensions used in the analyses are approximate and are for production costing only.

The following sections contain a review of the pertinent literature on room and pillar mining and the constraints and limitations on the proposed designs for pillar mining in oil shale. The two room and pillar variations are then presented in separate sections that include designs from preproduction through costing.

4.5.1 Review of Literature

In the 1973 edition of the SME Mining Engineering Handbook, Given (1973) describes the basic approach to room and pillar mining systems.

Some of the major advantages of room and pillar mining mentioned are its flexibility and mining selectivity. Some major disadvantages are: (1) higher costs for ventilation and roof support, (2) a large number of production areas are needed, (3) supplies and power must be delivered to more working areas, and (4) supervision is more difficult.

East and Gardner (1964) present an in-depth study of room and pillar mining in oil shale in USBM Bulletin 611. The room and pillar method of mining was selected after extensive physical testing of oil shale had been completed. Their design takes advantage of the strong and competent character of the oil shale. In the design of the mine, 60-foot square by 60-foot high pillars were used under an overburden varying from 300 to 700 feet. Results of the pilot mine study showed an average production of 148 tons per man-shift for direct underground labor, and 116 tons per man-shift for total labor. These figures indicate that the cost on a commerical basis at that time for mining oil shale at the Rifle mine (28 gpt) was less than $1.00 per barrel of oil recovered.

Dravo Corporation (1974) published a report that includes a review of room and pillar mining and describes several systems now in use. As the most common underground mining method in the United States, non-coal room and pillar mining accounts for over 75% of all the mines producing over 1200 tpd. Rooms from five to 100 feet high have been mined. The deepest room and pillar mines in North America are about 3200 feet below the surface. Extraction rates vary from 35% at depths below 3000 feet to over 90% at shallow depths (if the pillars are recovered).

A USBM report (Staff, 1969) presents a simplified design for the mining of a 60-foot thick bed of oil shale, containing nacholite and dawsonite, by a modified room and pillar system within a panel. The report assumes that the orebody is at a depth of 2,530 feet and the mining is limited to a 45% overall extraction. The report is primarily an economic evaluation of deep mining of oil shale.

In 1964 a number of petroleum companies, realizing the potential of oil shale as a possible source of synthetic fuels, invested time and money in the Anvil Points Oil Shale Research Program at Rifle, Colorado (Zambas et al, 1972). The purpose of this program was to further develop the gas combustion retort and to operate an experimental mine. The mine design consisted of a room and pillar method for mining the 78-foot thick Mahogany Zone. An estimated 80% extraction within the mining zone was obtained. The headings incorporated 60-foot wide rooms, a 40-foot full face heading, and a 38-foot vertical bench. Pillars 40 feet square were left that withstood blasting damage and safely maintained the roof.

Katelle and Wellman (1974) reviews two options for syncrude production of 50,000 and 100,000 barrels per calendar day with an integrated system of underground mining, above ground processing, and waste handling. The underground mining system utilizes a panel and pillar system with panel headings advanced by a 33-foot high heading round and a 28-foot high bench round. The headings are designed 60 feet wide with 60-foot regularly spaced square pillars left for support.

4.5.2 Advance Entry and Pillar Mine Design

Advance entry and pillar mining is a method in which multiple headings on evenly spaced centers are advanced (mining away from the shaft area) in one direction. Crosscuts are driven perpendicular to the headings resulting in square pillars left to separate headings and crosscuts. This method is similar to room and pillar mining except that entries or headings become modified rooms and are advanced until a level has reached its economic limits.

The entry and pillar system is multi-level with each level separated by sill pillars, Figure 4.18. Vertical shafts are used for access, hoisting, and ventilation. Overall resource recovery, assuming a 40-foot sill pillar, is approximately 45% with pillars left, or 55 to 60% with partial pillar recovery.

Once the shafts are completed, preproduction development begins. Pillar recovery is started when a level has been mined out to its economic limits. During pillar recovery on an upper level, the next

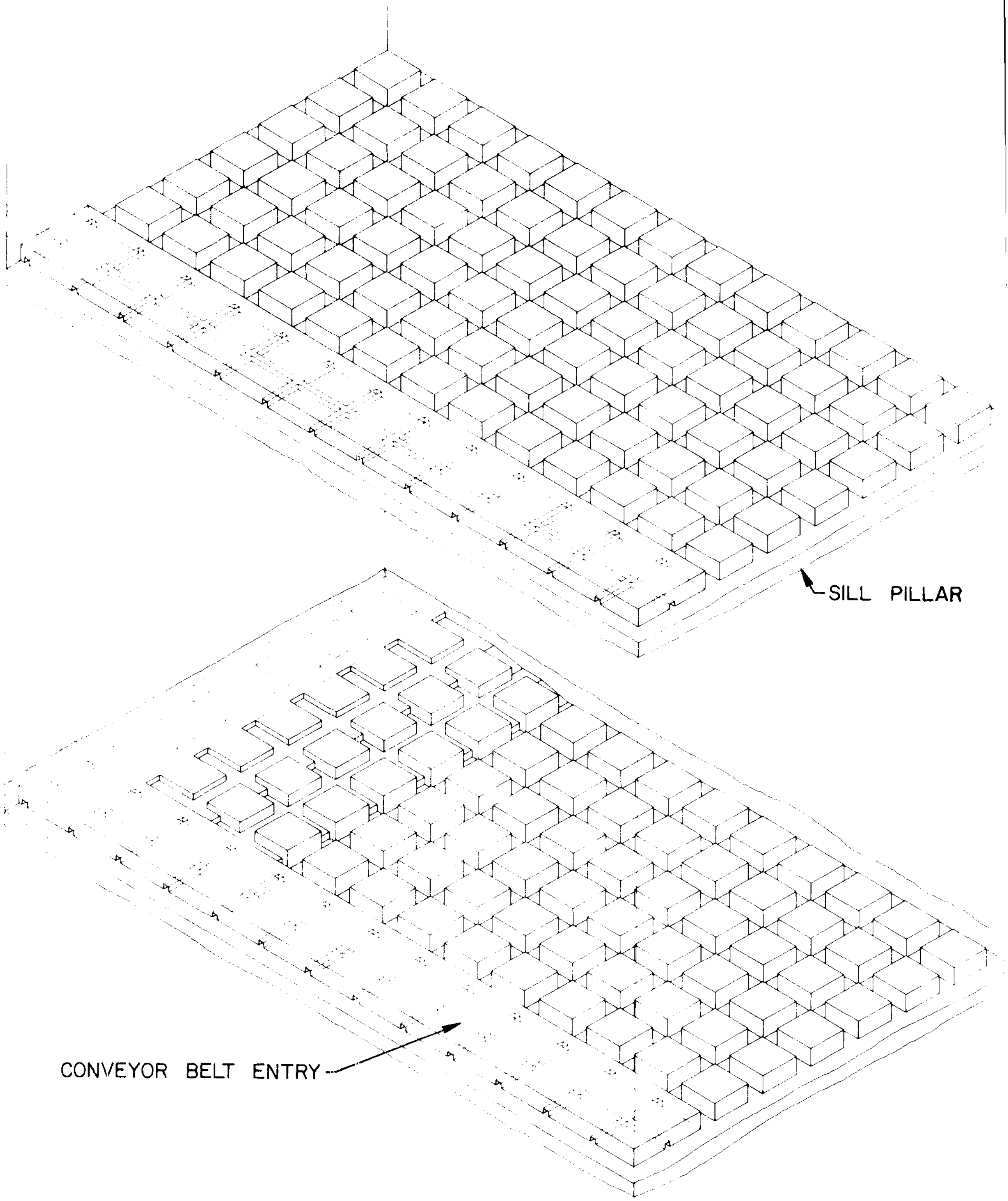


Figure 4.18 Isometric View, Advance Entry and Pillar Mining

level begins production. Spent shale can be returned to the mining level during the production phase and used to fill mined out crosscuts.

Pillar size and roof bolt length were determined using the principles presented in Section 3. Equipment selection, ventilation requirements, and heading round designs are presented in Appendix B.

4.5.2.1 Preproduction Development

A production level consists of four production sections divided by a 30 by 30-foot conveyor belt entry, Figure 4.19. Initial development begins by driving five, 30 by 30-foot entries a distance of 900 feet to the boundary of the shaft pillar. The middle entry is used for belt haulage and the two adjacent entries used for intake air and access for men and supplies. The two outer entries are used for return air and are located about 1490 feet from the intake entries. Once the return air entries are completed, a 90-foot wide by 100-foot high crosscut entry is driven about 1840 feet east to west. Seven 90 by 100-foot entries, on 307-foot centers, are then started and are the source of the production ore. Crosscuts 90 feet wide on 307-foot centers are driven east to west providing access to production entries and aiding in ventilation. Once all entries have been started, full production is achieved with approximately 85,000 tpd. Table 4.21 is a schedule of the estimated time to complete preproduction development. It is estimated that it will take 92 days to reach full production, assuming 180 feet per day advance in the 30 by 30-foot entries and 60 feet per day advance in the 90 by 100-foot crosscut entries. Total time to production, including the mine access and ventilation shafts, is estimated at 2.5 to three years.

4.5.2.2 Production

The mining operation is highly mechanized with equipment and men assigned to specific phases of the operation, such as drilling, blasting, loading, hauling, scaling, or rock bolting. As a unit completes its specific job in one heading it moves on to another and performs the same job again. Sufficient headings are available so all phases run

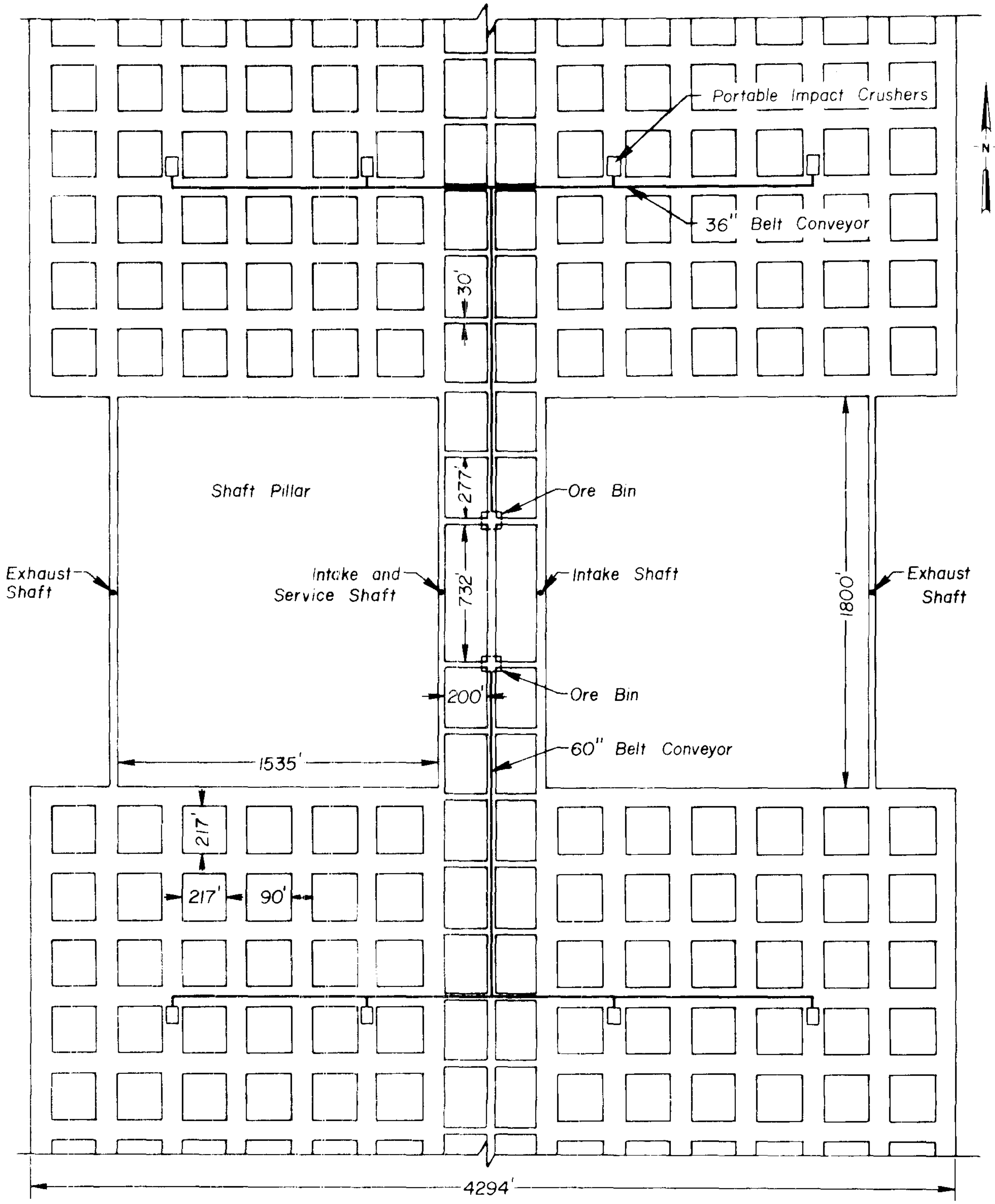


Figure 4.19 Plan View of Production Layout, Advance Entry and Pillar Mining

Table 4.21 Schedule of Preproduction Work for Advance Entry and Pillar Mining

Description of Task		Days (Preproduction)	Ore Production (tons)
Development of Belt Entry and Crosscuts		19	208,000
Return Entries		15	125,000
			125,000
Intake Entries		15	125,000
			125,000
Development of Crosscut Entry East and West Working From Both Ends	East	72	1,340,000
			1,340,000
	West		1,340,000
			1,340,000
		Start Full Production 92^{nd} Day → Production	
		Total	6,068,000

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continually without interruption. Approximately 40% of the daily production is provided from full-face headings with the remaining 60% coming from benching operations. Development of a full-face heading 400 feet ahead of the benches is necessary to provide production working areas in advance.

Mining in the crosscuts and entries is done by advancing a 40 by 90-foot full-face heading ahead of two 30-foot vertical benches, Figure 4.20. Ramps provide access to equipment on the upper face and benches. The pillars are 217 feet square on an evenly spaced pattern with the entries and crosscuts 90 feet wide. Sill pillars between levels are prebolted from above depending upon pillar thickness.

A crosscut or entry is driven by first advancing a 90-foot wide by 40-foot high heading. The heading is driven using a four-boom hydraulic drilling jumbo capable of drilling the entire round in one setup. While the round is being drilled, the roof is rock bolted using aerial roof bolting machines placing bolts on five-foot centers. Upon completion of roof bolting, the round is charged with truck mounted, pneumatic ANFO loading machine carrying about 3000 pounds of ANFO. The round is mucked using 12 yd^3 LHD's dumping into 1000 tph capacity portable impact crushers. The crushers feed minus eight-inch material onto 36-inch section belts that converge to a single 60-inch main line conveyor belt. The remaining 60 feet is mined in two 30-foot benches, the topmost approximately 400 feet behind the entry headings. The lowermost is maintained approximately 400 feet behind the top bench. The benches are drilled using two boom, vertical hydraulic-electric drilling jumbos. The vertical holes are charged using the same equipment used for the heading rounds. Blasted muck from the top bench is pushed with bulldozers to the bottom of the second bench where it is mucked, along with the ore blasted from the second bench.

A list of all the production and miscellaneous equipment is presented in a cost analysis section that follows. The capital equipment used for preproduction development and production mining is, for the most part, currently being produced or in the development stage. A

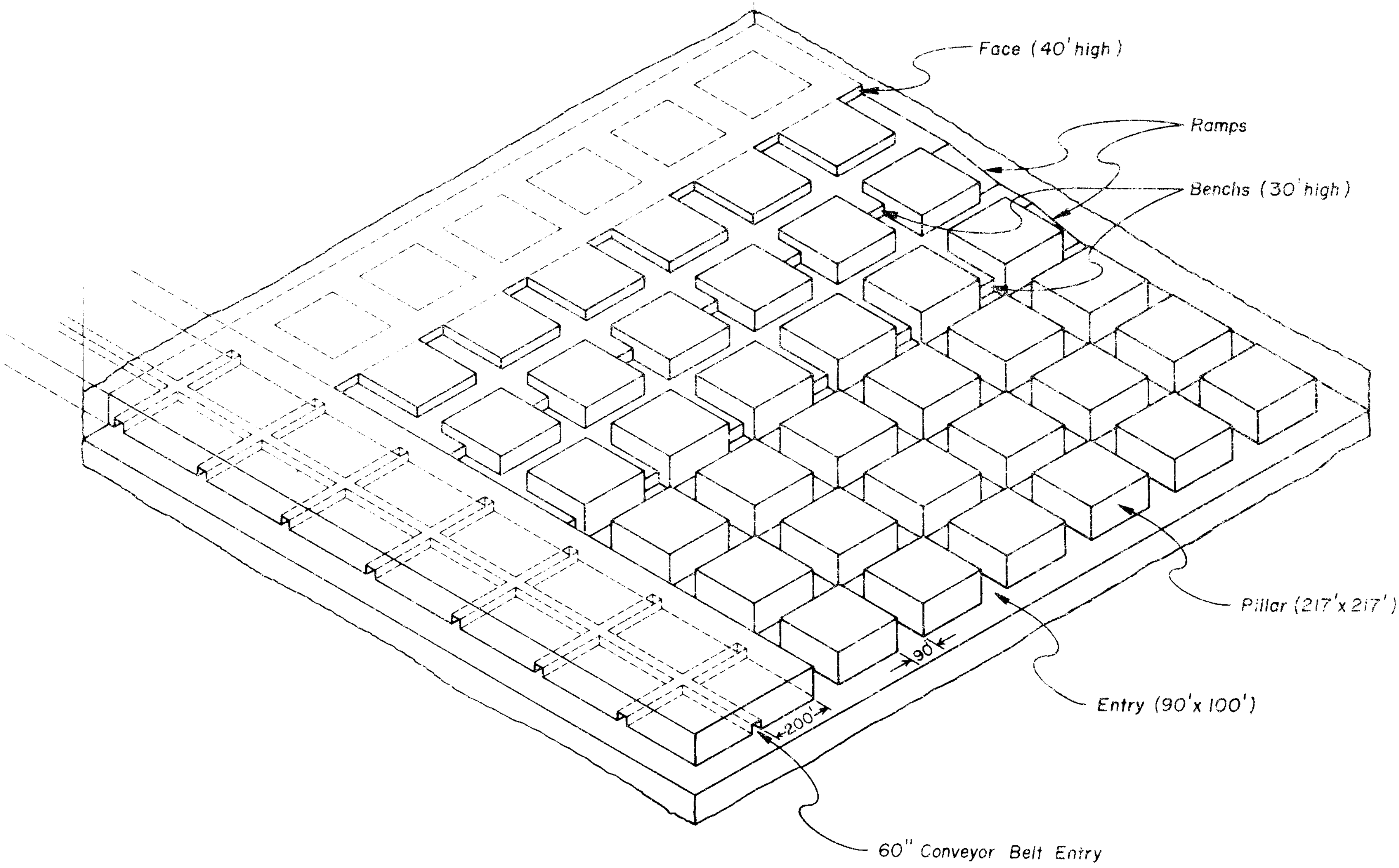


Figure 4.20 Isometric View Showing Full Face and Bench Mining System, Advance Entry and Pillar Mining

later section describes the calculations used to determine the size and quantity of equipment used in this design.

4.5.2.3 Ventilation

The following discussion is based on rough estimates assuming gassy mining conditions and approximating the diesel horsepower output of mechanized equipment.

Primary ventilation is provided by exhaust fans mounted near the return shafts, Figure 4.21. Air is pulled down the service and intake air shafts, circulated through the mine, and exhausted through two exhaust shafts. Colorado mining laws (Colorado Bureau of Mines, 1971 and State of Colorado, 1966) require 75 cfm of free air for each brake horsepower of diesel equipment and 100 cfm for each man underground. The calculations of the estimated air requirements are presented in Appendix B. A total of 3,500,000 cfm of air is exhausted by three large axial vane fans.

Air flow direction is controlled by fabricated regulators and bulkheads. Ventilation to the development headings is provided by 30,000 cfm auxiliary fans and ventilation tubing. Air flow control within a section is shown in Figure 4.22. Each section has 481,000 cfm of regulated air having a minimum velocity of 120 feet per minute or 1.36 miles per hour. A 45% leakage of fresh air is assumed. All fans used underground are permissible.

Dust created by mining and haulage operations is partially controlled using the following procedures:

1. Wet drilling is employed on upper and lower bench drilling jumbos,
2. Muck piles are wetted several times before and during mucking operations,
3. Haulage ways are watered to suppress traffic generated dust.

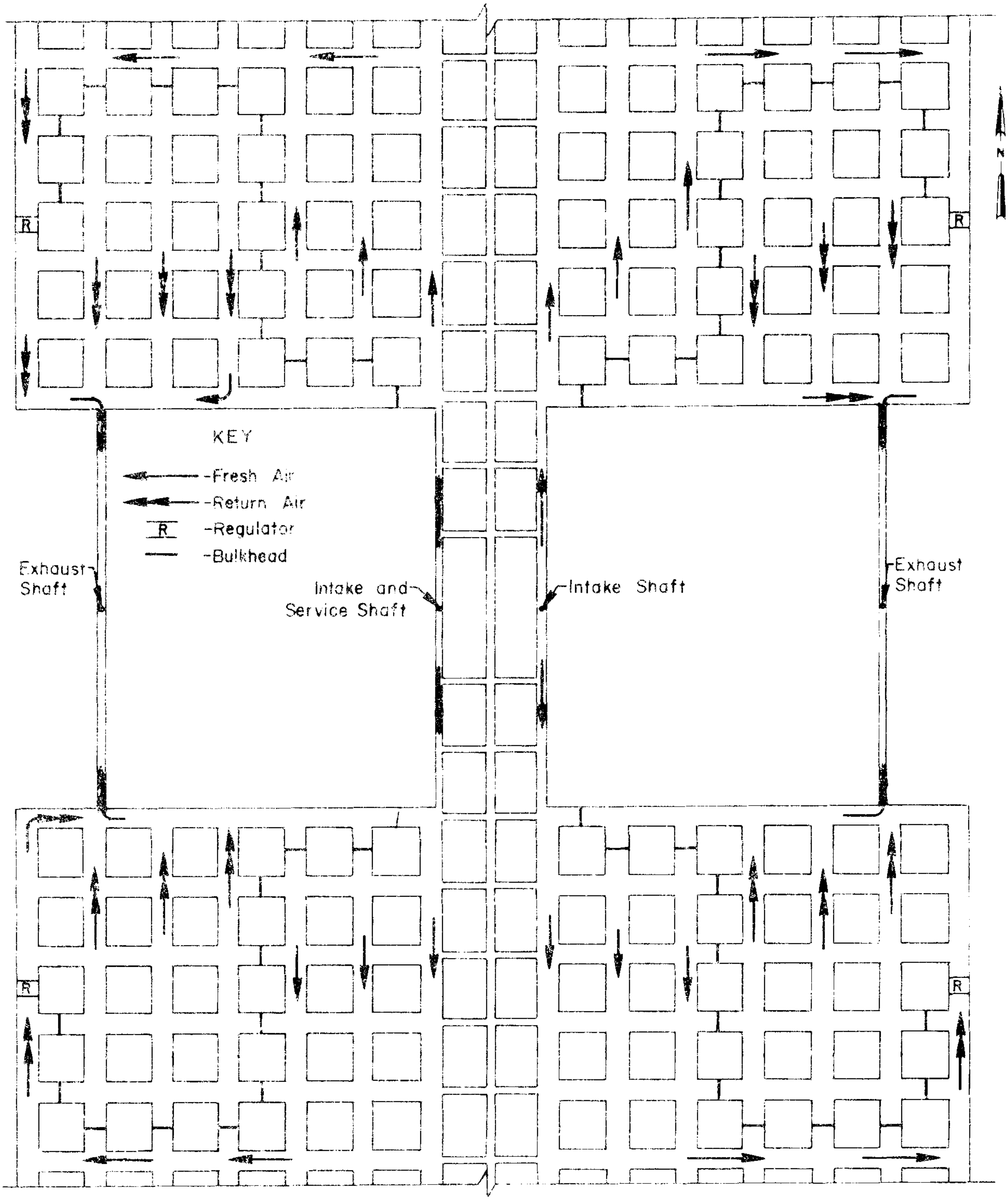


Figure 4.21 Plan View of Ventilation Network, Advance Entry and Pillar Mining

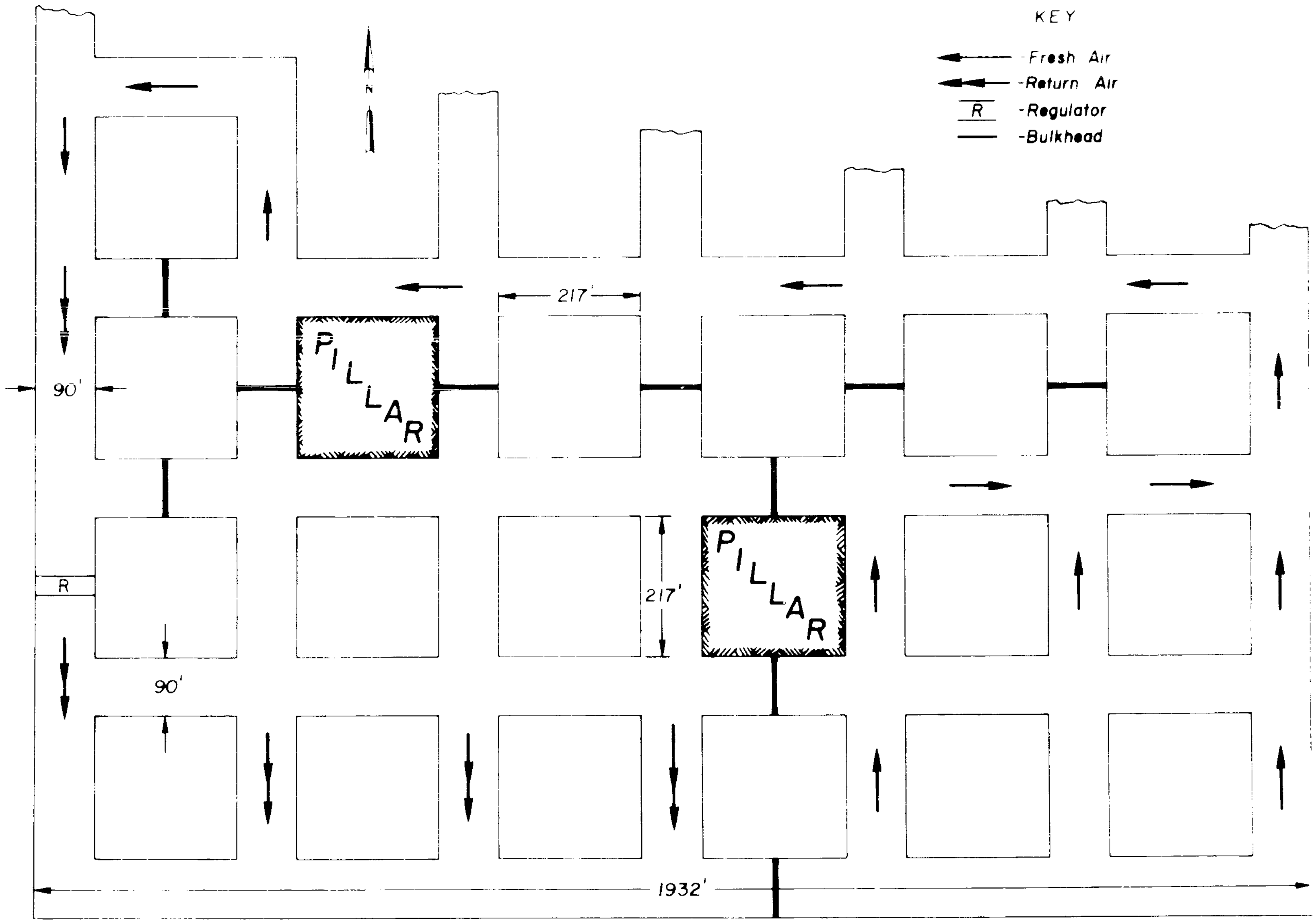


Figure 4.22 Plan View of Section Ventilation Network, Advance Entry and Pillar Mining

4.5.2.4 Production Equipment Selection

Appendix B contains the calculations used to estimate size, quantities, and cycle times for major equipment items. The capital expenditure tables in a following section on cost analysis summarize the equipment selections.

Evaluation of capital equipment for this large tonnage operation has been done by relying almost exclusively on current vendor data. Wherever possible, only that equipment currently manufactured, whether as a full production item or as a prototype, was selected for analysis.

For this initial evaluation operator efficiencies and minor equipment set up times are approximated by considering five hours of working time per eight-hour shift. The mine has been designed on a three-shift, seven-days per week work schedule. For the purpose of including holidays, week-ends and time off, a work-year is considered to be 355 days. Equipment availabilities are estimated and range from 65 to 80%, depending on use and past performance of similar models.

4.5.2.5 Pillar Recovery and Spent Shale Disposal

Once a section or level has been mined, it is possible to begin pillar recovery. Pillar slabbing starts at the outermost limits of the mine, retreating towards the shaft area. When slabbing has proceeded far enough, the remaining open areas are filled with spent shale.

The disposal of spent shale only after a section has been mined out is necessitated due to ventilation and bulkhead construction problems. A more detailed evaluation and design of a spent shale disposal system for pillar and entry mining is difficult because of the lack of information on spent shale properties. A knowledge of the specific retorting process is needed before more work can be done.

Several systems of getting the spent shale underground must be tried before an optimal system is selected. One such system is to drill boreholes from the surface into the mined out zone and pump either a slurry or air entrained fill into the excavated area. Another system involves pumping a slurry through a pipeline from the surface, through the shaft or access area, and into the mined out areas. It is estimated that once spent shale disposal starts, fill will be put back underground at a rate of about 20,000 tons per shift.

4.5.2.6 Production Cost

The method of production costing for advance entry and pillar mining is similar to the format used by the USBM (Staff, 1972 and Katell and Hemingway, 1974). This costing method necessarily assumes that the oil shale mine is the only income source to the corporation; therefore, development costs (including mine access and hoisting) are capitalized. In reality, these costs would most likely be charged to production and deducted as negative cash flow in the overall corporate cash flow determination. Excluded in this analysis are royalty payments, welfare payments, and surface transportation costs. All costs were collected during first quarter, 1975.

Table 4.22 lists the supervisory and hourly personnel needed for the operation of an 85,000 tpd mine. An additional 12% of the hourly manpower requirements is included to account for absenteeism. Figure 4.23 is an example of the anticipated management flow chart for the underground mine only. Table 4.23 is the capital investment summary that includes contingencies, mobilization of capitalized equipment to mine site, consulting, environmental impact statement, mine access, and preproduction development. The total capital investment is estimated at $84,200,000, not including the value of oil in the development ore (estimated to be worth $31,500,000).

Table 4.24 is the straight-line depreciation schedule for the equipment listed in Table 4.23. Table 4.25 is the estimated yearly interest cost on the money borrowed to finance the capital equipment. In this table, interest rates from eight to ten percent were assumed; however a rate of nine percent was used in the cost summary. Notice that the mine development costs are capitalized over a period of 30 years. Power and water consumption, Table 4.26 are estimated from vendor data, with the electric rate as

Table 4.22 Manning Table, 30.175 MM tpy
(Advance Entry and Pillar Mining)

Personnel	Total	Annual Cost per Employee	Annual Cost (260 Workdays)
Salary			
Superintendent	1	$33,000	$33,000
Technical superintendent	1	25,000	25,000
General mine foreman	1	27,000	27,000
Mine foreman	1	22,000	22,000
Shift foreman	3	20,000	60,000
Assistant shift foreman	3	19,000	57,000
Drilling foreman } section	12	18,000	216,000
Production foreman } section	12	18,000	216,000
Chief mechanical foreman	1	21,000	21,000
Shift mechanic foreman	3	19,000	57,000
Chief electrical foreman	1	21,000	21,000
Shift electrical foreman	3	19,000	57,000
Section mechanic foreman	6	18,000	108,000
Chief mine engineer	1	25,000	25,000
Mining engineer	5	20,000	100,000
Surveyor	3	10,800	32,400
Surveyor helper	3	9,000	27,000
Draftsman	2	9,600	19,200
Safety director	1	19,000	19,000
Safety foreman	3	16,500	49,500
Business manager	1	18,000	18,000
Accountant	1	9,600	9,600
Stenographer	2	6,600	13,200
Payroll agent	2	7,500	15,000
Purchasing supervisor	1	14,400	14,400
Warehouse supervisor	1	13,200	13,200
Warehouseman	3	7,200	21,600
Mine geologist	1	18,000	18,000
Watchman	3	6,000	18,000
Executive secretary	1	9,000	9,000
Secretary	1	7,800	7,800
Subtotal	83		$1,349,900

Table 4.22 Manning Table, 30.175 MM tpy
(Advance Entry and Pillar Mining) con't

Personnel	Total	Wages per Day	Annual Cost (260 Workdays)
Underground			
Drilling operator	29	$52.81	$ 398,200
Drilling operator helper	29	49.96	376,700
LHD operator	100	50.88	1,322,900
Bulldozer operator	29	50.88	383,600
Truck drivers	43	50.88	568,800
Powderman	58	52.81	796,400
Scaling & rock bolter	29	52.81	398,200
Crusherman	14	50.23	182,800
Laborer and slurryman	29	50.23	378,700
Carpenter	19	50.88	251,300
Conveyor beltman	22	50.23	287,300
Mechanic, first class	43	56.58	632,600
Mechanic, second class	43	52.90	591,400
Electrician, first class	14	56.58	206,000
Electrician, second class	14	52.90	192,600
Machinist	14	51.80	188,600
Welder	7	51.80	94,300
Master machinist	4	56.58	58,800
Subtotal	540		$7,309,200
Outside			
Hoistman	4	$51.70	$53,800
Cage tender	4	49.77	51,800
Lampman	4	47.15	49,000
Subtotal	12		$154,600
Contingencies for absenteeism (12%)	66		$895,700
Total labor and supervision	701		$9,709,400

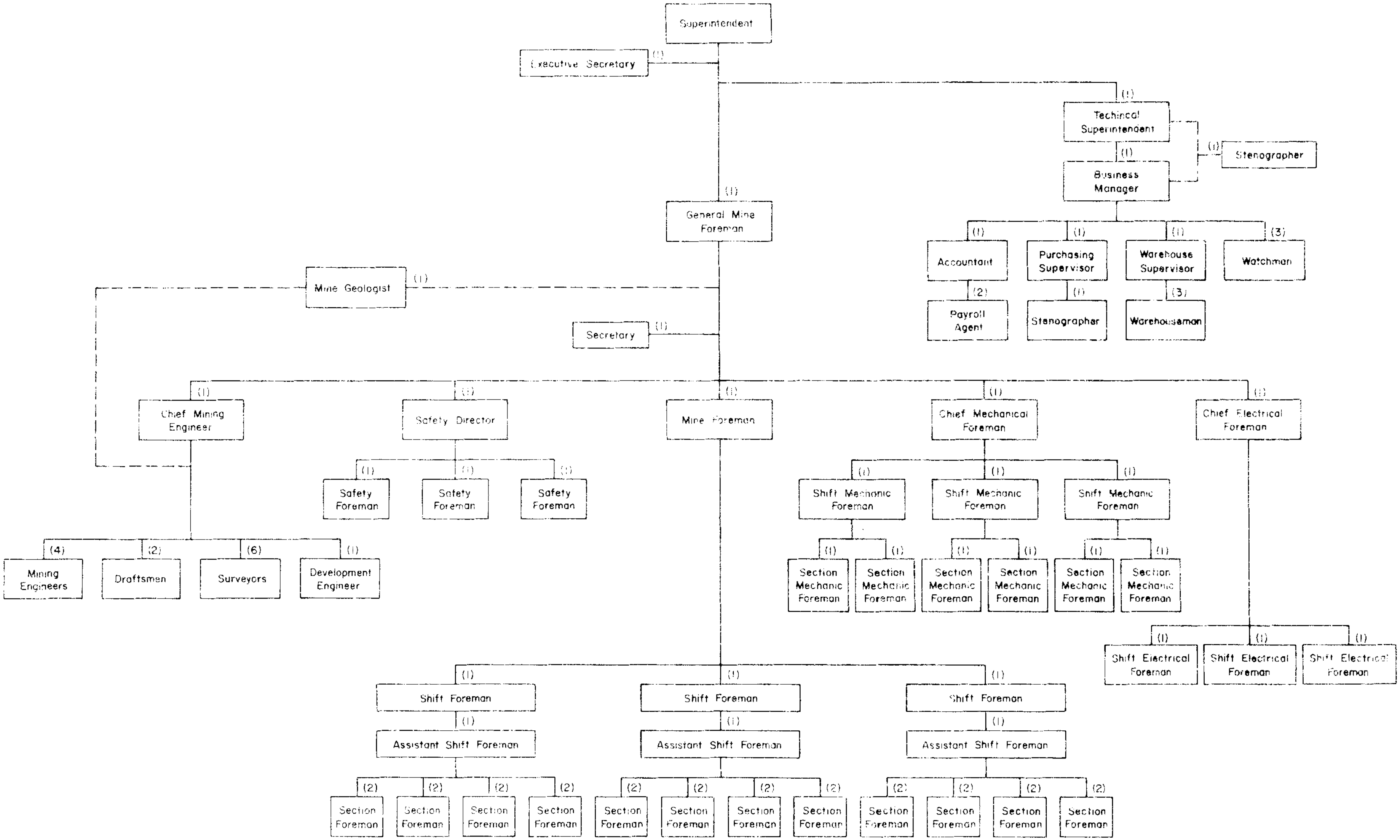


Figure 4.23 Management Flow Chart, 30.175 MM tpy, Advance Entry and Pillar Mining

Table 4.23 Capital Investment Summary, 30.175 MM tpy (Advance Entry and Pillar Mining)

ITEM	Quantity	Cost per Unit	Total Cost
Four boom hydraulic heading jumbo	5	$500,000	$2,500,000
Two boom hydraulic vertical drill	6	170,000	1,020,000
Two drill rock bolt machine	5	200,000	1,000,000
LHD machine (12yd^3)	38	180,000	6,840,000
Bulldozer	8	192,500	1,540,000
Primary ventilation fan	3	164,000	492,000
Portable impact crusher	10	250,000	2,500,000
36" floor mounted belt conveyor	11,840 ft	$52/ft	615,000
60" floor mounted belt conveyor	10,560 ft	$90/ft	955,000
Auxiliary fan (production)	10	7,400	74,000
Exhaust fan (machine shop)	1	8,000	8,000
Telephone (page phones)	10	450	5,000
Self rescuer	478	43	21,000
Stretcher set	10	235	2,400
First aid kits	20	30	600
Lamp (including accessories)	458	85	39,000
Fire chemical truck	4	25,000	100,000
Water truck	4	30,000	120,000
Lubrication and fuel truck	4	30,000	120,000
Personnel carriers	8	41,500	332,000
Powder and ANFO loading truck	8	28,000	224,000
Bathhouse, office and lamphouse	1	2,000,000	2,000,000
Underground machine shop	1	1,000,000	1,000,000
Warehouse and supply yard	1	130,000	130,000
Forklift	1	25,000	25,000
Substation and distributor	1	900,000	900,000
Rectifier (250 KW)	24	6,000	144,000
Rectifier (150 KW)	12	4,900	58,000
High voltage cable (8KV-4°)	10,000 ft.	$7/ft.	70,000
Methane detector	40	235	10,000
Portable fire extinguisher (20lb)	20	42	1,000
Fire suppression system for belt drive	6	600	4,000
Water pumps (10,000 gpm total)	16	33,000	528,000
Spent shale disposal equipment	1	500,000	500,000
8-inch diameter pipe	3,200 ft.	$4/ft.	13,000
4-inch diameter pipe	7,000 ft.	$2/ft.	14,000
Total direct cost			$23,905,000

Table 4.23 Capital Investment Summary, 30.175 MM tpy (Advance Entry and Pillar Mining) con't

ITEM	Total Cost
Contigencies (9% of total direct cost)	$ 2,195,000
Subtotal	26,057,000
Mobilization charge (2% of line above)	521,000
Total construction	26,578,000
Consulting engineering fees, overhead and administration (6% of line above)	1,595,000
Subtotal	28,173,000
Environmental Impact Statement	2,000,000
*Mine access (included are ventilation, production and service shafts, three double drum production hoists, one double drum service hoist and surge bins)	45,000,000
* Preproduction cost	8,975,000
Net capital investment estimate	$84,148,000
Value of 6,068,000 tons of 20 gpt oil shale stockpiled or retorted during preproduction development (assumes $11/barrel)	$31,785,000

* Assumes capitalization

Table 4.24 Depreciation Schedule, 10.175 MM tpy (Advance Entry and Pillar Mining)

ITEM	Straight-line depreciation (years)	Yearly charge
Four boom hydraulic heading jumbo	8	$ 312,500
Two boom hydraulic vertical drill	8	127,500
Two drill rock bolt machine	10	100,000
LHD (12 yd^3)	8	855,000
Bulldozer	10	154,000
Primary ventilation fan	30	16,400
Portable impact crusher	10	250,000
36" floor mounted belt conveyor	10	61,500
60" floor mounted belt conveyor	10	95,500
Auxiliary fan (production)	10	7,400
Exhaust fan (machine shop)	30	300
Mine safety equipment	5	36,200
Water truck	10	12,000
Lubrication and fuel truck	10	12,000
Personnel carrier	10	33,200
Powder and ANFO loading truck	10	22,400
Bathhouse, office and lamphouse	30	66,700
Underground machine shop	30	33,300
Warehouse and supply yard	30	4,400
Forklift	10	2,500
Substation	30	30,000
Rectifier (250 KW)	15	9,800
Rectifier (150 KW)	15	3,900
High voltage cable (8KV-4°)	30	2,400
8-inch pipe (waterline)	15	900
4-inch pipe (waterline)	15	1,000
Shaft access and ventilation	30	1,500,000
Water pumps (10,000 gpm total)	15	35,200
Spent shale equipment	10	50,000
Preproduction development	30	299,200
Environmental impact statement	30	66,700
Contingencies, mobilization, and engineering	30	142,300
Total		$4,344,200

Table 4.25 Estimated Yearly Interest Cost on Capitilized Items, 30.175 MM tpy
(Advance Entry and Pillar Mining)

Depreciable Life (yrs)	5	8	10	15	30	Total
Capital required	181,000	10,360,000	7,005,000	762,000	64,849,000	83,157,000
Interest charge						
10%	10,900	582,800	385,300	40,600	3,350,500	4,370,100
9.5%	10,300	553,600	366,000	38,600	3,183,000	4,151,500
9%	9,800	524,500	346,700	36,600	3,015,500	3,933,100
8.5%	9,200	495,300	327,500	34,500	2,848,000	3,714,500
8%	8,700	466,200	308,200	32,500	2,680,400	3,496,000

Table 4.26 Power and Water Cost, 30.175 MM tpy (Advance Entry and Pillar Mining)

Number of Units	Operation	H.P. per unit	H.P. per load	Hr. per day, full load	KW total load	Total KWH Requirement	Total* Water (gpd)
4	Four boom heading jumbo	300	1,200	15	895	13,428	86,400
4	Two boom vertical drill	150	600	15	448	6,714	43,200
2	60" conveyor belt drive	400	800	18	597	10,742	10,800
4	36" conveyor belt drive	75	300	18	224	4,028	21,600
8	Portable impact crushers	600	4,800	15	3,581	53,712	7,200
10	Auxiliary fans	40	400	15	298	4,476	--
3	Primary ventilation fans	2,000	6,000	24	4,476	107,424	--
4	Rock bolt machines	150	600	18	448	8,057	30,240
1	Auxiliary exhaust fan (machine shop)	105	105	24	78	1,872	--
	Miscellaneous (underground)	--	--	--	--	1,000	--
	Miscellaneous (surface)	--	--	--	--	1,000	10,000
3	Shaft hoist (production)	10,000	30,000	20	22,380	447,600	--
1	Shaft hoist (service)	5,000	5,000	20	3,730	74,600	--
2	Spent shale disposal	100	200	20	150	3,000	360,000
					Total	828,853	569,440

Electric power cost/yr = \$0.013 X 828,853 X 355 = \$3,825,000

* Water assumed to be provided by site wells.

quoted by a local Colorado public utility for the Central Piceance Basin. Water cost is neglected because it is assumed that more than enough water will be available from the mine itself. Table 4.27 is the estimated cost of preproduction, including the cost of interest on capital. The cost per ton figure is determined by dividing the total cost by the amount of preproduction development ore (6,068,000 tons).

Table 4.28 is a summary of the estimated annual production costs for an 85,000 tpd mining operation. Note that this cost includes only mining costs and no processing costs. Also included are yearly reclamation, exploration and environmental monitoring costs. The total estimated cost per ton is $1.27.

4.5.3 Chamber and Pillar Mine Design

Chamber and pillar mining is a modification of room and pillar mining in which crosscuts, driven perpendicular to main entries, are enlarged into chambers by fan drilling, Figure 4.24. This method has been designed especially for underground disposal of spent shale.

The chamber and pillar mining design is quite flexible in that variable mining heights, chamber lengths, and the mining distance away from the shaft or access area can vary according to geologic or economic conditions. In this analysis, a mining height of 60 feet and a chamber length of 450 feet were selected simply as a basis for costing. Pillar designs are also based on the 60-foot mining height.

The overall mine plan is multilevel with the distance between levels determined by either oil shale grade or a minimum safe sill pillar thickness. A high degree of mechanization is possible in this method, producing approximately 140 tons per man-shift. All designs and equipment selections are based on the assumption that the mine is gassy.

Preproduction development will not begin until the mine access, hoisting, and surface facilities are completed. The following discussions describe the mine design from preproduction through costing. The appendices at the end of the report contain some of the calculations used in the mine design.

Table 4.27 Estimated Preproduction Cost, 30.175 MM tpy (Advance Entry and Pillar Mining)

ITEM	Total Cost	Cost/Ton*
Total labor and supervision	$3,435,000	$0.57
Operating supplies		
Bits.	10,000	
Rock bolts, shells, plate	106,000	
Fuel and lubrication.	148,000	
ANFO, wire, and caps.	341,000	
Drill steel and couplings	46,000	
Contingencies (8%).	52,000	
Interest on capitalized equipment (three months).	870,000	
Subtotal.	1,573,000	0.26
Power	969,000	0.16
Payroll overhead (35%).	1,202,000	0.20
Indirect costs (10% labor, supervision, operating supplies).	414,000	0.07
Fixed costs		
Depreciation (three months)	1,011,000	0.17
Taxes and insurance (3% of 49,380,000 for three months)	371,000	0.06
Total . .	$8,975,000	$1.48

* Total tons = 6,068,000

Table 4.28 Estimated Annual Production Cost, 30.175 MM tpy (Advance Entry and Pillar Mining)

ITEM	Annual Cost	Cost/Ton
Direct costs		
Labor and supervision.	$ 9,709,000	$0.32
Operating supplies:		
Machine parts	1,084,000	
Lubrication and fuel.	654,000	
Rock bolts, shells, and plates. . .	286,000	
Drill bits.	80,000	
ANFO, caps, wire.	1,920,000	
Drill steel	1,070,000	
Ventilation tubing, bulkheads . . .	1,810,000	
Spent shale disposal.	550,000	
Subtotal.	7,454,000	
Operating contingencies (5% of line above).	373,000	
Yearly interest cost.	3,933,000	
Subtotal.	11,760,000	0.39
Power.	3,825,000	0.14
Reclamation.	1,000,000	0.03
Payroll overhead (35%)	3,398,000	0.11
Exploration.	500,000	0.02
Environmental monitoring	300,000	0.01
Indirect costs (10% of labor, supervision, and operating supplies, not including yearly interest)	1,754,000	0.06
Fixed costs		
Taxes and insurance (3% of 49,380,000)	1,482,000	0.05
Depreciation	4,344,000	0.14
Total	$38,072,000	$1.27

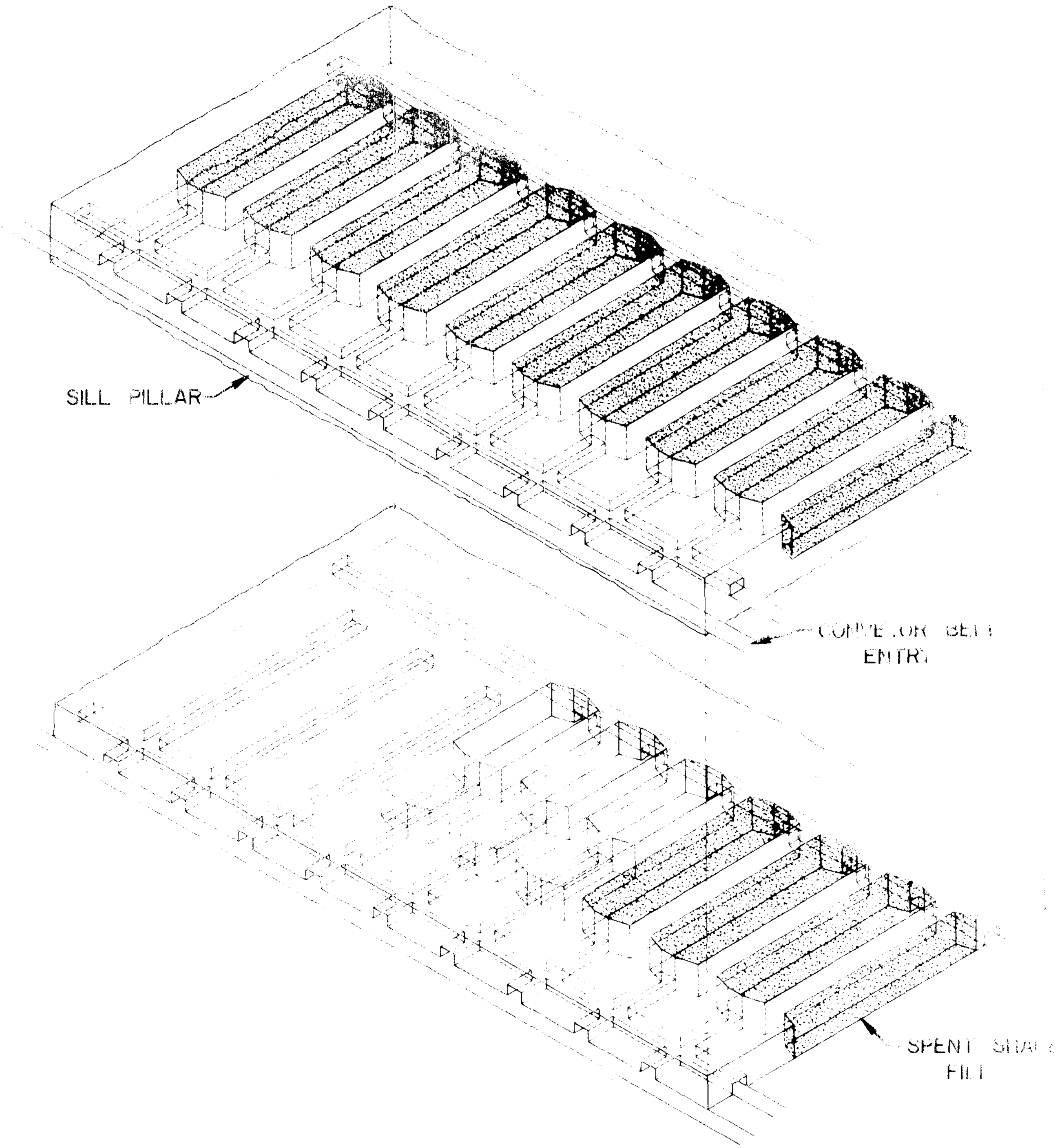


Figure 4.24 Isometric View, Chamber and Pillar Mining

4.5.3.1 Preproduction Development

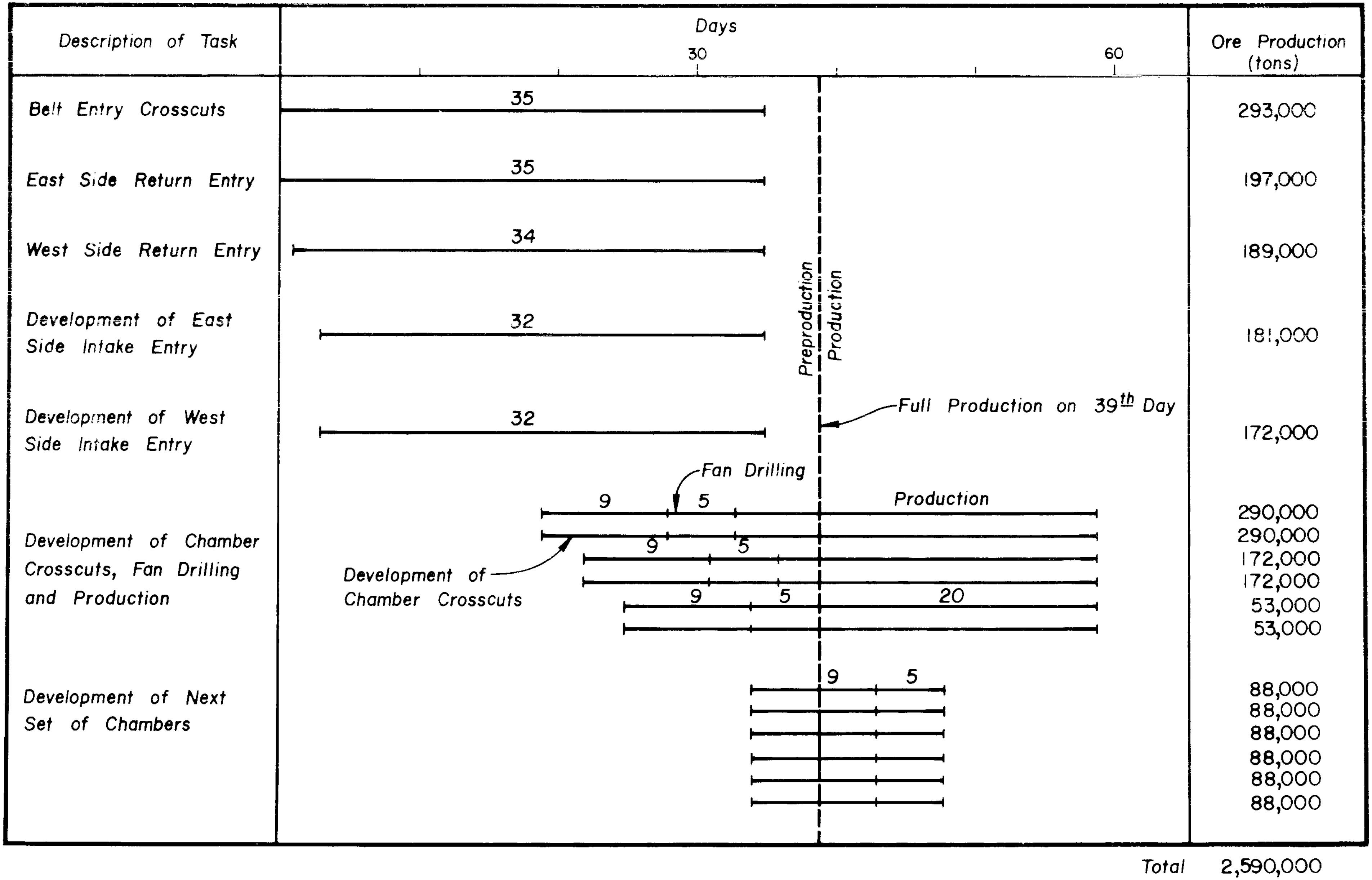
Development begins by driving five 30 by 30-foot entries a distance of 900 feet to the shaft pillar boundaries. Three of these entries are on 82-foot centers with crosscuts staggered on 140-foot centers, forming 52 by 110-foot chain pillars. The middle entry is used for belt haulage with the two adjacent entries used for intake air and access for men and supplies. The two return air entries are located 540 feet from either side of the intake entries and are used for exhaust air return. At the shaft pillar boundary, 20 by 30-foot chamber crosscuts are driven east and west from the intake air entries to the return air entries. The crosscuts are on 140-foot centers and 540 feet long. A total of 12 chamber crosscuts are developed during preproduction. It is estimated that preproduction development will take about 39 days to complete, assuming 60 feet advance per shift in the development headings (Table 4.29).

4.5.3.2 Production

Full production of 85,000 tpd is derived from the simultaneous mining of 12 chambers, six on each side of the shaft (76%), and the development of an additional 12 chamber crosscuts (24%). Chambers are initially driven 20 feet high and 30 feet wide from intake to return air entries. When a chamber crosscut is completed, fan drilling is started, enlarging the drift to 100 feet wide by 60 feet high, Figure 4.25. A 100-foot barrier pillar is left on the intake entry side and a 30-foot pillar on the return air side.

Fan drilling of the entire chamber crosscut is completed before loading and blasting. Once blasting begins, a minimum of two fans are blasted at once with mucking done using 12 yd^3 LHD diesel equipment. The LHD's dump the broken muck into 1000-tph capacity portable impact crushers, producing minus eight-inch ore. The crushed ore is fed onto a 60-inch conveyor belt, conveyed to the shaft storage pocket and hoisted to the surface. A total of 21 days is needed to mine 12 chambers at the rate of about 8100 tpd per chamber.

Table 4.29 Schedule of Preproduction Work for Chamber and Pillar Mining



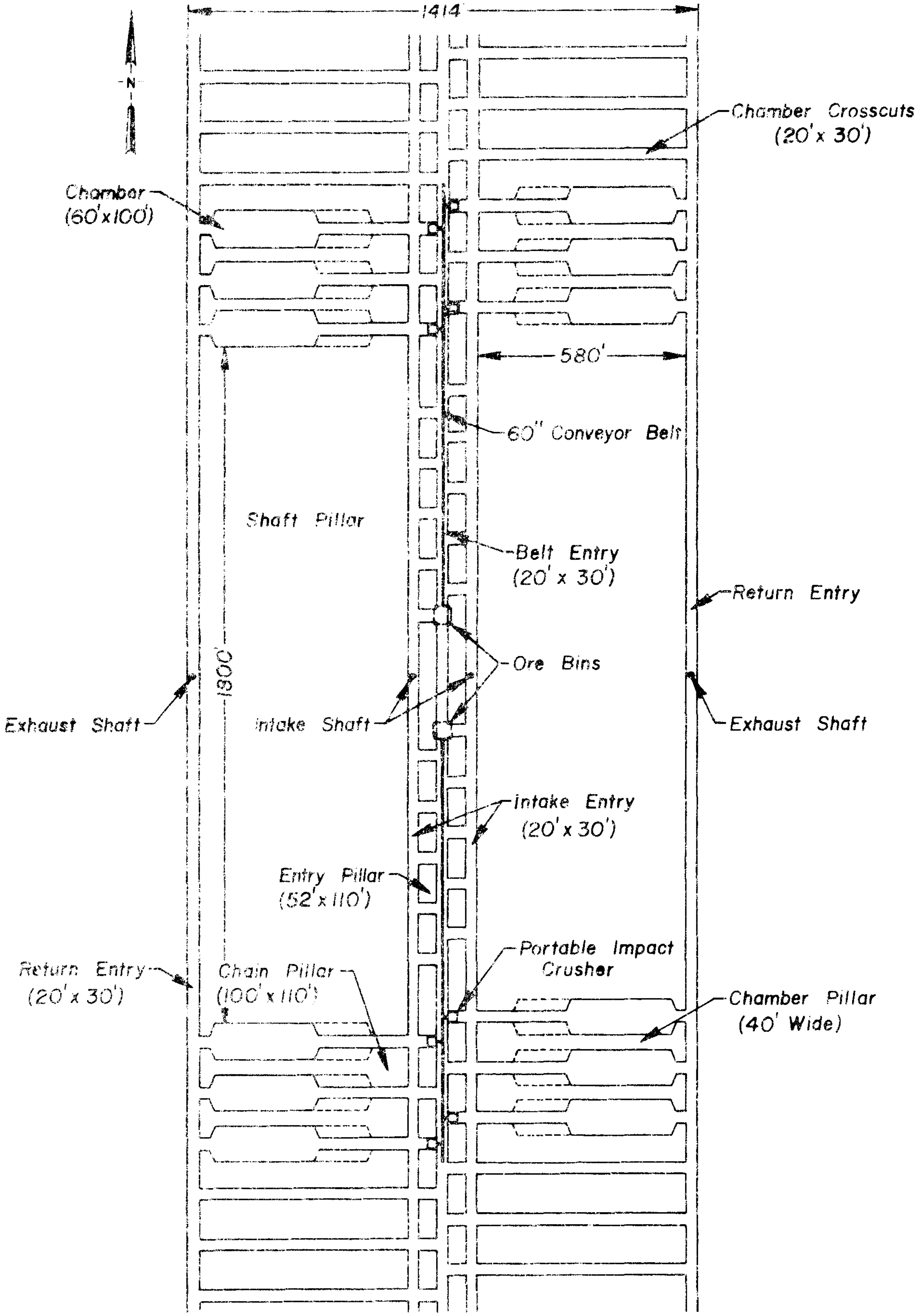


Figure 4.26 Plan View of Production Layout, Chamber and Pillar Mining

Roof bolting and scaling is done only in the development stages of the entry drifts and chamber crosscuts. Roof bolting patterns are eight-foot bolts on seven-foot centers in the entry drifts and five-foot bolts on five-foot centers in the chamber crosscuts.

Entry and chamber development drilling is done using six, two-boom rotary hydraulic drilling jumbos. Roof bolting and scaling are done simultaneously using six machines having two hydraulic drills per machine. Mucking of ore, for both development and chamber mining, is accomplished using 28, 12 yd^3 LHD's. A total of eight portable crushers is used to crush all ore before it reaches the 60-inch conveyor belt.

4.5.3.3 Spent Shale Backfilling

Spent shale backfilling is started when a chamber is completely mined out. The physical properties of the spent shale are assumed to be similar to those of a coarse sand. Facilities on the surface mix the water and spent shale to a desired pulp density and pump the mixture down a shaft, or through boreholes from the surface, in rubber lined pipe. As a chamber is widened, a backfill crew installs fill and drain piping along the length of the chamber. Porous bulkheads are constructed at the neck of the 20 by 30-foot chamber crosscut drift, prior to filling, to retain the fill material. Sumps in the chamber crosscuts collect the draining water and pump it to the surface for re-cycling. It is estimated that it will take less than 21 days to fill a chamber with spent shale.

4.5.3.4 Ventilation

The following discussion is based on rough estimates, assuming gassy mining conditions and approximations of the diesel horsepower output of mechanized equipment.

Primary ventilation is provided by exhaust fans mounted near the return air shafts, Figure 4.26. Air is pulled down the service and intake air shafts, circulated through the mine, and exits through the two exhaust shafts. Colorado mining laws (Colorado Bureau of Mines, 1971

4.5.3.6 Production Cost

The method of production costing for advance entry and pillar mining is similar to the format used by the USBM (Staff, 1972 and Katell and Hemingway, 1974). This costing method necessarily assumes that the oil shale mine is the only income source to the corporation; therefore, development costs (including mine access and hoisting) are capitalized. In reality, these costs would most likely be charged to production and deducted as negative cash flow in the overall corporate cash flow determination. Excluded in this analysis are royalty payments, welfare payments, and surface transportation costs.

Table 4.30 lists the supervisory and hourly personnel needed for the operation of an 85,000 tpd mine. An additional 12% of the hourly manpower requirements is included to account for absenteeism. Figure 4.26 is an example of the anticipated management flow chart for the underground mine only. Table 4.31 is the capital investment summary that includes contingencies, mobilization of capitalized equipment to mine site, consulting, environmental impact statement, mine access, and preproduction development. The total capital investment is estimated to be $77,200,000, not including the value of oil in the development ore (estimated to be worth $13,570,000).

Table 4.32 is the straight-line depreciation schedule for the equipment listed in Table 4.31. Table 4.33 is the estimated yearly interest cost on the money borrowed to finance the capital equipment. In this table, interest rates from eight to ten percent were assumed; however a rate of nine percent was used in the cost summary. Notice that the mine development costs are capitalized over a period of 30 years. Power and water consumption, Table 4.34, are estimated from vendor data, with the electric rate as estimated by a local Colorado public utility for the Central Piceance Basin. Water cost is neglected because it is assumed more than enough water will be available from the mine itself. Table 4.35 is the estimated cost of preproduction, including the cost of interest on capital. The cost per ton figure is determined by dividing the total cost by the amount of preproduction development ore (2,590,000 tons).

Table 4.30 Manning Table, 30.175 MM tpy
(Chamber and Pillar Mining)

Personnel	Total	Annual Cost per Employee	Annual Cost (260 Workdays)
Salary			
Superintendent	1	$33,000	$33,000
Technical superintendent	1	25,000	25,000
General mine foreman	1	27,000	27,000
Mine foreman	1	22,000	22,000
Shift foreman	3	20,000	60,000
Assistant shift foreman	3	19,000	57,000
Drilling foreman } section	12	18,000	216,000
Production foreman } section	12	18,000	216,000
Chief mechanical foreman	1	21,000	21,000
Shift mechanic foreman	3	19,000	57,000
Chief electrical foreman	1	21,000	21,000
Shift electrical foreman	3	19,000	57,000
Section mechanical foreman	6	18,000	108,000
Chief mine engineer	1	25,000	25,000
Mining engineer	5	20,000	100,000
Surveyor	3	10,800	32,400
Surveyor helper	3	9,000	27,000
Draftsman	2	9,600	19,200
Safety director	1	19,000	19,000
Safety foreman	3	16,500	49,500
Business manager	1	18,000	18,000
Accountant	1	9,600	9,600
Stenographer	2	6,600	13,200
Payroll agent	2	7,500	15,000
Purchasing supervisor	1	14,400	14,400
Warehouse supervisor	1	13,200	13,200
Warehouseman	3	7,200	21,600
Mine geologist	1	18,000	18,000
Watchman	3	6,000	18,000
Executive secretary	1	9,000	9,000
Secretary	1	7,800	7,800
Subtotal	83		$1,349,900

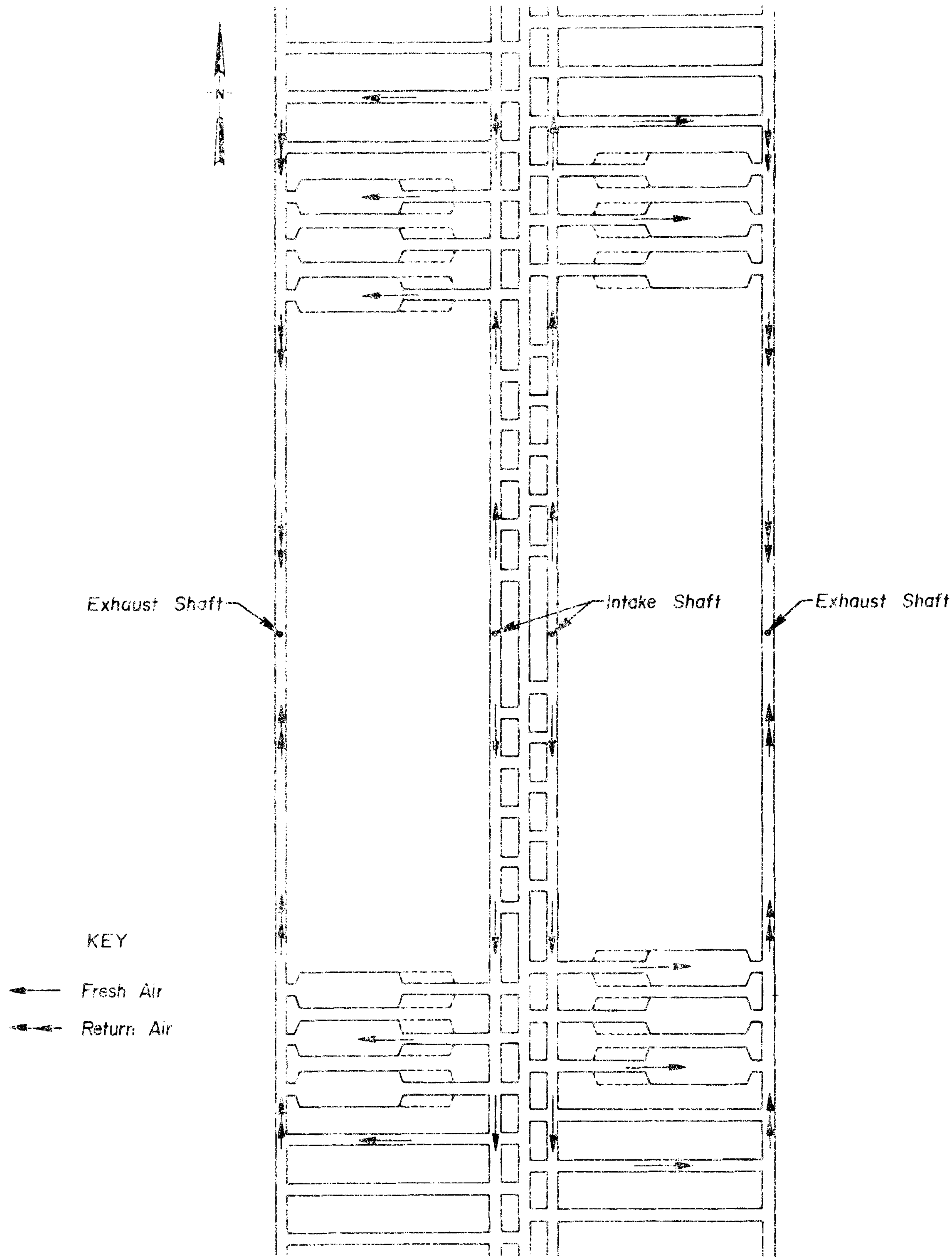


Figure 4.26 Plan View of Ventilation Network, Chamber and Pillar Mining

and State of Colorado, 1966) require 75 cfm of free air for each brake horsepower of diesel equipment and 100 cfm for each man. The calculations for estimated air requirements are presented in Appendix B. A total of 2,400,000 cfm of air is exhausted by two large axial-vane fans. Air flow direction is controlled by regulators and bulkheads. Ventilation to the entry and chamber crosscut headings are provided by 30,000 cfm auxiliary fans and ventilation tubing. Each chamber has about 110,000 cfm of regulated air, having a minimum velocity of 150 feet per minute. A 45% leakage loss of fresh air is assumed. All fans used underground are permissible.

Dust created by mining and haulage operations is partially controlled using the following procedures:

1. Wet drilling is employed on all drilling jumbos,
2. Muck piles are wetted several times before and during mucking operations,
3. Haulage ways are watered to suppress traffic generated dust.

4.5.3.5 Production Equipment Selection

Appendix B contains the calculations used to estimate equipment size, quantities, and cycle times for major equipment items. The capital expenditure tables in the following section on cost analysis summarize the equipment selections.

Evaluation of capital equipment for this large tonnage operation has been done relying almost exclusively on current vendor data. Wherever possible, only that equipment currently manufactured, whether as a full production item or as a prototype, was selected for analysis.

For this initial evaluation, operator efficiencies and minor equipment set-up times were approximated by considering only five hours of working time per eight-hour shift. The mine has been designed on a three-shift, seven-day per week work schedule. For the purpose of including holidays and vacations, a work-year is considered to be 355 days. Equipment availablities are estimated and range from 65 to 80%, depending on use and past performance of similar models.

Table 4.30 Manning Table, 30.175 MM tpy (Chamber and Pillar Mining) con't

Personnel	Total	Wages per day	Annual Cost (260 Workdays)
Underground			
Drilling operator	22	$51.81	$ 296,400
Drilling operator helper	22	49.96	285,800
LHD operator	86	50.88	1,137,700
Scaling & rock bolter	43	52.81	590,400
Truck drivers	29	50.88	383,600
Powderman	58	52.81	796,400
Crusherman	14	50.23	182,800
Laborer & slurryman	29	50.23	378,700
Conveyor beltman	18	50.23	235,100
Carpenter	10	50.88	132,300
Mechanic, first class	29	56.58	426,600
Mechanic, second class	29	52.90	398,900
Electrician, first class	14	56.58	206,000
Electrician, second class	14	52.90	192,600
Master machinist	4	56.58	58,800
Machinist	14	51.80	188,600
Welder	7	51.80	94,300
Subtotal	442		$5,985,000
Outside			
Hoistman	4	$51.70	$ 53,800
Cage tender	4	49.77	51,800
Lampman	4	47.15	49,000
Subtotal	12		$ 154,600
Contingencies for absenteeism (12%)	55		$ 736,752
Total labor and supervision	592		$8,226,300

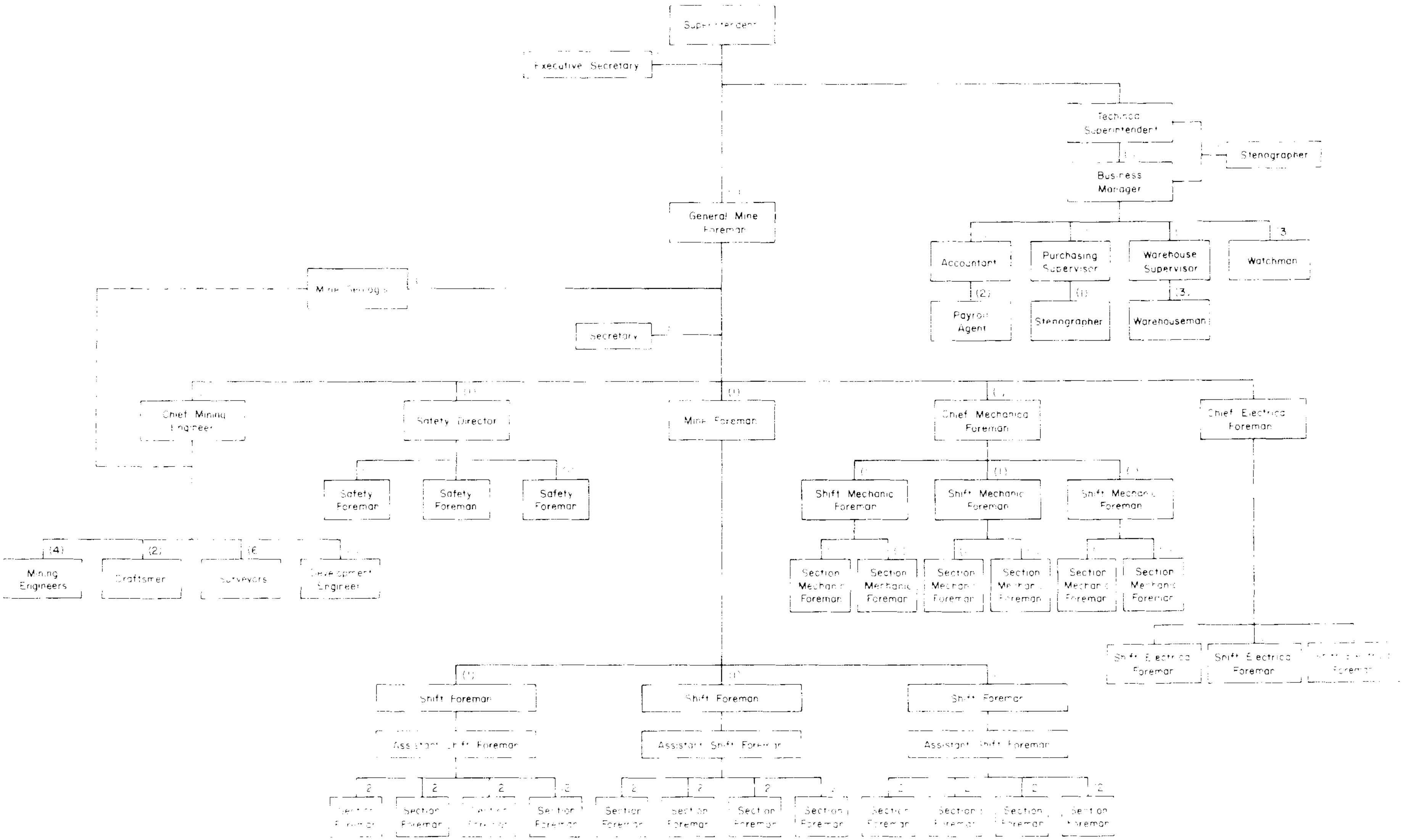


Figure 4.27 Management Flow Chart, 30.175 MM tpy, Chamber and Pillar Mining

Table 4.31 Capital Investment Summary, 30.175 MM tpy (Chamber and Pillar Mining)

ITEM	Quantity	Cost per Unit	Total Cost
Two boom hydraulic drill jumbo	8	$310,000	$2,480,000
LHD machine (12yd^3)	38	180,000	6,840,000
Two drill rock bolt machine	8	200,000	1,600,000
Primary ventilation fan	2	164,000	328,000
Portable impact crusher	10	250,000	2,500,000
60" floor mounted belt conveyor	21,120 ft	$126/ft	2,662,000
Auxiliary fan (production)	10	7,400	74,000
Exhaust fan (machine shop)	1	8,000	8,000
Telephone (page phones)	10	450	5,000
Self rescuer	463	43	20,000
Stretcher set	10	235	3,000
Lamp (including accessories)	440	85	38,000
First aid kits	20	30	1,000
Personnel carriers	6	41,500	249,000
Water trucks	2	30,000	60,000
Lubrication and fuel truck	4	30,000	120,000
Fire chemical truck	4	25,000	100,000
Powder and ANFO loading truck	8	28,000	224,000
Bathhouse, office and lamphouse	1	2,000,000	2,000,000
Underground machine shop	1	1,000,000	1,000,000
Warehouse and supply yard	1	130,000	130,000
Forklift	1	25,000	25,000
Substation and distribution	1	900,000	900,000
Rectifier (250 KW)	14	6,000	84,000
Rectifier (150 KW)	1	4,900	5,000
Rectifier (75 KW)	4	3,000	12,000
Water pumps (10,000 gpm total)	16	33,000	528,000
High voltage Cable (8KV-4°)	20,000 ft	$7/ft	140,000
Methane detector	40	235	10,000
Portable fire extinguisher (20 lb)	20	42	1,000
Fire suppression system for belt drive	2	600	1,000
8" pipe (water line)	3,200 ft	$4/ft	13,000
4" pipe (water line)	11,000 ft	$2/ft	22,000
Spent shale disposal equipment	500,000		500,000
Total direct cost			$22,683,000

Table 4.31 Capital Investment Summary, 30.175 MM tpy (Chamber and Pillar Mining) con't

ITEM	Total Cost
Contingencies (9% of total direct cost).	$ 2,042,000
Subtotal	24,725,000
Mobilization charge (2% of line above)	495,000
Total construction	25,220,000
Consulting engineering fees, overhead, and administration (6% of line above).	1,513,000
Subtotal	26,733,000
Environmental impact statement	2,000,000
* Mine access (included are ventilation, production, and service shafts, three double drum production hoists, one double drum service hoist and surge bins).	45,000,000
* Preproduction cost	3,469,000
Net capital investment estimate.	$77,200,000
Value of oil in 2,590,000 tons of 20 gpt oil shale stockpiled or retorted during preproduction development (assumes $11/BBL).	$13,570,000

* Assumes capitalization

Table 4.32 Depreciation Schedule, 30.175 MM tpy (Chamber and Pillar Mining)

ITEM	Straight-line depreciation (years)	Yearly charge
Two boom hydraulic drill jumbo	8	$ 310,000
LHD machine (12 yd^3)	8	855,000
Two drill rock bolt machine	10	160,000
Primary ventilation fan	30	11,000
Portable impact crusher	10	250,000
60" floor mounted belt conveyor	10	266,000
Auxiliary fan (production)	10	7,400
Exhaust fan (machine shop)	30	300
Mine safety equipment	5	35,200
Water truck	10	6,000
Lubrication and fuel truck	10	12,000
Personnel carrier	10	24,900
Powder and ANFO loading truck	10	22,400
Bathhouse, office and lamphouse	30	66,700
Underground machine shop	30	33,300
Warehouse and supply yard	30	4,300
Forklift	10	2,500
Substation	30	30,000
Rectifier (250 KW)	15	5,700
Rectifier (150 KW)	15	300
Rectifier (75 KW)	15	800
Water pumps (10,000 gpm total)	15	35,200
High voltage cable (8 KV-4°)	30	4,700
8" pipe (water line)	15	900
4" pipe (water line)	15	1,600
Spent shale disposal equipment	10	50,000
Shaft access and ventilation	30	1,500,000
Preproduction development	30	115,600
Environmental impact statement	30	66,700
Contingencies, mobilization, and engineering	30	135,000
	Total	$4,013,700

Table 4.33 Estimated Yearly Interest Cost on Capitalized Items, 30.175 MM tpy (Chamber and Pillar Mining)

Depreciable Life (yrs)	5	8	10	15	30	Total
Capital required	176,000	9,320,000	8,014,000	668,000	59,028,000	77,206,000
Interest charge 10%	10,600	524,300	440,700	35,600	3,049,800	4,061,000
9.5%	10,000	498,100	418,700	33,800	2,897,300	3,857,900
9%	9,500	471,800	396,700	32,100	2,897,300	3,654,900
8.5%	9,000	445,600	374,700	30,300	2,592,300	3,451,900
8%	8,500	419,400	352,600	28,500	2,439,800	3,248,800

Table 4.34 Power and Water Cost, 30.175 MM tpy (Chamber and Pillar Mining)

Number of Units	Operation	H.P. per Unit	H.P. total Load	Hr. per day full Load	KW total Load	Total KWH Requirement	Total* Water (gpd)
6	Two boom drilling jumbo	135	810	15	604	9,064	81,000
2	60" conveyor belt drive	400	800	18	597	10,742	9,000
8	Portable impact crusher	600	4,800	15	3,581	53,712	7,000
2	Primary ventilation fan	2,000	4,000	24	2,984	71,616	--
10	Auxiliary fan	40	400	15	298	4,476	--
1	Auxiliary exhaust fan (machine shop)	105	105	24	78	1,872	--
6	Rock bolt machine	150	900	18	671	12,078	--
	Miscellaneous (underground)	--	--	--	--	1,000	--
	Miscellaneous (surface)	--	--	--	--	1,000	10,000
3	Shaft hoist (production)	10,000	30,000	20	22,380	447,600	--
1	Shaft hoist (service)	5,000	5,000	20	3,730	74,600	--
2	Spent shale disposal	100	200	20	150	3,000	360,000
12	Water pumps	425	5,100	24	3,800	91,200	--
					Total	781,960	467,000

Electric power cost/yr = \$0.013 X 781,960 X 355 = \$3,609,000

* Water assumed to be provided by site wells.

Table 4.35 Estimated Preproduction Cost, 30.175 MM tpy (Chamber and Pillar Mining)

ITEM	Total Cost	Cost/Ton*
Total labor and supervision	$1,234,000	$0.47
Operating supplies		
Bits	14,000	
Rock bolts, shells and plates. . . .	34,000	
Fuel and lubrication	63,000	
ANFO, wire and caps.	146,000	
Drill steel and couplings.	27,000	
Contingencies (8%)	23,000	
Interest on capitalized equipment. (39 days)	374,000	
Subtotal	681,000	0.26
Power	397,000	0.15
Payroll overhead (35%).	432,000	0.17
Indirect costs (10% labor, supervision, operating supplies)	154,000	0.06
Fixed costs		
Depreciation (39 days)	417,000	0.16
Taxes and insurance (3% of 48,048,000 for 39 days)	154,000	0.07
Total.	$3,469,000	$1.34

* Total tons = 2,590,000

Table 4.36 is a summary of the estimated annual production cost for an 85,000 tpd mining operation. Note that this cost includes only mining costs and no processing costs. Also included are yearly interest, reclamation, exploration, and environmental monitoring costs. The total estimated cost per ton is $1.04.

Table 4.36 Estimated Annual Production Cost, 30.175 MM tpy (Chamber and Pillar Mining)

ITEM	Annual cost	Cost/Ton
Direct costs		
Labor and supervision	$ 8,226,000	$0.27
Operating supplies		
Machine parts.	627,000	
Lubrication and fuel	653,000	
Rock bolts, shells and plates. . .	82,000	
Drill bits	235,000	
ANFO, caps, and wire	1,920,000	
Drill steel.	273,000	
Ventilation (tubing, bulkheads). .	123,000	
Spent shale disposal	280,000	
Subtotal	4,193,000	
Operating contingencies (5% of line above)	210,000	
Yearly interest cost	3,655,000	
Subtotal	8,058,000	0.24
Power	3,609,000	0.12
Reclamation	1,000,000	0.03
Payroll overhead (35%).	2,879,000	0.10
Exploration	500,000	0.02
Environmental monitoring.	300,000	0.01
Indirect costs (10% of labor, supervision, and operating supplies, not including yearly interest cost)	1,263,000	0.04
Fixed costs		
Taxes and insurance (3% of $48,048,000)	1,442,000	0.05
Depreciation	4,014,000	0.13
Total	$31,291,000	$1.04

4.6 DESIGN AND ANALYSIS OF BLOCK CAVING SYSTEMS IN OIL SHALE

Block caving is considered to be one of the lowest cost underground mining methods for mining large, thick ore bodies. For block caving to be successful, the ore must be weak or fractured enough so that the ore does not arch or cave catastrophically. As mentioned in Section 2, there are several areas where extensive leaching has occurred, resulting in fracturing and reduced rock mass strength. These areas may be readily adaptable to block caving techniques.

Current block caving methods, as used in the United States, can be readily identified by their respective systems of ore transfer and haulage. The two most common techniques are slusher loading and LHD loading. In the designs presented here, both systems are investigated to determine the lowest cost system. Also included for each analysis are estimates of the cost of inducing caving (commonly termed boundary weakening) should the ore be difficult to cave.

The following sections cover a review of literature and the designs of block caving systems using slushers and LHD machines. Each analysis includes a description of the mine design from preproduction through production costing.

4.6.1 Review of Literature

A comprehensive discussion of block caving is presented by Julin and Tobie (1973) in the SME Mining Engineering Handbook. The paper includes a general description of block caving together with pertinent case studies. The listed advantages of block caving include: (1) higher degree of safety, (2) low cost mining, (3) high degree of mechanization, and (4) better supervision due to a more centralized operation. The disadvantages mentioned are (1) much longer preproduction development period, (2) drift maintenance costs are high, (3) drift support costs are high, (4) system is not flexible, and (5) ore dilution is necessary and can be excessive.

Dravo Corporation (1974) has compiled a large amount of data from several block caving operations throughout the world. Resource recovery rates are reported to vary from 67 to 100% with dilution varying from 10 to 15%. Mining costs were found to vary from $0.75 to $1.92 per ton of ore.

Retardo (1972) discusses a block caving system at the Philex Mine, Philippine Islands, that uses the slusher system for primary ore haulage. Ore is scraped into ore passes using 25 to 35 hp slushers and then loaded into ore cars. The distance between the slusher and haulage levels is approximately 30 feet. Total daily production is approximately 21,000 tpd with a 48 to 55 tons per manshift factor for development and production, respectively.

A modified induced block caving system (Kenedo and Higashi, 1972) has been adapted by the Kamioka Mine in Japan. The mine previously operated for eight years using sublevel caving tehcniques. After changing to induced block caving the labor efficiency increased from five to 21 tons per manshift, resulting in a decrease in production costs from $6.56 to $1.96 per ton.

Allsman (1964) suggests mining oil shale by block caving and includes a system for restoring the subsidence cavity using spent shale. The design includes 600-foot square by 300-foot high blocks having a draw hole spacing of 80 by 80 feet. Other ideas include having a primary crusher underground and using a 7,000 tph capacity belt conveyor to transport ore to the surface. Mining production costs are estimated at $1.19 per ton.

4.6.2 Block Caving Design Using Slusher Transport

The block caving system presented here is designed to mine a 2000-foot thick ore body under 1000 feet of cover. The ore body is mined in three successive lifts starting at the top. The caving system is retreat in nature and starts within an economic distance from the access area, estimated for this analysis to be 4,100 feet (Figure 4.28), including

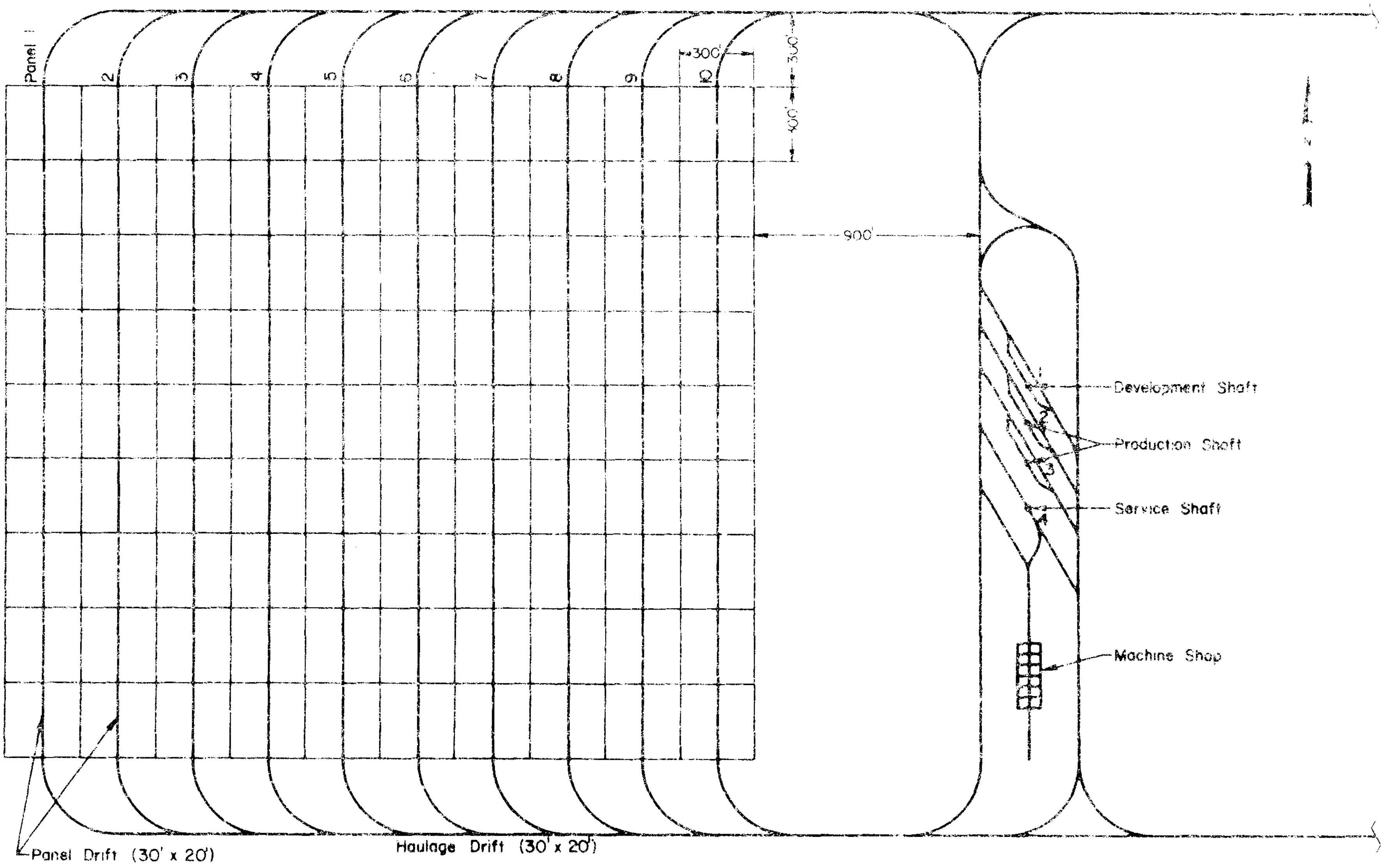


Figure 4.28 Plan View of Rail Haulage Layout, Block Caving with Slushers

1100 feet for a shaft pillar. Each panel contains nine blocks, 300 by 300 feet square and 550 feet high. Ore is loaded directly into ore cars on the main haulage level from the slusher dashes above and transported by trolley locomotive to a primary crusher at the shaft station. Ore is crushed in a 60-inch gyratory crusher and loaded into skips to be hoisted to the surface. The mine access consists of two 28-foot diameter production shafts, one 28-foot diameter development shaft, one 28-foot diameter service shaft, and two 16-foot diameter exhaust ventilation shafts.

Preproduction development starts after the shafts are sunk to the desired depth and the primary crushing facilities are completed. Included in preproduction development is driving the main haulage level to the outer boundary, and driving the necessary footages of ventilation sub-drifts, slusher dashes, and undercutting drifts needed to produce 85,000 tpd.

4.6.2.1 Preproduction Development

Main Haulage Level. The primary functions of the 20 by 30-foot haulage level are rail haulage transport of ore from the panels and blocks and fresh air ventilation. Drifts are driven using two boom hydraulic drill jumbos, 8 yd^3 FEL's, 80-ton rail cars, rock bolting and scaling machines, and truck mounted ANFO loading machines. Average heading rounds are 15 feet long with an estimated 45 feet per day advance rate. Rock bolting provides primary back support and consists of 6-foot bolts on five-foot centers. Double track is laid in the panel drifts to facilitate car loading and increase track usage.

An 85,000 tpd production rate requires that the main haulage drifts be driven around the boundary of the panels, and five panel drifts be completed. As the main haualge drifts are advanced, work also begins on the ventilation levels, Table 4.37.

Ventilation Level. Figure 4.29 is an isometric drawing of a single block in a panel showing the relationship between the haulage, production, and development levels. The primary functions of the ventilation level

Table 4.37 Schedule of Preproduction Work for Block Caving with Slushers

Description of Task	Days	Ore Production Tons
Develop shaft access drifts	42	[illegible]4,000
Drive main haulage drifts	150	616,000
Finish shaft access area and machine shop	100	277,000
First ventilation crosscut	111	[illegible]71,000
2nd, 3rd, 4th, 5th and 6th ventilation crosscuts	55, 55, 55, 55, 55	239,000
Drive 2nd, 3rd, 4th, and 5th panel drifts	55, 55, 55, 55	736,000
Drive 108 slusher dashes and drive finger raises	327	341,000
Concrete the slusher dashes	327	
Ramp up to the undercut level and drive undercut drifts	488	319,000
Fan drill pillars	243	
Blast pillars, intermediate production	255	10,837,000
	Total	13,750,000

Days: 30 60 90 120 150 180 210 240 270 300 330 360 390 420 450 480 510 540 570 600 630 660

Full Production

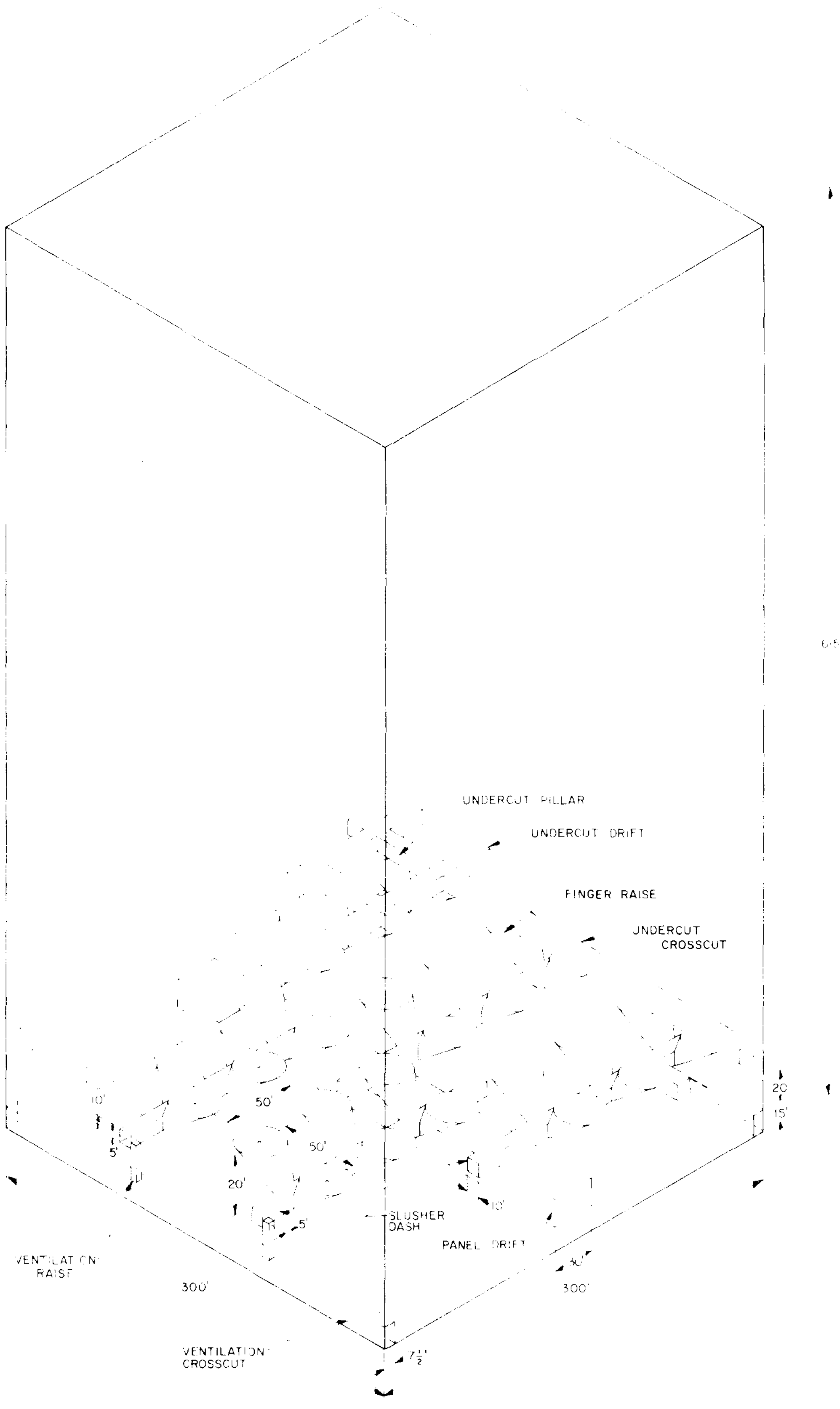


Figure 4.29 Isometric View of Block Layout Within a Panel, Block Caving With Slushers

are to provide a return air system that has no working areas in which air is reused and to control ventilation in the slusher dashes. Figure 4.30 shows the overall ventilation layout. The primary ventilation drifts are 30 by 20 feet and 20 feet below the main haulage level. Access to the ventilation level is from the main haulage level down a ten percent incline. Ventilation crosscuts (15 by 15 feet) are driven parallel to the panels at right angles to the slusher dashes. As each crosscut passes under the end of a slusher dash, a small diameter raise is driven to connect the two and complete the ventilation circuit. Two 16-foot diameter shafts as shown in Figure 4.30, are used for exhaust ventilation only and have no hoisting or service facilities. Equipment used to develop the ventilation level is similar to that used on the main haulage level.

Slusher Dashes. The floor of a slusher dash, at the intersection with a panel drift, is even with the roof of the panel drift. The slusher dashes are driven perpendicular to the panel drifts at a plus five percent grade alternately going east and west, Figure 4.31. The dashes are on 50-foot centers and completely concreted, having a finished inside dimension of 10 by 10 feet. Access to the slusher level is by incline from the haulage level. Ore from development and production is loaded directly into ore cars without going through grizzlies or transfer raises, Figure 4.32.

Finger Raises. Finger raises, shown in cross section in Figure 4.33, are the ore transfer raises from the undercut or caving level to the slusher level. The raises are spaced on 50-foot centers under the caving zone so that an even drawdown of the ore is possible. To reduce wear, the raises are concreted two-thirds of their length. Driving the raises is accomplished using a combination of drilling jumbos and stopers. The initial dimension of a raise is 10 by 10 feet at the junction of the slusher dash with the top belled to produce an inverted cone.

Undercut Drifts. Undercut drifts are 10 by 10 feet in cross section, on 50-foot centers, and connect all of the finger raise drawpoints, Figure 4.33. Access to the under cut level is by an extension of the incline from the haulage level to the slusher level. Small-boom jumbos and 5 yd^3 LHD

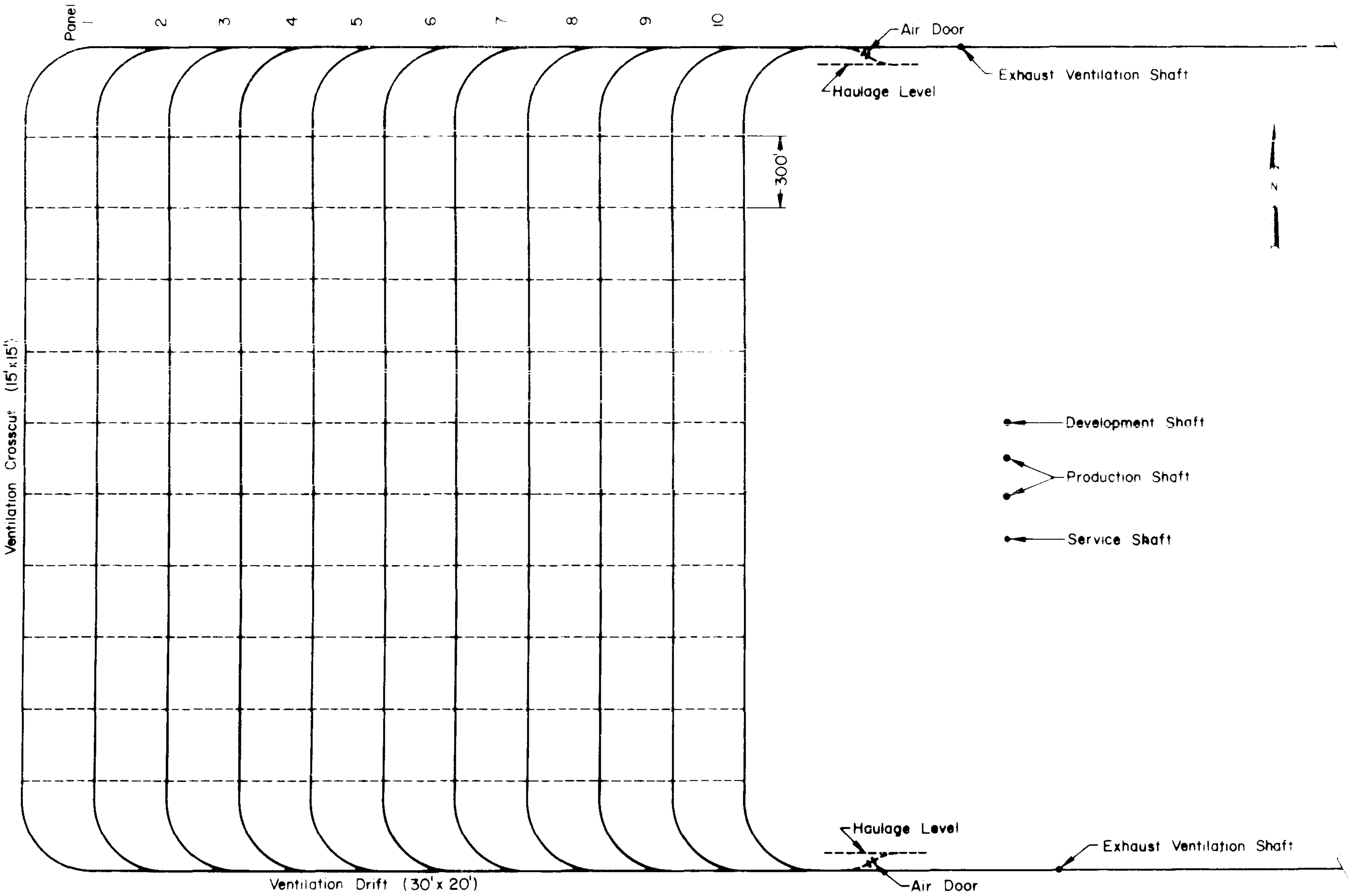


Figure 4.30 Plan View of Ventilation Level, Block Caving with Slushers

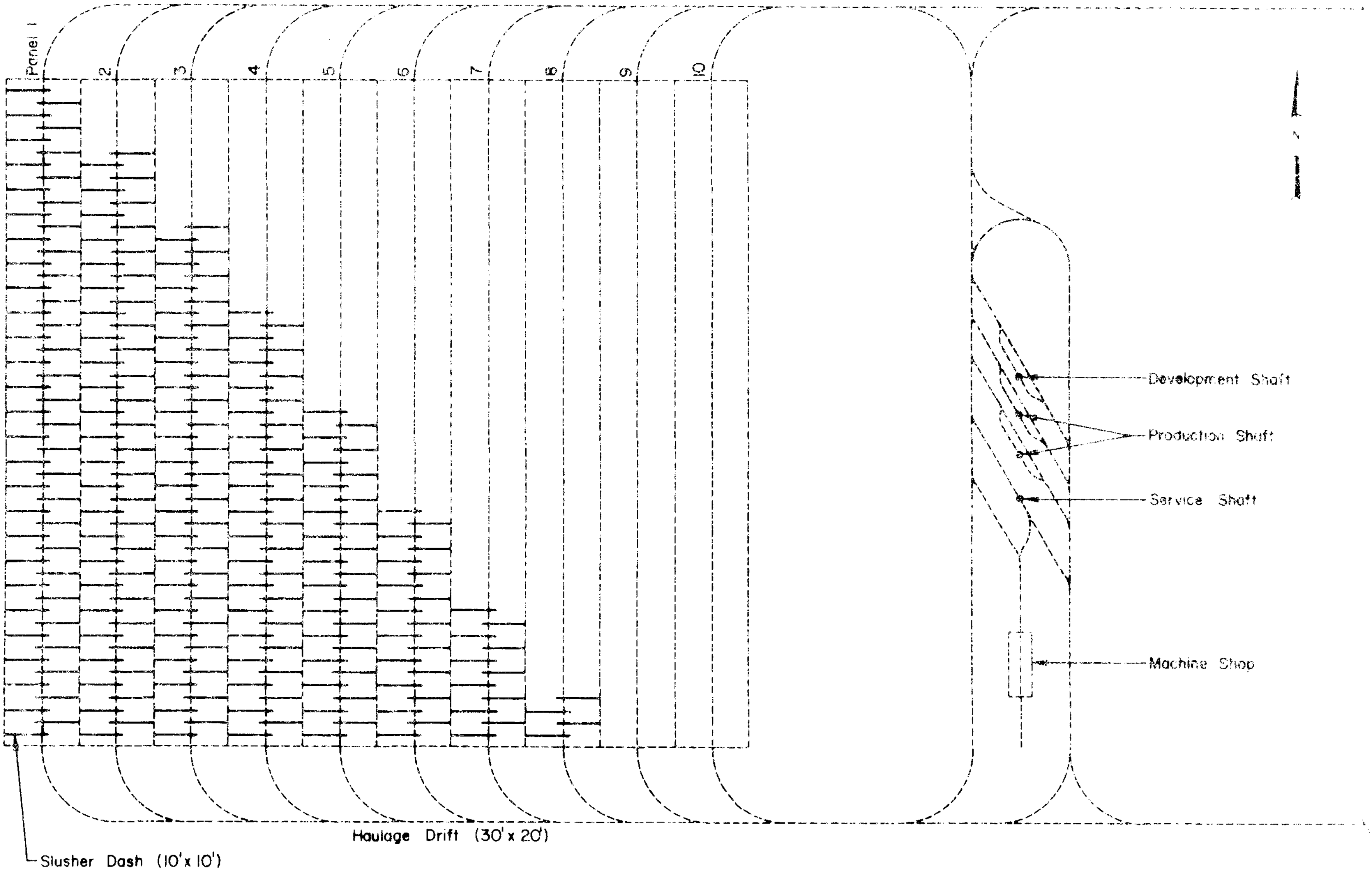


Figure 4.31 Plan View of Slusher Level, Block Caving with Slushers

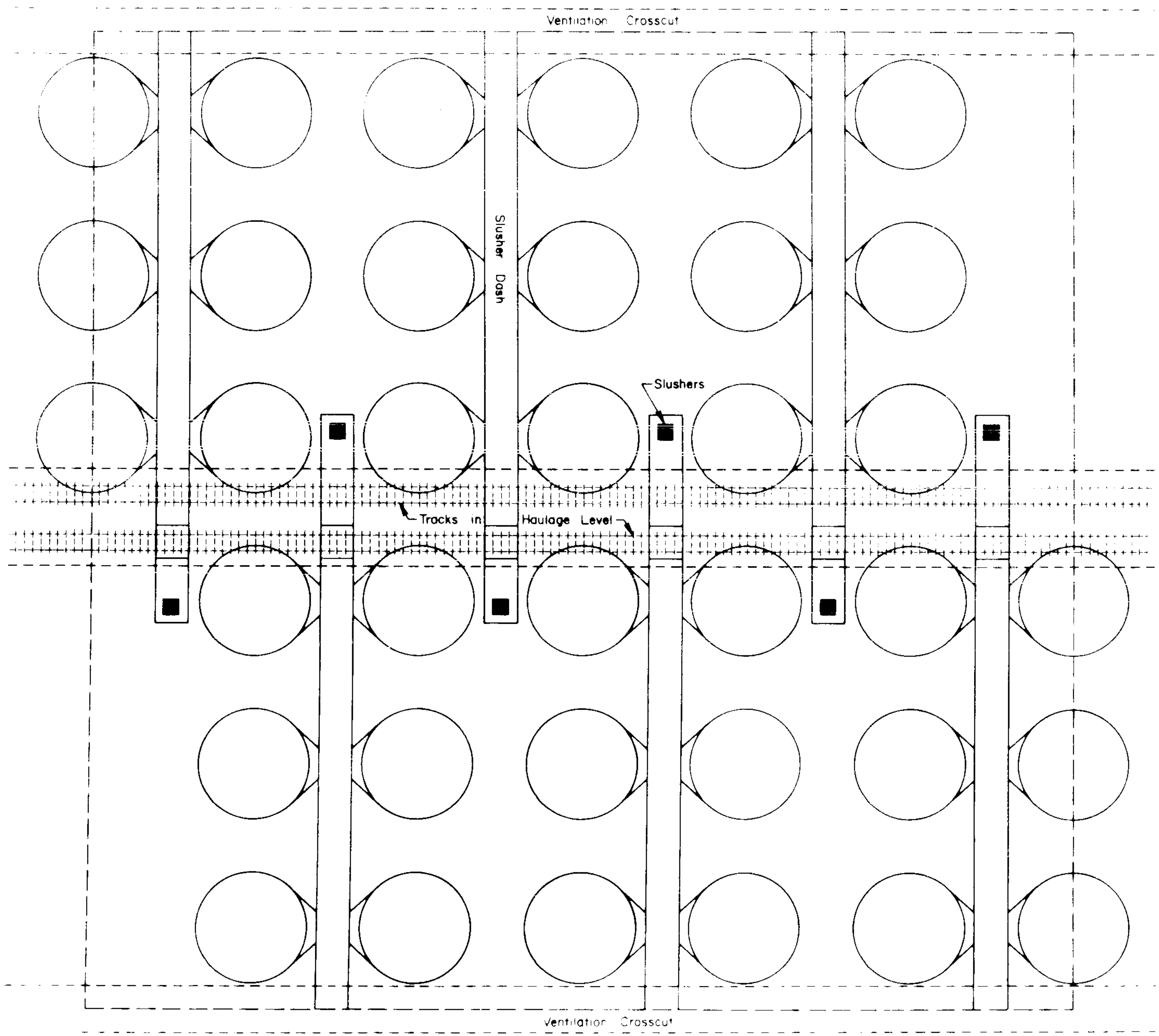


Figure 4.32 Plan View of Slusher Level within a Block, Block Caving with Slushers

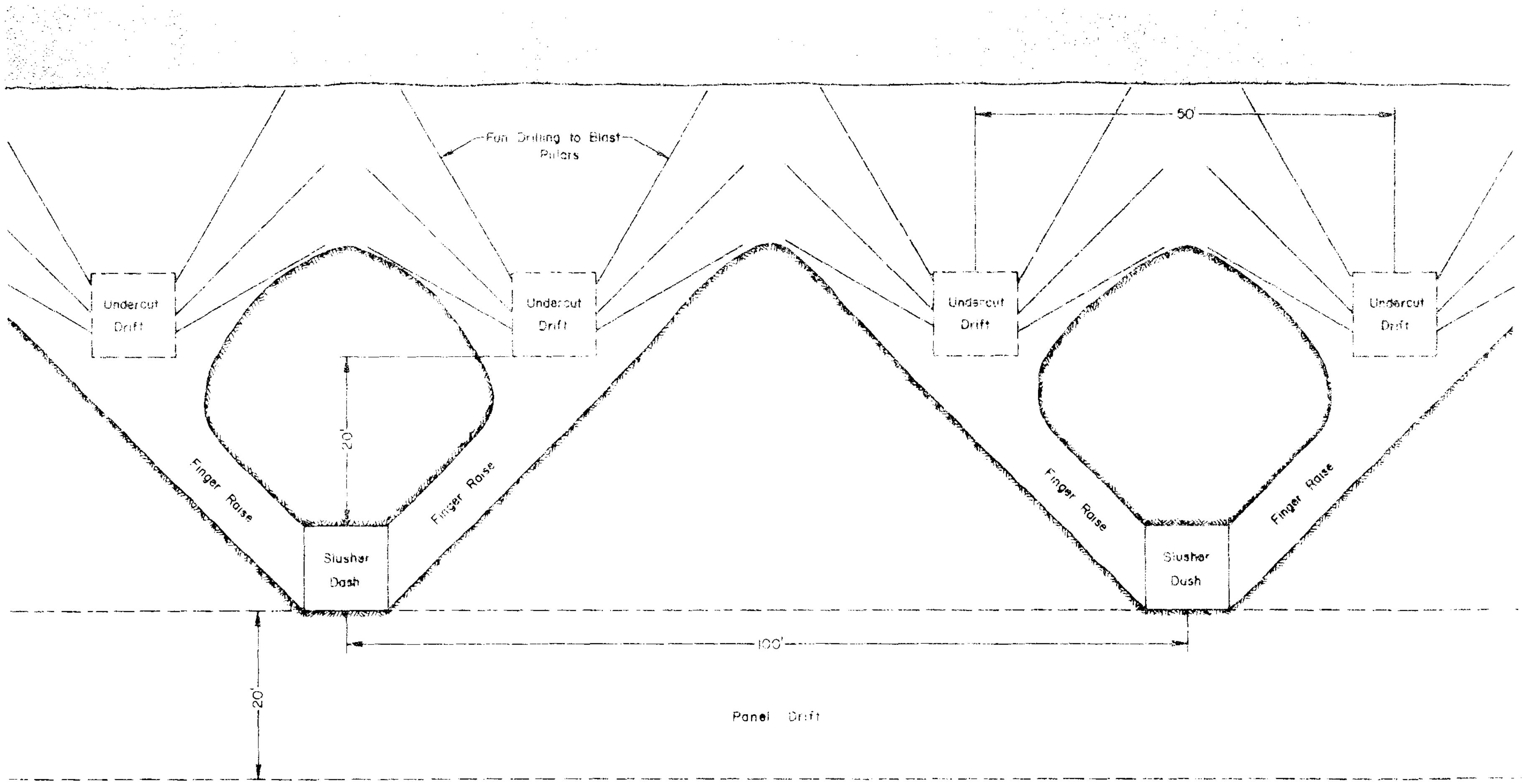


Figure 4.33 Vertical Section Showing Haulage Drift, Slusher Lines, Finger Raises and Undercut Drifts, Block Caving with Slushers

equipment is used to develop the level. Development muck is dumped down the finger raises. As development proceeds, the pillars over the slusher dash are fan drilled to form a peaked pillar and blasted to remove support from the overlying rock. The fan drill holes are three inches in diameter and average 20 to 30 feet in length. Pillar blasting starts in one corner and retreats toward the shaft, as illustrated in Figure 4.31.

4.6.2.2 Production

At full production 18 blocks or 108 slusher dashes are needed, having an average draw rate of 9 inches per day. The 9-inch draw rate, including slusher down-time and operator efficiencies, gives an average block life of two years. The slusher dashes are manned by 54 slusher operators with each man responsible for two slusher dashes. Nine blocks are developed each year to ensure adequate production is maintained. The ore flows by gravity down the finger raises and is drawn by a 150-hp slusher, with 84-inch scraper, along the slusher dash and dropped directly into 80-ton cars. Fifty-ton trolley locomotives then pull ten-car trains to the primary crusher, using a one-way traffic pattern.

When development is completed on the first level, a second level is started. Men and equipment are gradually moved to the lower level so that there is no interruption in minimum daily production. Development ore is hoisted up a shaft used specifically for development hoisting. Increased production above 85,000 tpd is possible by starting a similar caving operation on the east side of the shaft area. The mining system, as it is designed here, has a capacity, with some men and equipment additions, to produce up to 170,000 tpd.

4.6.2.3 Ventilation

Air quality requirements are computed considering a gassy mine, using diesel equipment underground, and using only permissible and explosion proof machinery. Colorado mining laws (Colorado Bureau of Mines, 1971 and State of Colorado, 1966) require an additional 75 cfm of free air for each brake horsepower of diesel equipment used underground. Preliminary calculations for estimated mine ventilation requirements are presented in Appendix C.

Air comes into the mine through the service and production shafts to the main haulage level at the rate of approximately 1,500,000 cfm. The air is then regulated through the various panel drifts into the slusher dashes and out the ventilation level. Two large axial-vane fans in each of two exhaust shafts, Figure 4.30, are used to develop the required air flow. A 60 to 70% air loss is assumed through the caved zone and regulators.

Air quantity in the undercut and slusher levels during development is maintained through the use of secondary fans. Two exhaust fans are used to ventilate the maintenance shop area. Dust suppression is accomplished by using water sprays and wetting muck piles before mucking.

4.6.2.4 Production Equipment Selection

Appendix C contains the calculations used to estimate equipment size, quantities, and cycle times for major equipment items. The capital expenditure tables in the following section on cost analysis summarize the equipment selections.

Evaluation of capital equipment for this large tonnage operation has been done relying almost exclusively on current vendor data. Wherever possible, only that equipment currently manufactured, whether as a full production item or as a prototype, was selected for analysis.

For this initial evaluation operator efficiencies and minor equipment set up times were approximated by considering only five hours of working time per eight-hour shift. The mine has been designed on a three-shift, seven-day per week work schedule. For the purpose of including holidays and unexpected work stoppages, a work year was considered to be 355 days. Equipment availabilities are estimated and range from 65 to 90%, depending on use and past performance of similar models.

4.6.2.5 Production Cost

The method of production costing for block caving using slushers is similar to the format presented in the USBM (Staff, 1972, Katell and Hemingway, 1974). This costing method necessarily assumes that the oil shale mine is the only income source to the corporation; therefore, development costs (including mine access and hoisting) are capitilized. In reality, these costs would most likely be charged to production and deducted as negative cash flow in the overall corporate cash flow determination. Excluded in this analysis are royalty payments, welfare payments, and surface transportation costs. All costs were collected during first quarter, 1975.

Table 4.38 lists the supervisory and hourly personnel needed for the operation of an 85,000 tpd mine. An additional 12% of the hourly manpower requirements are included to account for absenteeism. Figure 4.34 is an example of the anticipated management flow chart for the underground mine only. Table 4.39 is the capital investment summary that includes contingencies, mobilization of capitalized equipment to mine site, consulting, environmental impact statement, mine access, and preproduction development. The total capital investment is estimated at $128,700,000, not including the value of oil in the development ore (estimated to be worth $72,000,000).

Table 4.40 is a straight-line depreciation schedule for the equipment listed in Table 4.39. Table 4.41 is the estimated yearly interest cost on the money borrowed to finance the capital equipment. In this table, interest rates from eight to ten percent are assumed; however a rate of nine percent is used in the cost summary. Notice that the mine development costs are capitalized over a period of 30 years. Power and water consumption, Table 4.42 are estimated from vendor data, and the electric rate as quoted by a local Colorado public utility for the Central Piceance Creek Basin. Water cost is neglected because it is assumed more than enough water will be available from the mine itself. Table 4.43 is the estimated cost of preproduction, including the cost of interest on capital. The cost per ton figure is determined by dividing the total cost by the amount of preproduction development ore ($13,750,000 tons).

Table 4.38 Manning Table, 30.175 MM tpy (Block Caving with Slushers)

Personnel	Total	Annual Cost Per Employee	Annual Cost (260 Workdays)
Salary			
Superintendent	1	$33,000	$33,000
Administrative superintendent	1	25,000	25,000
General mine foreman	1	27,000	27,000
Mine foreman	1	22,000	22,000
Shift foreman	3	20,000	60,000
Assistant shift foreman	3	19,000	57,000
Development foreman	15	18,000	270,000
Production foreman	27	18,000	486,000
Chief mechanical foreman	1	21,000	21,000
Chief electrical foreman	1	21,000	21,000
Shift mechanic foreman	3	19,000	57,000
Production mechanic foreman	3	18,000	54,000
Development mechanic foreman	3	18,000	54,000
Shift electrical foreman	3	19,000	57,000
Chief engineer	1	25,000	25,000
Mining engineer	5	20,000	100,000
Draftsman	2	9,600	19,200
Surveyor	3	10,800	32,400
Surveyor helper	3	9,000	27,000
Safety director	1	19,000	19,000
Safety foreman	3	16,500	49,500
Business manager	1	18,000	18,000
Accountant	1	9,600	9,600
Purchasing supervisor	1	14,400	14,400
Warehouse supervisor	1	13,200	13,200
Watchman	4	6,000	24,000
Payroll agent	2	7,500	15,000
Warehouseman	4	7,200	28,800
Stenographer	1	6,600	6,600
Secretary	1	7,800	7,800
Executive secretary	1	9,000	9,000
Mine geologist	1	18,000	18,000
Subtotal	102		$1,680,500

Table 4.38 Manning Table, 30.175 MM tpy (Block Caving with Slushers) cont'd.

Personnel	Total	Wages Per Day	Annual Cost (260 Workdays)
Underground			
Driller	58	$51.81	$781,300
Driller helper	58	49.96	753,400
FEL operator	15	50.88	198,400
LHD operator	29	50.88	383,600
Slusher operator	196	51.81	2,640,200
Locomotive operator	18	50.88	238,100
Scaler and rock bolter	43	52.81	590,400
Powderman	43	52.81	590,400
Trackman	22	50.88	291,000
Ventilation man	7	50.88	92,600
Electrician, first class	14	56.58	206,000
Electrician, second class	14	52.90	192,600
Pipefitter	7	56.58	103,000
Pipefitter helper	7	52.90	96,300
Mechanic, first class	24	56.58	353,100
Mechanic, second class	24	52.90	330,100
Warehouseman	4	49.96	52,000
Concrete man	22	49.96	285,800
Form man	22	49.96	285,800
Supplyman	29	49.96	376,700
Grader operator	4	50.88	52,900
Laborer	14	49.96	181,900
Crusher operator	4	52.90	55,000
Batch plant operator	7	49.96	90,900
Truck driver	8	50.88	105,800
Master mechanic	4	56.58	58,800
Dispatcher	4	50.88	52,900
Hangup man	44	56.58	647,300
Welder	4	51.81	53,900
Subtotal	749		10,140,200
Outside			
Mechanic	1	$54.76	$14,200
Mechanic helper	1	50.88	13,200
Forklift operator	4	49.96	52,000
Lampman	4	47.15	49,000
Hoistman	8	51.70	107,500
Janitor	4	47.15	49,000
Cageman	4	49.77	51,800
Subtotal	26		$ 336,700
Contingency for absenteeism (12%)	93		1,257,200
Total labor and supervision	970		13,414,600

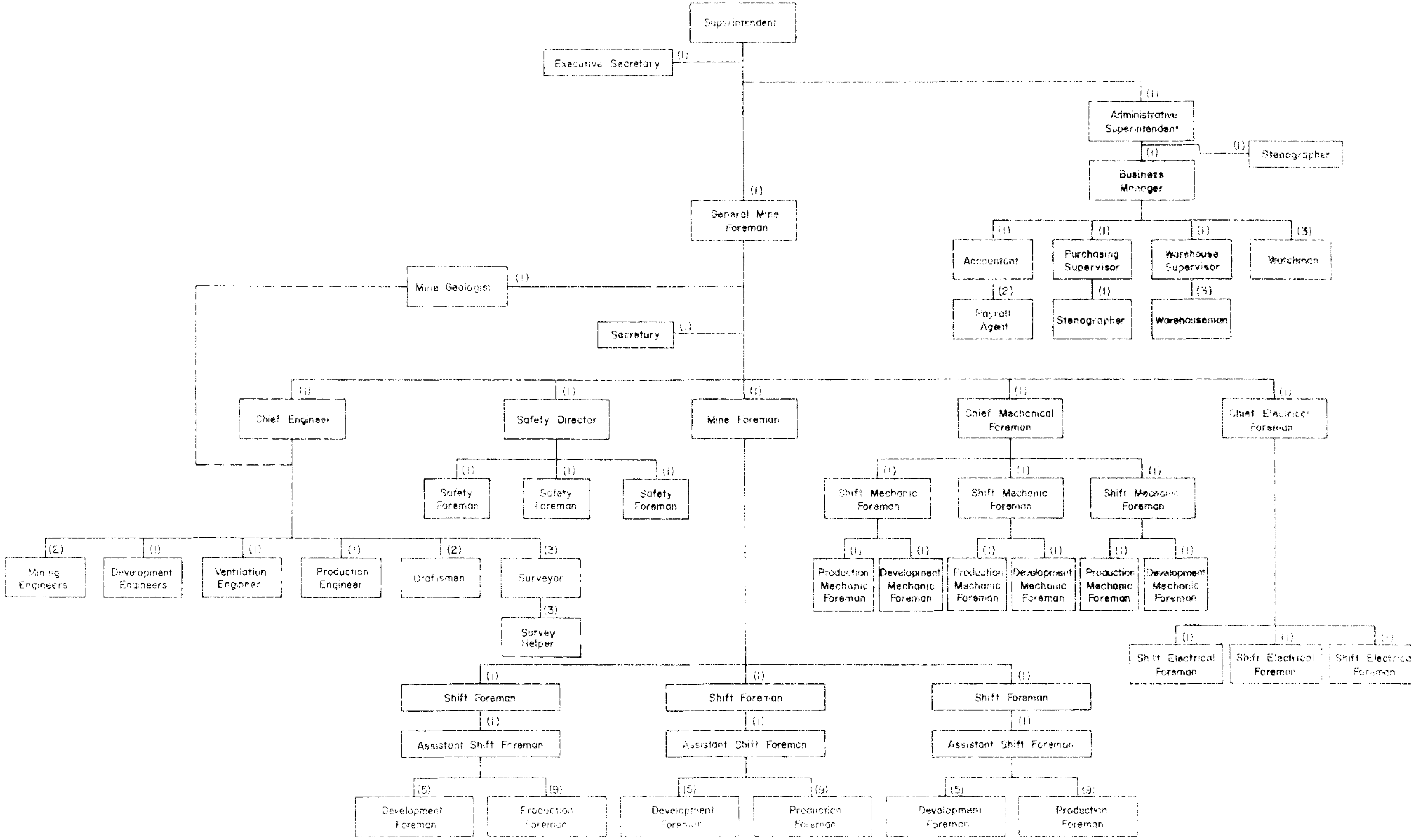


Figure 4.34 Management Flow Chart, 30.175 MM tpy, Block Caving with Slushers

Table 4.39 Capital Investment Summary, 30.175 MM tpy (Block Caving with Slushers)

ITEM	Quantity	Cost Per Unit	Total Cost
Two boom hydraulic drill jumbo	6	$310,000	$1,860,000
Two boom medium drill jumbo	6	62,000	372,000
Two boom small drill jumbo	6	38,000	228,000
Two boom fan drill	6	300,000	1,800,000
FEL (8 yd^3)	6	160,000	960,000
LHD (8 yd^3)	12	140,000	1,680,000
Slushers (150 hp)	108	40,000	4,320,000
Scrapers (84")	108	6,100	659,000
50-ton trolley locomotive	6	250,000	1,500,000
80-ton cars (air brakes)	60	23,000	1,380,000
30-ton trolley locomotive	5	190,000	950,000
8-ton battery locomotive	2	75,000	150,000
Flatcars and supply car	20	10,000	200,000
Concrete vat	20	20,000	400,000
Two drill rock bolt machine	8	150,000	1,200,000
Personnel carrier	6	41,500	249,000
Powder and ANFO loading truck	4	28,000	112,000
Lubrication and fuel truck	4	30,000	120,000
Service vehicle	10	30,000	300,000
Underground machine shop	1	1,000,000	1,000,000
Mine rail (115 lb)	260,000	$4.81/ft	1,251,000
First aid kit	30	30	1,000
Track plates	65,000	$2.15	140,000
Treated ties	65,000	$6.64	432,000
Battery charging equipment	2	10,000	20,000
Road grader	2	60,000	120,000
Lamp (including accessories)	800	85	68,000
Fire chemical truck	4	25,000	100,000
Water truck	2	30,000	60,000
Bathhouse, office and lamphouse	1	2,000,000	2,000,000
Warehouse ard supply yard	1	130,000	130,000
Forklift	1	25,000	25,000
Substation and distribution	1	900,000	900,000
Transformers and rectifiers			200,000
High voltage cable (8KV-4°)	30,000 ft	$7/ft	210,000
Methane detector	40	435	10,000
Portable fire extinguisher (20 lb)	20	42	1,000
Primary ventilation fan	4	115,000	460,000
Auxiliary fans	10	7,400	74,000
Exhaust fans (for machine shop)	2	8,000	16,000
Self rescuer	800	43	35,000
telephone (page phones)	10	450	5,000
Stretcher set	10	235	3,000
Pumps (10,000 gpm total)	16	33,000	528,000

Table 4.39 Capital Investment Summary, 30.175 MM tpy (Block Caving with Slushers) con't.

ITEM	Quantity	Cost per Unit	Total Cost
Compressor	5	20,000	$100,000
Tow truck	1	30,000	30,000
60-inch gyratory crusher	2	1,300,000	2,600,000
Track maintenance machine	1	50,000	50,000
Concrete plant	1	250,000	250,000
Trolley wire	130,000	$0.68/ft	88,000
Total direct cost			$29,347,000
Contingencies (9% of total direct cost)			2,641,000
Subtotal.			31,988,000
Mobilization charge (2% of the line above).			640,000
Total Construction.			32,628,000
Consulting engineering fees, overhead and administration (6% of line above)			1,958,000
Subtotal.			34,586,000
Environmental impact statement.			2,000,000
* Mine access (included are ventilation, production, and service shafts, two double drum production hoists, one double drum development hoist, one double drum service hoist, surge bins, and two rotary dump machines.			50,538,000
Subtotal.			87,124,000
* Preproduction cost.			41,572,000
Net capital investment.			$128,696,000
Value of oil in 13,750,000 tons of oil shale stockpiled or retorted during preproduction development (assumes $11/bbl)			$72,024,000

* Assumes capitalization

Table 4.40 Depreciation Schedule, 30.175 MM tpy (Block Caving with Slushers)

ITEM	Straight-line depreciation (years)	Yearly charge
Two boom hydraulic drill jumbo	8	$232,500
Two boom medium drill jumbo	8	46,500
Two boom small drill jumbo	8	28,500
Two boom fan drill	8	225,000
FEL (8-yd^3)	8	120,000
LHD (6 to 8-yd^3)	8	210,000
Slusher (150 hp)	15	288,000
Scrapers (84")	10	65,900
50-ton trolley locomotive	15	100,000
80-ton car (air brakes)	10	138,000
30-ton trolley locomotive	15	63,300
8-ton battery locomotive	15	10,000
Flatcar and supply car	10	20,000
Concrete vat	10	40,000
Two drill rock bolt machine	10	120,000
Personnel carriers	10	24,900
Powder and ANFO loading truck	10	11,200
Lubrication and fuel truck	10	12,000
Service vehicle	10	30,000
Underground machine shop	30	33,300
Mine rail (115 lb)	30	41,700
Track plates	30	4,700
Treated ties	30	14,400
Battery charging equipment	10	2,000
Road grader	8	15,000
Water truck	10	6,000
Bathhouse, office and lamphouse	30	66,700
Warehouse and supply yard	30	4,300
Forklift	10	2,500
Substation and distributor	30	30,000
Transformers and rectifiers	15	13,300
High voltage cable	30	7,000
Primary ventilation fan	30	15,300
Auxiliary fan	10	7,400
Exhaust fan (for machine shop)	30	500
Mine safety equipment	5	44,600
Pumps (10,000 gpm total)	15	35,200
Air compressor	10	10,000
Tow truck	10	3,000
Gyratory crusher	30	86,700
Track maintenance machine	15	3,300
Shaft access, ventilation, rotary dump	30	1,684,600
Environmental impact statement	30	66,700

Table 4.40 Depreciation Schedule, 30.175 MM tpy (Block Caving with Slushers) cont'd.

ITEM	Straight-line depreciation (years)	Yearly charge
Contingencies, mobilization, and engineering	30	\$ 165,400
Concrete plant	30	8,300
Trolley wire	30	2,900
Preproduction development	30	1,385,700
	Total	\$5,546,300

Table 4.41 Estimated Yearly Interest Cost on Capitilized Items, 30.175 MM tpy (Block Caving with Slushers)

Depreciable Life (yrs)	5	8	10	15	30	Total
Capital Required	223,000	7,020,000	4,829,000	7,696,500	105,561,000	125,329,500
Interest Charge						
10%	13,380	394,880	265,600	410,480	5,454,000	6,538,340
9.5%	12,700	375,130	252,300	389,960	5,181,300	6,211,390
9%	12,040	355,390	239,000	369,430	4,908,600	5,884,460
8.5%	11,370	335,640	225,800	348,910	4,635,900	5,557,620
8%	10,700	315,900	212,500	328,380	4,363,200	5,230,680

Table 4.42 Power and Water Cost, 30.175 MM tpy (Block Caving with Slushers)

Number of Units	Operation	H.P. per unit	H.P. Total load	Hrs. per day full load	KW total load	Total KW requirements	Total* water (gpd)
4	Two boom hydraulic drill jumbo	135	540	15	403	6,040	54,000
4	Two boom hydraulic fan drill	135	540	15	403	6,040	54,000
4	Two boom medium drill jumbo	-	-	15	-	-	40,000
4	Two boom small drill jumbo	-	-	15	-	-	30,000
6	Rock bolting machine	150	900	18	671	12,080	33,000
4	Primary ventilation fan	1,350	5,400	24	4,027	96,680	
10	Auxiliary fan	150	1,500	24	1,119	26,860	
2	Exhaust fan	105	210	24	157	3,760	
5	50-ton trolley locomotive	720	3,600	15	2,685	40,280	
4	30-ton trolley locomotive	380	1,520	15	1,133	17,000	
2	Double drum hoist (production)	10,000	20,000	20	14,914	298,400	
2	Double drum hoist (service, development)	5,000	10,000	20	7,460	149,200	
2	Underground crusher	700	1,400	20	1,044	20,880	12,000
56	Slusher	150	8,400	15	6,266	94,000	
8	Compressors (1,000 cfm)	100	800	15	597	8,950	
12	Pumps	425	5,100	24	3,803	91,300	
	Miscellaneous (underground)				1,000	12,000	12,000
	Miscellaneous (surface)				1,000	12,000	6,000
					Total	895,470	241,000

Electric power cost/yr = \$0.013 X 895,470 X 355 = \$4,132,600

* Water assumed to be provided by site wells.

Table 4.43 Estimated Preproduction Cost, 30.175 MM tpy (Block Caving with Slushers)

ITEM	Total cost	Cost/Ton*
Total labor and supervision.	$12,631,00	$0.97
Operating supplies		
Machine parts.	714,000	
Bits	103,000	
Rock bolts, shells, and plates . .	268,000	
Lubrication and fuel	900,000	
ANFO, caps and wire.	333,000	
Drill steel and couplings.	88,000	
Concrete	809,000	
Operating contingencies (8%) . . .	257,000	
Interest on capitilized equipment (635 days).	7,048,000	
Subtotal.	10,519,000	0.76
Power.	2,428,000	0.18
Payroll overhead (35%)	4,421,000	0.32
Indirect costs (10% of labor, supervision and operating supplies) . .	1,610,000	0.12
Fixed cost:		
Depreciation (635 days).	7,238,000	0.52
Taxes and insurance (3% of 52,214,000 for 635 days).	2,725,000	0.20
Total . . .	$41,572,000	$3.02

* Total tons = 13,750,000

Table 4.44 is a summary of the estimated annual production cost for an 85,000 tpd mining operation. It must be understood that this cost includes only mining costs and does not include processing costs. Also included are yearly interest, reclamation, exploration, and environmental monitoring costs. The total estimated cost per ton is $1.35.

Table 4.44 Estimated Annual Production Cost, 30.175 MM tpy (Block Caving with Slushers)

ITEM	Annual cost	Cost/Ton
Direct costs		
Labor and supervision.	$13,415,000	$0.44
Operating supplies		
Machine parts	733,000	
Drill bits.	61,000	
Rock bolts, shells and plates . .	152,000	
Fuel and lubrication.	520,000	
ANFO, wire and caps	200,000	
Drill steel and couplings	81,000	
Concrete cost	415,000	
Subtotal. . .	2,162,000	
Operating contingencies (5% of line above).	108,000	
Yearly interest cost (9%)	5,884,000	
Subtotal. . .	8,154,000	0.27
Power	4,133,000	0.14
Reclamation	1,000,000	0.03
Payroll overhead (35%).	4,695,000	0.16
Exploration	500,000	0.02
Environmental monitoring.	300,000	0.01
Indirect costs (10% of labor, supervision and operating supplies not including yearly interest cost). .	1,569,000	0.05
Fixed costs		
Taxes and insurance (3% of mine cost, $52,214,000).	1,566,000	0.05
Depreciation	5,546,000	0.18
Total	$40,878,000	$1.35

4.6.3 Block Caving Design Using LHD Equipment

The block caving system presented here is in many respects, similar to the one discussed in Section 4.6.2. The major difference lies in the method of ore handling. Ore is loaded into ore cars through transfer raises using LHD equipment. The main advantages of LHD's over slushers are: (1) LHD's have greater mobility and can be easily moved to different loading points in the event of production interruption, and (2) LHD's can handle larger pieces of broken ore, thereby reducing secondary blasting and drawpoint repair costs (Young, 1973).

Haulage, ventilation, and undercut levels are similar in both LHD and slusher designs. In the design using LHD's the haulage level is below the production level with the ventilation level between the two, Figure 4.35. In the discussion that follows, preproduction development, production, ventilation, equipment selection, and production costing are explained in detail except for parts that are identical to the previous design.

4.6.3.1 Preproduction Development

Preproduction development on the main haulage level for block caving using LHD's is identical to the previous design. Drifts are driven using the same equipment with the same cycle times. Differences occur in the development of production and ventilation levels. The main haulage level layout is identical to that of the previous block caving design, Figure 4.28, and requires the same development time, Table 4.45.

Production Level. The production level for the LHD design is 40 feet above the main haulage level. Inclines are driven from the haulage level at each panel location to the production level. The incline to the production level also serves as the intake ventilation access for an entire panel. Five production crosscuts, 14 by 12 feet, are driven parallel to, and directly above the panel drifts, Figure 4.36. Production lines 12 by 12 feet are driven perpendicular to production crosscuts on 100-foot centers, Figure 4.37. Each block has three production lines for a total of 36 finger raises. At the end of each

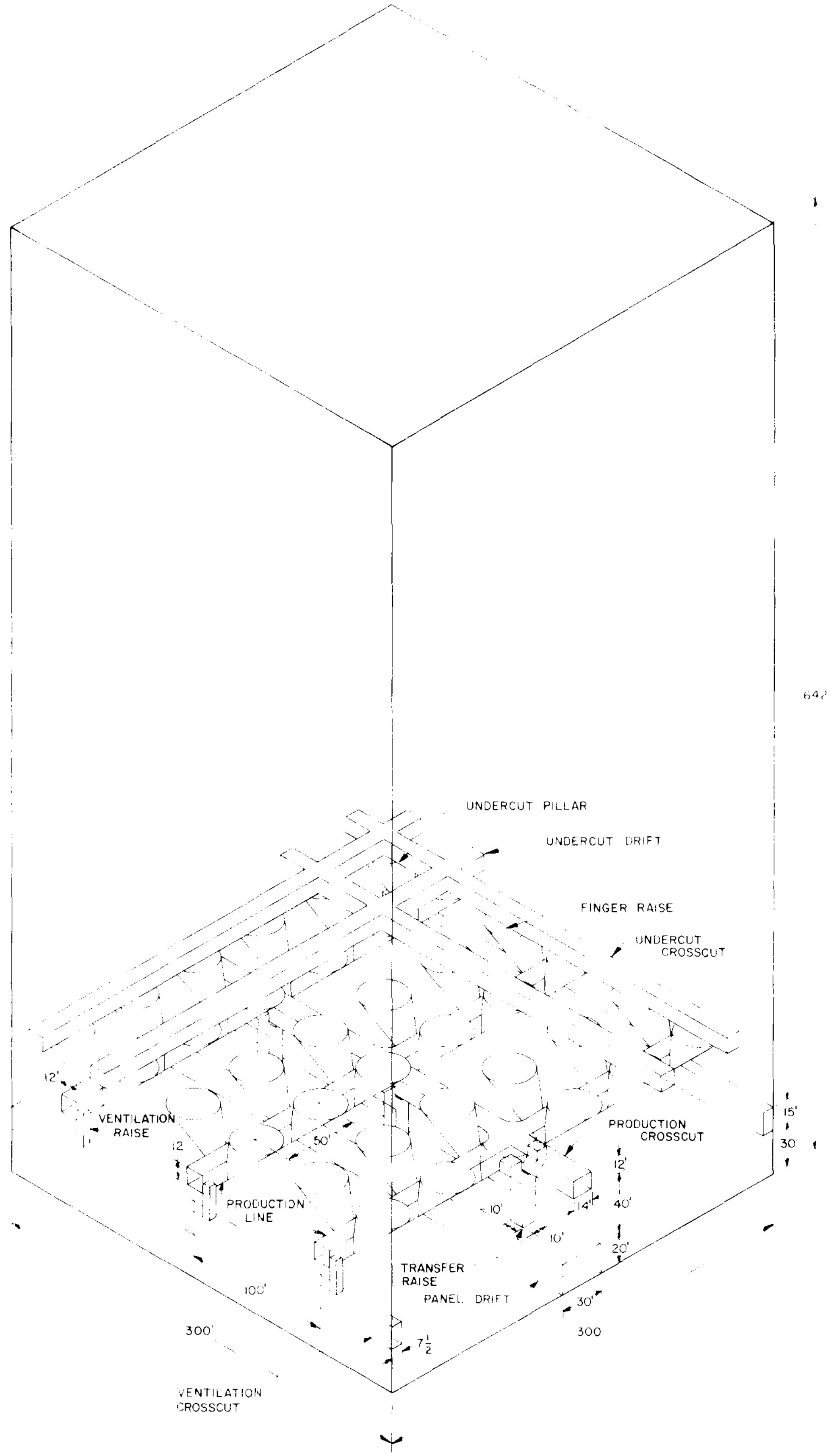


Figure 4.35 Isometric View of One Block, Block Caving with LHD's

Table 4.45 Schedule of Preproduction Work for Block Caving with LHD's

Description of Task	Days	Ore Production (tons)
Develop shaft access drifts	42	154,000
Drive main haulage drifts	150	616,000
Finish shaft access area and machine shop	100	277,000
First ventilation crosscut	111	171,000
2nd, 3rd, 4th, 5th and 6th ventilation crosscuts	55, 55, 55, 55, 55	298,000
Drive 2nd, 3rd, 4th, and 5th panel drifts	55, 55, 55, 55	989,000
Drive 36 production lines	245	370,000
Concrete the production lines	245	
Ramp up to the undercut level and drive undercut drifts	325	319,000
Fan drill pillars	162	
Blast pillars, intermediate production	170	7,225,000
	Total	10,419,000

Days: 30 60 90 120 150 180 210 240 270 300 330 360 390 420 450 480 510 540 570

FULL PRODUCTION

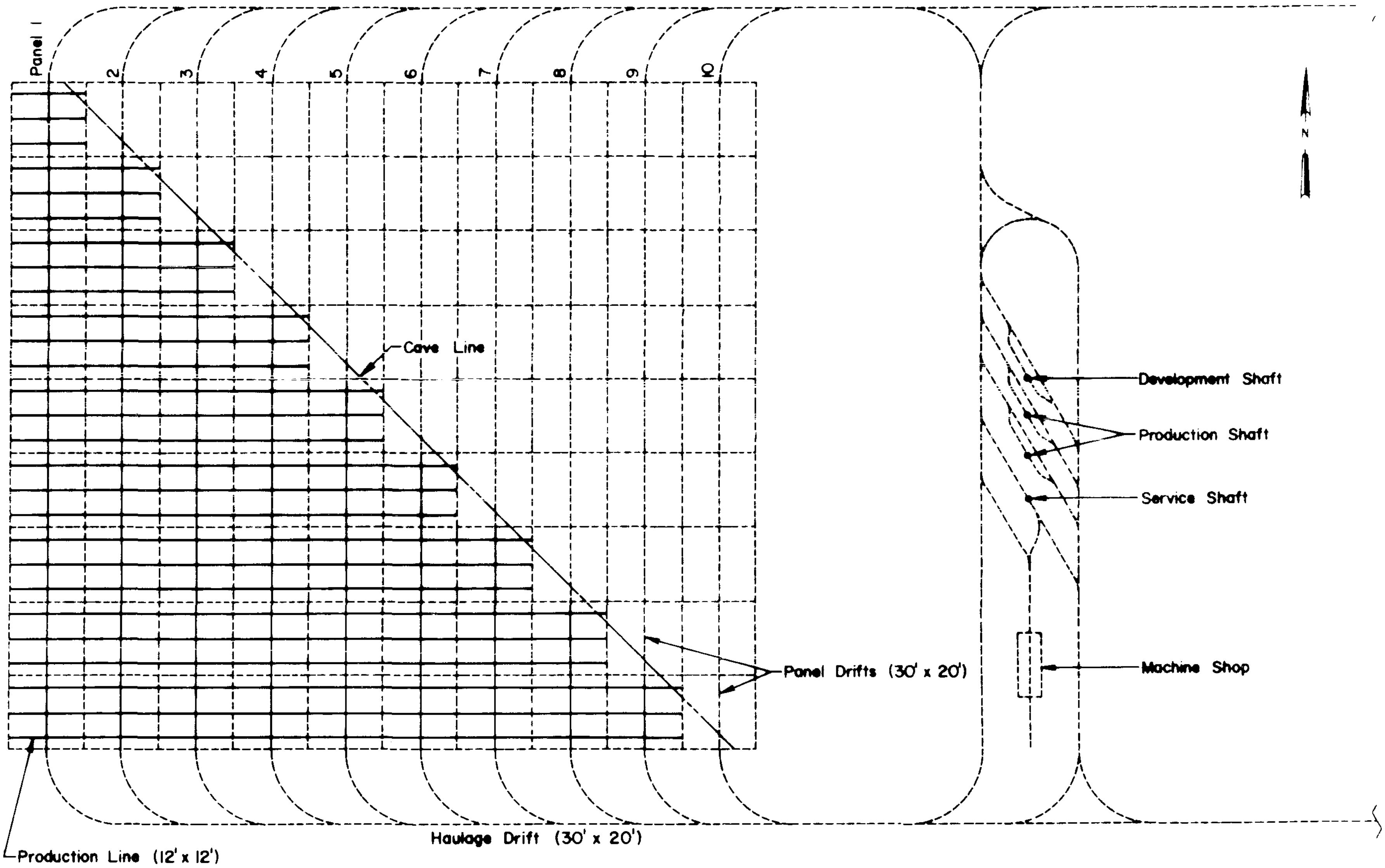


Figure 4.36 Plan View of Production Level, Block Caving with LHD's

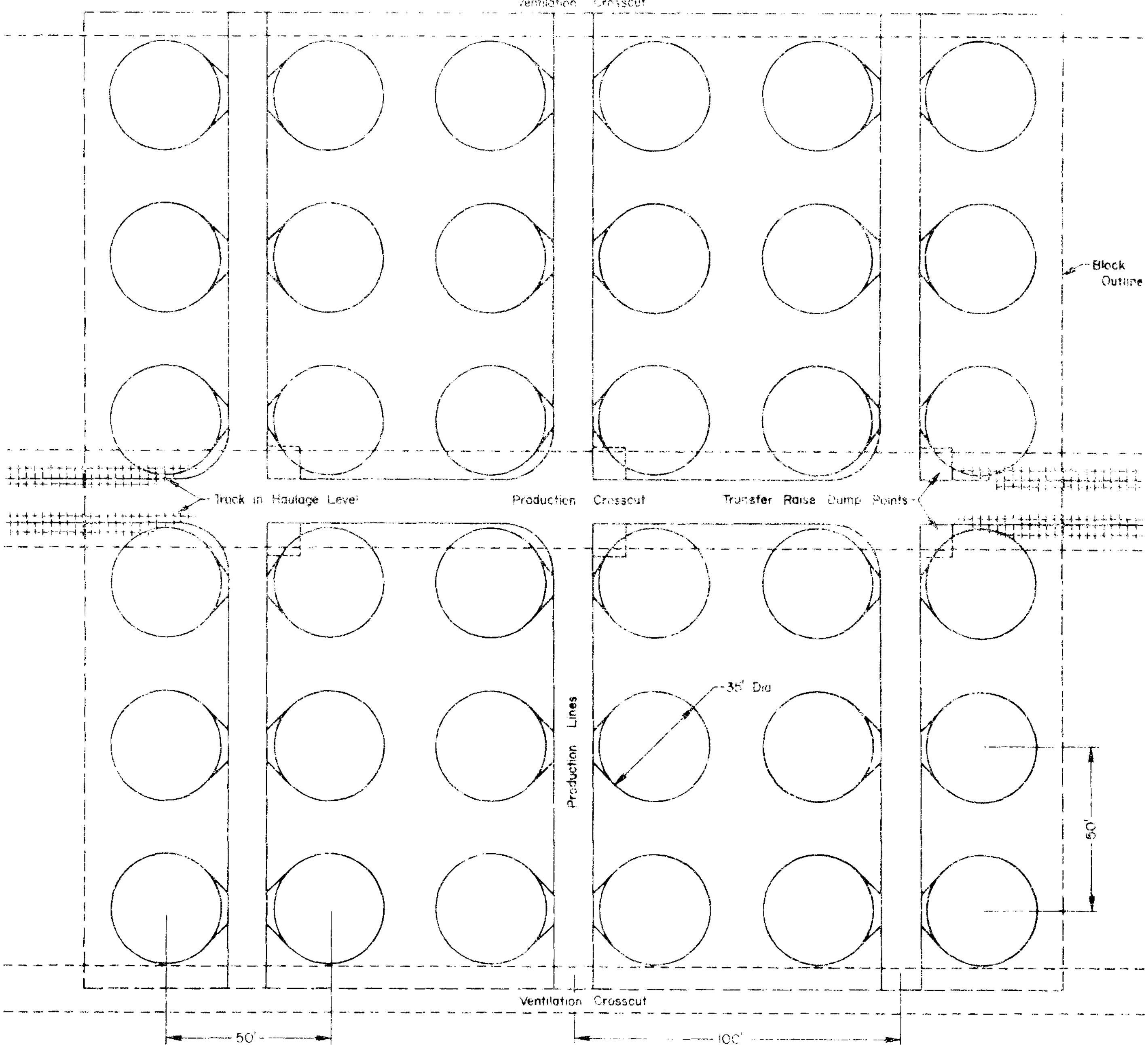


Figure 4.37 Plan View of Production Level, Block within a Panel, Block Caving with LHD's

production line a ventilation raise is bored 15 feet down to an exhaust ventilation crosscut. For full production of 85,000 tpd a total of 12 blocks (432 finger raises) must be developed.

Ore transfer raises (10 by 10 feet) are then driven from the haulage level to the production level, Figure 4.38. Each block has three transfer raises, for a total of six dumping points. Rail mounted, two-boom, pneumatic jumbos are used to drill the ten by ten-foot raises 22 feet high. Then two, eight by eight-foot branches in the raises are drilled using stopers. Each transfer raise is concreted to reduce wear and hang-ups and holds approximately 300 tons of ore. This storage capacity allows the LHD operator to work continually without having to wait for empty ore cars.

Ventilation Level. The ventilation level layout is identical to that shown in Figure 4.30 for the previous block caving design. The major difference is that the ventilation level is above rather than below the main haulage level. Development starts by driving a 30 by 20-foot incline up to the ventilation level and out to panel number one ventilation crosscut. Six, 15 by 15-foot ventilation crosscuts are driven to complete the ventilation circuit necessary for full production. Ventilation drifts are also driven to connect the 16-foot diameter exhaust ventilation shafts.

Undercut Level. Access to the undercut level is by a ten by ten-foot incline from the main haulage level near the shaft pillar. The undercut drift is driven out to the first panel where a room and pillar undercut is started. Crosscuts ten by ten-feet, are then driven on 50-foot centers, leaving 40-foot square pillars, Figure 4.38. Finger raises are holed through and muck dumped to the production level below. When a section has been undercut sufficiently, the finger raises are completed and the pillars drilled and blasted. Development is accomplished using four two-boom jumbos and five cubic yard LHD's.

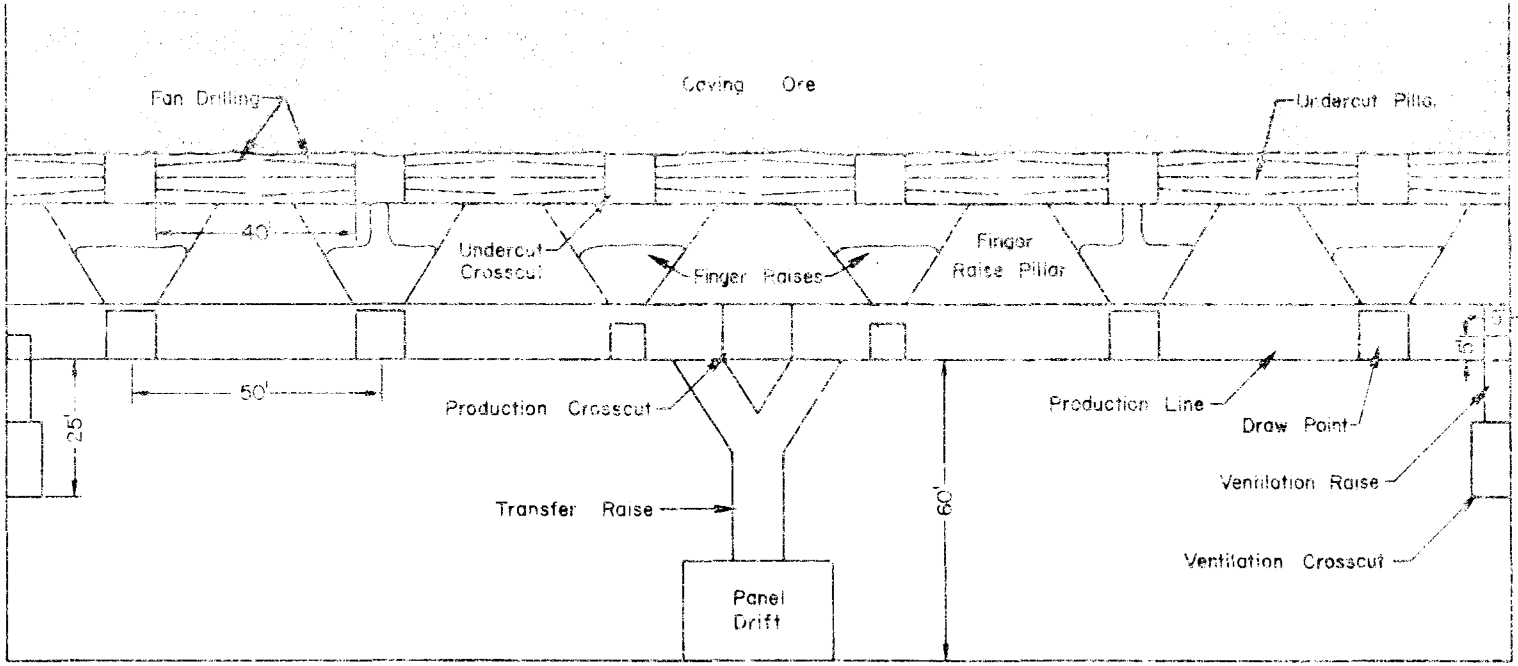


Figure 4.38 Vertical Section, Showing Panel Drift, Transfer Raise, Production Level and Undercut Level, Block Caving with LHD's

4.6.3.2 Production

Full production requirements for each block are identical to that of block caving using slushers. A total of 27, eight cubic yard LHD's per shift haul muck from finger raises and dump into transfer raises. Ore in the transfer raises is then loaded into 80-ton rail cars through hydraulically operated chute gates. Fifty-ton trolley locomotives then haul the ore to the gyratory crushers at the shaft station. Five, 50-ton trolley locomotives must dump ten, ten-car strings per hour. Development muck is hauled using 30-ton locomotives. The drawing life of one block is approximately one year with nine new blocks brought into production each year. Blocks are brought into production retreating towards the shaft area along a 45 degree cave line.

4.6.3.3 Ventilation

Air quantity requirements are computed considering a gassy mine with diesel equipment underground and using only permissible and explosion proof machines. Colorado mining laws (Colorado Bureau of Mines, 1971 and State of Colorado, 1966) require an additional 75 cfm of free air for each brake horsepower of diesel equipment used underground. Preliminary calculations for estimated mine ventilation requirements are presented in Appendix C.

Intake air enters the mine through the production and service shafts and is directed to the production level through inclines at each panel. The air is then circulated through the production crosscuts, regulated through each operating production line, and exits down a vertical raise to the ventilation level. Two 12-foot, axial-vane exhaust fans located in each ventilation shaft, are used to draw the air out of the mine. This ventilation network is designed so that air is not reused once it reaches the production level.

4.6.3.4 Production Equipment Selection

Appendix C contains the calculations used to estimate equipment size, quantities, and cycle times for major equipment items. The capital expenditure tables in the following section on cost analysis summarize the equipment selections.

Evaluation of capital equipment for this large tonnage operation has been done relying almost exclusively on current vendor data. Wherever possible, only that equipment currently manufactured, whether as a full production item or as a prototype, was selected for analysis.

For this initial evaluation operator efficiencies and minor equipment set up times were approximated by considering only five hours of working time per eight hour shift. The mine has been designed on a three-shift, seven-days per week work schedule. For the purpose of including holidays and unexpected work stoppages, a work year was considered to be 355 days. Equipment availabilities are estimated and range from 65 to 90%, depending on use and past performance of similar models.

4.6.3.5 Production Cost

The method of production costing for block caving using LHD's is similar to the format presented in the USBM (Staff, 1972, Katell and Hemingway, 1974). This costing method necessarily assumes that the oil shale mine is the only income source to the corporation; therefore, development costs (including mine access and hoisting) are capitalized. In reality, these costs would most likely be charged to production and deducted as negative cash flow in the overall corporate cash flow determination. Excluded in this analysis are royalty payments, welfare payments, and surface transportation costs. All costs were collected during first quarter, 1975.

Table 4.46 lists the supervisory and hourly personnel needed for the operation of an 85,000 tpd mine. An additional 12% of the hourly manpower requirements are included to account for absenteeism. Figure 4.39 is an example of the anticipated management flow chart for the underground mine only. Table 4.47 is the capital investment summary that includes contingencies, mobilization of capitalized equipment to mine site, consulting, environmental impact statement, mine access, and preproduction development. The total capital investment is estimated at $122,900,000 not

Table 4.46 Manning Table 30.175 MM tpy (Block Caving with LHD's)

Personnel	Total	Annual Cost Per Employee	Annual Cost (260 Workdays)
Salary			
Superintendent	1	$33,000	$33,000
Administrative Superintendent	1	25,000	25,000
General Mine foreman	1	27,000	27,000
Mine foreman	1	22,000	22,000
Shift foreman	3	20,000	60,000
Assistant shift foreman	3	19,000	57,000
Development foreman	15	18,000	270,000
Production foreman	24	18,000	432,000
Chief mechanical foreman	1	21,000	21,000
Chief electrical foreman	1	21,000	21,000
Shift mechanic foreman	3	19,000	57,000
Production mechanic foreman	3	18,000	54,000
Development mechanic foreman	3	18,000	54,000
Shift electrical foreman	3	19,000	57,000
Chief engineer	1	25,000	25,000
Mining engineer	5	20,000	100,000
Draftsman	2	9,600	19,200
Surveyor	3	10,800	32,400
Surveyor helper	3	9,000	27,000
Safety director	1	19,000	19,000
Safety foreman	3	16,500	49,500
Business manager	1	18,000	18,000
Accountant	1	9,600	9,600
Purchasing supervisor	1	14,400	14,400
Warehouse supervisor	1	13,200	13,200
Watchman	4	6,000	24,000
Payroll agent	2	7,500	15,000
Warehouseman	4	7,200	28,800
Stenographer	1	6,600	6,600
Secretary	1	7,800	7,800
Executive secretary	1	9,000	9,000
Mine geologist	1	18,000	18,000
Subtotal	99		$1,626,500

Table 4.46 Manning Table 30.175 MM tpy (Block Caving with LHD's) con't

Personnel	Total	Wages Per Day	Annual Cost (260 Workdays)
Underground			
Driller	62	$51.81	$835,200
Driller helper	62	49.96	805,400
FEL operator	15	50.88	198,400
LHD operator	126	50.88	1,666,800
Car loader	18	49.96	233,800
Locomotive operator	18	50.88	238,100
Scaling and rock bolter	43	52.81	590,400
Powderman	43	52.81	590,400
Trackman	22	50.88	291,000
Ventilation man	7	50.88	92,600
Electrician, first class	7	56.58	103,000
Electrician, second class	7	52.90	96,300
Pipefitter	7	56.58	103,000
Pipefitter helper	7	52.90	96,300
Mechanic, first class	30	56.58	441,300
Mechanic, second class	30	52.90	412,600
Warehouseman	4	52.90	55,000
Concrete man	22	49.96	285,800
Form man	25	49.96	324,700
Supplyman	29	49.96	376,700
Grader operator	4	50.88	52,900
Laborer	14	49.96	181,900
Crusher operator	4	52.90	55,000
Concrete mixer	7	49.96	90,900
Truck driver	8	50.88	105,800
Master mechanic	4	56.58	58,800
Dispatcher	4	50.88	52,900
Hipacker	22	56.58	323,600
Welder	4	51.81	53,900
Subtotal	655		$8,812,500
Outside			
Mechanic	1	$54.76	$14,200
Mechanic helper	1	50.88	13,200
Forklift operator	4	49.96	52,000
Lampman	4	47.15	49,000
Hoistman	8	51.70	107,500
Janitor	4	49.15	49,000
Cageman	4	49.77	51,800
Subtotal	26		$336,700
Contingency for absenteeism (12%)	82		$1,098,000
Total labor and supervision	862		$11,873,700

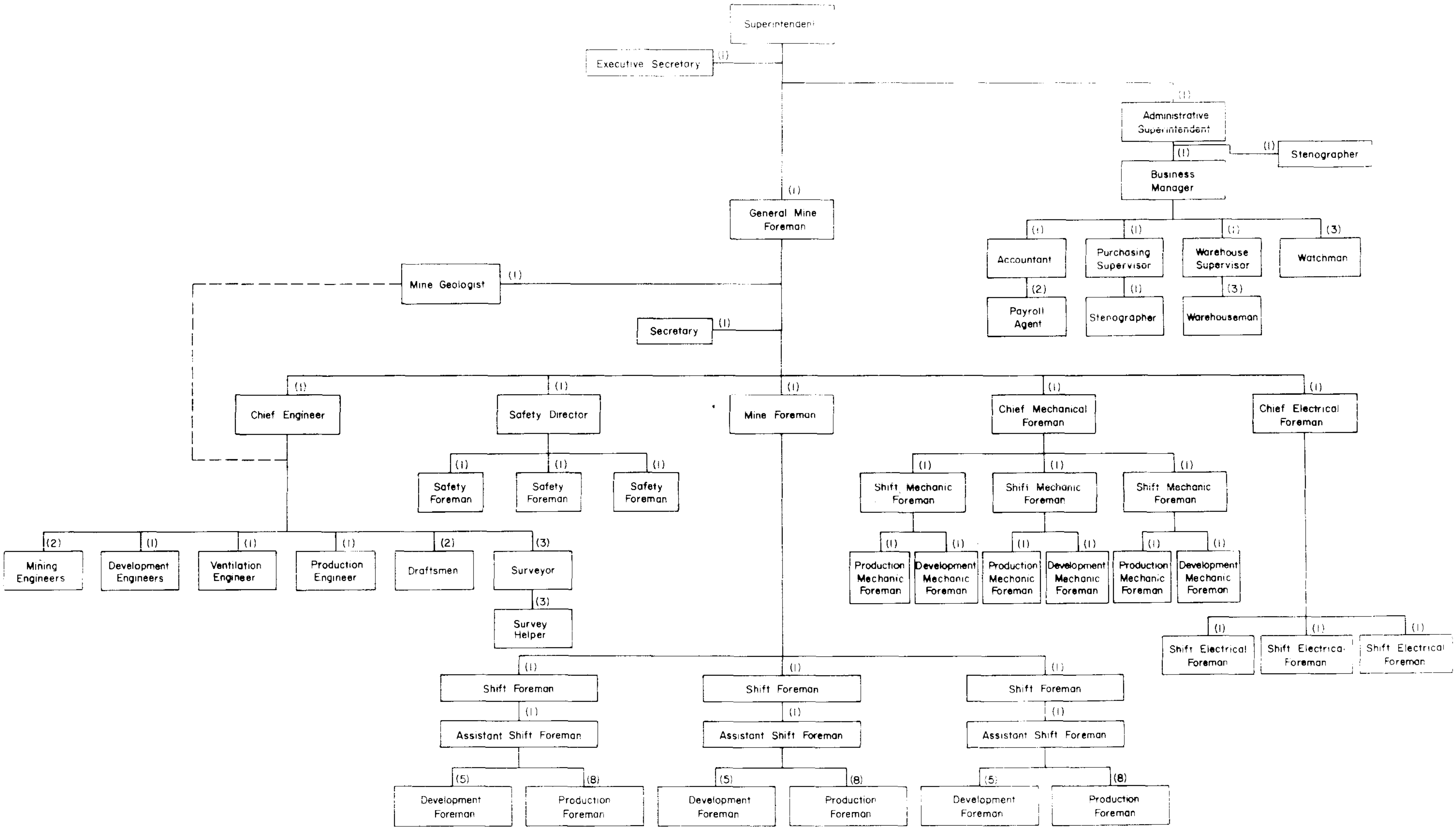


Figure 4.39 Management Flow Chart, 30.175 MM tpy, Block Caving with LHD's

Table 4.47 Capital Investment Summary, 30.175 MM tpy (Block Caving with LHD's)

ITEM	Quantity	Cost Per Unit	Total Cost
Two boom hydraulic drill jumbo	6	$ 310,000	$1,860,000
Two boom medium drill jumbo	6	62,000	372,000
Two boom small drill jumbo	6	38,000	228,000
Two boom hydraulic fan drill	6	300,000	1,800,000
FEL (8 yd^3)	6	160,000	960,000
LHD (5 yd^3)	6	105,000	630,000
LHD (8 yd^3)	39	145,450	5,673,000
Stoper	4	2,700	11,000
50-ton trolley locomotive	6	250,000	1,500,000
80-ton rail car (air brakes)	60	23,000	1,380,000
30-ton trolley locomotive	5	190,000	950,000
8-ton battery locomotive	2	75,000	150,000
Flatcar and Supply car	20	10,000	200,000
Concrete vats	20	20,000	400,000
Two drill rock bolt machine	8	150,000	1,200,000
Personnel carrier	6	41,500	249,000
Powder and ANFO loading truck	4	28,000	112,000
Lubrication and fuel truck	4	30,000	120,000
Service vehicle	10	30,000	300,000
Battery charing equipment	2	10,000	20,000
Road grader	2	60,000	120,000
Fire chemical truck	4	25,000	100,000
Underground machine shop	1	1,000,000	1,000,000
Mine rail (115 lb)	260,000 ft	$4.8/foot	1,248,000
Track plates	65,000	$2.15	140,000
Treated ties	65,000	$6.64	432,000
Lamp (including accessories)	860	85	73,000
Water truck	2	30,000	60,000
Bathhouse, office and lamphouse	1	2,000,000	2,000,000
Warehouse and Supply yard	1	130,000	130,000
Forklift	1	25,000	25,000
Substation and distributor	1	900,000	900,000
Transformer and rectifier	-	-	148,000
High voltage cable (8KV-4°)	30,000 ft	$7.2/foot	216,000
Methane detector	40	435	17,000
Portable fire extinguisher (20 lb)	20	42	1,000
Primary ventilation fan	4	115,000	460,000
Auxiliary fan	10	7,400	74,000
Exhaust fan (machine shop)	2	8,000	16,000
Self rescuer	800	43	35,000
Telephone (page phones)	10	450	5,000
Stretcher set	10	235	3,000
First aid kit	30	30	1,000
Water pumps (10,000 gpm total)	16	33,000	528,000
Compressor	5	20,000	100,000
Tow truck	1	30,000	30,000

Table 4.47 Capital Investment Summary 30.175 MM tpy (Block Caving with LHD's) con't

ITEM	Quantity	Cost Per Unit	Total Cost
60-inch gyratory crusher	2	1,300,000	2,600,000
Track maintenance machine	1	50,000	50,000
Trolley wire	130,000 ft	$0.68/foot	88,000
Two boom fan drill	2	73,200	146,000
Compressor (350 hp)	1	50,000	50,000
Total direct cost			$28,911,000
Contingencies (9% of total direct cost)			2,602,000
Subtotal			31,513,000
Mobilization charge (2% of line above)			630,000
Total Construction			32,143,000
Consulting engineering fees, overhead and administration (6% of line above)			1,929,000
Subtotal			34,072,000
Environmental Impact Statement			2,000,000
* Mine access (included are ventilation, production, and service shafts, two production shaft double drum hoist, one service shaft double drum hoist, one development double drum hoist, two crusher rooms and two rotary car dumps, two surge bins)			50,538,000
Subtotal			86,610,000
* Preproduction cost			36,262,000
Net capital investment			$122,872,000
Value of oil in 10,419,000 tons of oil shale stockpiled or retorted during preproduction development (assumes $11/bbl)			$54,576,000

* Assumes capitalization.

including the value of oil in the development are estimated to be $54,600,000.

Table 4.48 is a straight-line depreciation schedule for the equipment listed in Table 4.47. Table 4.49 is the estimated yearly interest cost on the money borrowed to finance the capital equipment. In this table interest rates from eight to ten percent are considered; however, a rate of nine percent is used in the cost summary. Notice that the mine development costs are capitalized over a period of 30 years. Power and water consumption, Table 4.50 are estimated from vendor data, and the electric rate as quoted by a local Colorado public utility for the Central Piceance Creek Basin. Water cost is neglected because it is assumed more than enough water will be available from the mine itself. Table 4.51 is the estimated cost of preproduction, including the cost of interest on capital. The cost per ton figure is determined by dividing the total cost by the amount of preproduction development ore (10,419,000 tons).

Table 4.52 is a summary of the estimated annual production cost for an 85,000 tpd mining operation. It must be understood that this cost includes only mining costs and does not include processing costs. Also included are yearly interest, reclamation, exploration and environmental monitoring costs. The total estimated cost per ton is $1.31.

4.6.4 Additional Cost of Inducing Caving

In the computations of production costs for each caving design it was assumed that the ore would cave after the undercutting of a block was completed. In the case where the ore is difficult to cave, the caving must be induced by weakening the block boundaries. The design presented here includes driving an incline from the haulage level, a vertical distance of 150 feet above the undercut level. A ten-foot square drift is then driven parallel to the outer boundary of the mine, Figure 4.40. The distance between the outer boundary of the blocks and the drift is maintained at 50 feet. Ten by ten-foot crosscuts, perpendicular to the boundary drifts, are driven to the block boundary and used as fan drilling stations. A fan drill jumbo drills a ring of

Table 4.14 Depreciation Schedule, 30.175 MM tpy (Blcok Caving with LHD's)

ITEM	Straight-line depreciation (years)	Yearly charge
Two boom hydraulic drill jumbo	8	$232.500
Two boom medium drill jumbo	8	46,500
Two boom small drill jumbo	8	28,500
Two boom hydraulic fan drill	8	225,000
FEL (3 yd.)	8	120,000
LHD (5 yd.)	8	78,800
LHD (8 yd.)	8	709,100
Stoper	8	1,400
50-ton trolley locomotive	15	100,000
30-ton trolley locomotive	15	63,300
8-ton battery locomotive	15	10,000
80-ton rail car (air brakes)	10	138,000
Flatcar and Supply car	10	20,000
Concrete vats	10	40,000
Two drill rock bolt machine	10	120,000
Mine safety equipment	5	9,200
Powder and ANFO loading truck	10	11,200
Lubrication and fuel truck	10	12,000
Service vehicle	10	30,000
Underground machine shop	30	33,000
Track and trolley material	30	63,700
Battery charging equipment	10	2.000
Road grader	8	15,000
Water truck	10	6,000
Bathhouse office and lamphouse	30	66,700
Warehouse and supply yard	30	4,300
Forklift	10	2,500
Substation and distribution	30	30,000
High voltage cable (8KV-4°)	30	7,000
Primary ventilation fan	30	15,400
Exhaust fan (machine shop)	30	500
Auxiliary fan	10	7,400
Water pump (10,000 gpm total)	15	35,200
Air compressor	10	10,000
Tow truck	10	3,000
60-inch gyratory crusher	30	86,700
Track maintenance machine	15	3,300
Transformer and rectifier	15	9,900
Two boom fan drill	8	18,300
Environmental impact statement	30	66,700
Fire chemical truck	10	10,000
Mine access	30	1,684,600
Preproduction development	30	1,207,900
Compressor (350 hp)	10	5,000
Contingencies, mobilization and engineering	30	172,000
	Total	$5,561,600

Table 4.49 Estimated Yearly Interest Cost on Capitalized Items, 30.175 MM tpy (Block Caving with LHD's)

Depreciable Life (yrs)	5	8	10	15	30	Total
Capital Required	46,000	11,800,800	4,171,000	3,325,500	103,155,000	122,498,300
Interest Charge						
10%	2,800	663,800	229,400	177,400	5,329,700	6,403,100
9.5%	2,600	630,600	217,900	168,500	5,063,200	6,082,800
9%	2,500	597,400	206,500	159,600	4,796,700	5,762,700
8.5%	2,300	564,200	195,000	150,800	4,530,200	5,442,500
8%	2,200	531,000	183,500	141,900	4,263,700	5,122,300

Table 4.60 Power and Water Cost 30.175 MM tpy (Block Caving with LHD's)

Number of Units	Operation	H.P. per Unit	H.P Total load	Hrs. per day full load	KW total load	Total KW requirements	Total* water (gpd)
4	Two boom hydraulic drill jumbo	135	540	15	403	6,050	54,000
4	Two boom hydraulic fan drill	135	540	15	403	6,050	54,000
4	Two boom medium drill jumbo	-	-	15	-	-	40,000
4	Two boom small drill jumbo	-	-	15	-	-	30,000
6	Rock bolting machine	-	-	18	-	-	33,000
4	Primary ventilation fan	1,350	5,400	24	4,027	96,650	
10	Auxiliary fan	150	1,500	24	1,120	26,880	
2	Fan (machine shop)	105	210	24	157	3,770	
5	50-ton trolley locomotive	720	3,600	15	2,685	40,280	
4	30-ton trolley locomotive	380	1,520	15	1,133	17,000	
2	Double drum hoist, production	10,000	20,000	20	14,914	298,280	
2	Double drum hoist, service and production	5,000	10,000	20	7,457	149,140	
2	60-inch gyratory crusher	700	1,400	20	1,044	20,880	12,000
8	Compressors (1000 cfm)	100	800	15	597	8,960	
12	Pumps	425	5,100	24	3,805	91,320	
2	Rotary dump	60	120	20	90	1,800	
1	Compressor	350	350	15	261	3,920	
	Miscellaneous (Underground)	-	-	-	1,000	12,000	12,000
	Miscellaneous (Surface)	-	-	-	1,000	12,000	6,000
					Total	794,980	241,000

Power Cost = $0.013 x 794,980 x 355 = $3,587,300

* Water assumed to be provided by site wells.

Table 4.51 Estimated Preproduction Cost, 30.175 MM tpy (Block Caving with LHD's)

ITEM	Total Cost	Cost/Ton*
Total labor and supervision	$11,000,000	$1.06
Operating supplies		
Machine parts	620,000	
Drill bits	94,000	
Rock bolts, shells and plates . .	264,000	
Fuel, lubrication and tires . . .	543,000	
ANFO, wire and caps	366,000	
Drill steel and couplings	83,000	
Concrete	600,000	
Subtotal	2,570,000	
Operating contingencies (5% of line above)	128,000	
Interest on capitalized equipment (550 days)	6,109,000	
Subtotal	8,807,000	0.85
Power	2,301,000	0.22
Payroll overhead (35%)	3,850,000	0.37
Indirect costs (10% of labor, supervision and operating supplies)	1,370,000	0.13
Fixed Cost		
Depreciation (550 days)	6,560,000	0.63
Taxes and insurance (3% of 52,508,000 for 550 days)	2,374,000	0.23
Total .	$36,262,000	$3.49

* Total tons = 10,419,000

Table 4.52 Estimated Annual Production Cost, 30.175 MM tpy (Block Caving with LHD's)

ITEM	Annual Cost	Cost/Ton
Direct costs		
Labor and supervision	$11,874,000	$0.39
Operating supplies		
Machine parts	950,000	
Lubrication, fuel and tires	1,413,000	
Rock bolts, shells and plates . . .	202,000	
Drill bits	70,000	
ANFO, caps and wire	250,000	
Drill steel and couplings	85,000	
Concrete	420,000	
Subtotal . .	3,390,000	
Operating contingencies (5% of line above)	170,000	
Yearly interest cost (9%)	5,763,000	
Subtotal . .	9,323,000	0.31
Power	3,687,000	0.12
Reclamation	1,000,000	0.03
Payroll overhead (35%)	4,156,000	0.14
Exploration	500,000	0.02
Environmental monitoring	300,000	0.01
Indirect costs (10% of labor, supervision and operating supplies not including yearly interest cost)	1,543,000	0.05
Fixed costs		
Taxes and insurance (3% of mine cost, $52,508,000).	1,575,000	0.05
Depreciation	5,562,000	0.19
Total	$39,520,000	$1.31

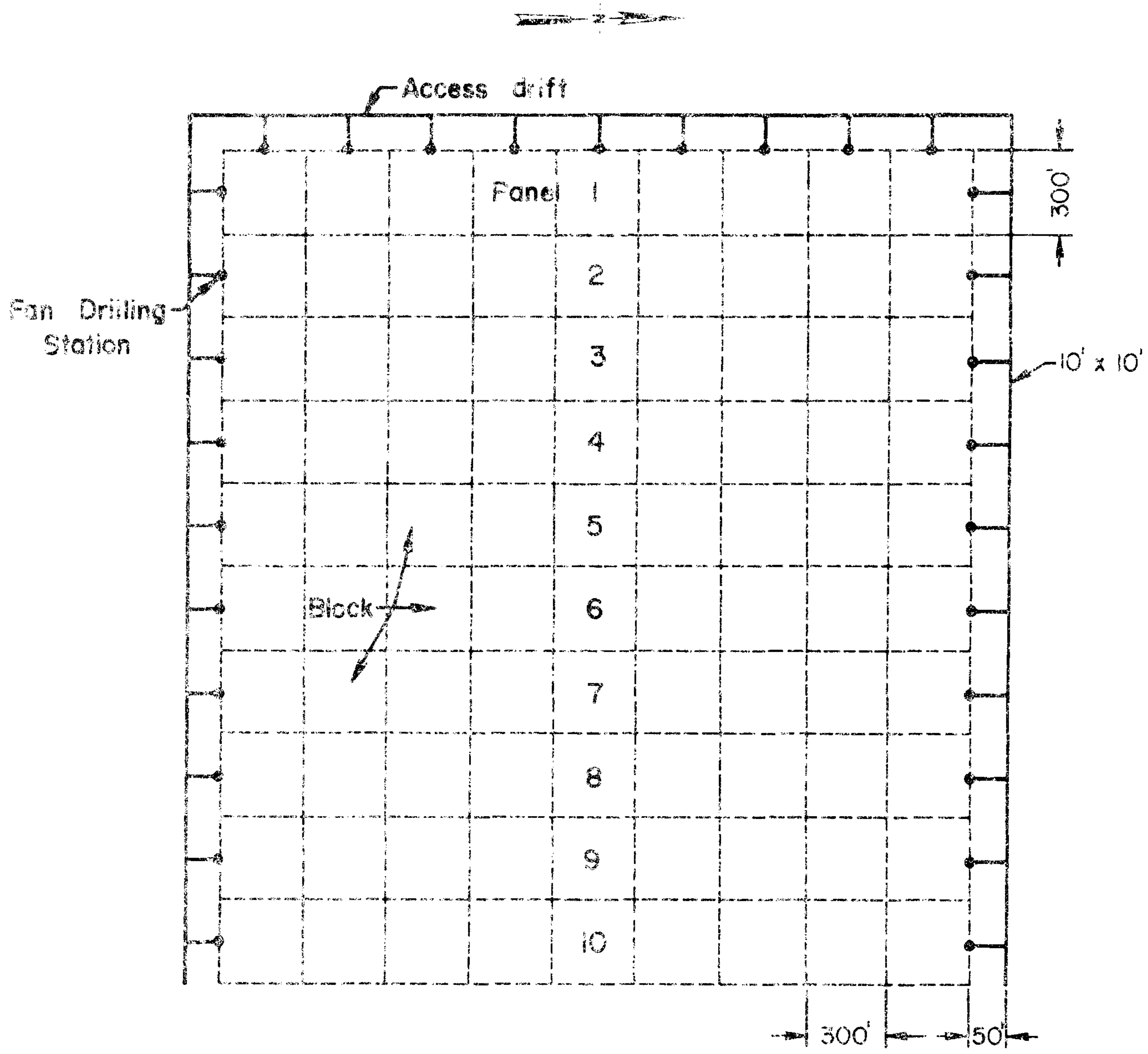


Figure 4.40 Plan View of Access Drift and Fan Drilling Stations for Induced Caving

16, three-inch diameter holes in a vertical plane parallel to the block boundary. It is assumed that when two sides of a block are fan drilled and blasted, together with undercutting, caving is induced. Boundary weakening is provided for all peripheral blocks.

Additional costs include $750,000 for drilling equipment, $500,000 per year for additional personnel, and $75,000 per year for operating supplies. The total additional production cost for this method is approximately $0.04 per ton.

4.7 ENVIRONMENTAL IMPACT

The environmental impact of oil shale mining in the Piceance Creek Basin is a controversial and sometimes emotional subject. In the discussion that follows an attempt is made to view the problems arising from mining from an environmental standpoint. The amount of data available in the region of selected site 2 is almost non-existent; therefore, a number of assumptions and generalizations are made.

The environmental analysis is broken into four general sections: (1) the existing environment, (2) environmental regulations, (3) mining impacts, and (4) cost estimates for environmental programs. Each of these sections are further subdivided to provide a reasonably comprehensive overview of the environmental problems expected due to the underground mining of oil shale resources.

4.7.1 Existing Environment

4.7.1.1 Physical Environment

Topography and Soils. The elevation of the study area varies from about 6,100 to 6,800 feet. It is characterized by broken terrain divided by numerous gullies and canyons with a single main drainage, Ryan Gulch. Draining approximately southwest to northeast, Ryan Gulch is intersected by numerous smaller canyons having slopes that vary from moderate to very steep.

Soils in the study area are young and poorly developed. The soil mantle varies from none at all through immature, lithic soils, to deep alluvial soils found in the floor of Ryan Gulch. These deep soils are derived from sandstone and shale colluvium eroded from adjacent slopes. As a result, they are young and largely mineral in composition.

Upland soils are thin to medium depth, with deeper pockets of sandy, aeolian soils occurring on lower portions of many of the area's ridges.

Climatology and Meteorology. Precipitation in the study area has not been measured but is probably about 15 to 17 inches annually (Harbert and Berg, 1974), about half of which is believed to occur as snow. Occasional thunderstorms during late summer can be severe enough to cause flash flooding along the major drainages, such as Ryan Gulch. Air temperatures are characterized by extremes varying from over 100°F during the summer months, to -40°F, or lower, during winter.

Airflow is strongly influenced by the topography. Local valley flow circulation patterns are dependent on diurnal variations typically resulting in upslope breezes during the daytime and downslope drainage winds at night. However, in the presence of significant upper airflow, these valley flow patterns can be complex and difficult to predict. In the broader valley bottoms, temperature inversions may be expected, especially during the winter months.

Surface Water. Surface water is scarce in the study area, as it is in the rest of the Piceance Creek Basin. Ryan Gulch displays only intermittent surface flow, primarily during spring runoff. However, as mentioned previously, occasional severe thunderstorms can produce significant flows for short periods of time during late summer. Although no data on Ryan Gulch flows have been found, the erratic nature of stream flow under such conditions is inferred by data on Piceance Creek flow. According to Coffin, Welder, and Glanzman (1971), the Piceance Creek (below Ryan Gulch) had an average flow of 12.5 cfs with extremes of 0.80 to 400 cfs.

4.7.1.2 Biological Environment

Vegetation. Vegetation in the area consists of pinyon-juniper woodland, (Pinus edulis, Juniperus Osteosperma) mixed mountain shrub communities, and grassland in some of the primary drainage bottoms. Sagebrush, (Artemisia Spp.) a component of most communities on a variety of sites, may also occur as the characteristic species, especially where overgrazing has historically occurred. As in most semi-arid ecosystems, vegetation distribution is strongly influenced by slope and aspect.

Aspect is important because of its influence on available soil moisture. North-facing slopes retain more snow for percolation into the soil as a result of slower melting and less surface runoff than occurs on south-facing slopes.

Detailed vegetation inventories have not been conducted on the study area. Therefore, information on the occurrence and distribution of species or even specific plant communities is not available. Previous studies (Ward, Slauson, and Dix, 1974) have suggested as many as 35 potential plant communities may be expected in the Piceance Basin, but that 15 would probably account for most of the area.

Although the major vegetation types listed above are sufficient to convey a general impression of the vegetational character, they fall far short of describing the ecological capability of the site in terms of wildlife habitats, land reclamation requirements, and environmental damage mitigation requirements. Additional detailed information on the area vegetation would be required for a more definitive estimate of expected impacts from the proposed mining operations.

Wildlife. Probably the single most important wildlife species of the Piceance Creek Basin is the mule deer (Odocoileus Hemionus). The study area lies near the center of the winter range area of the White River mule deer herd, one of the largest in North America. The animals migrate annually from higher elevation summer range areas surrounding the basin to spend the winter in the Central Piceance Creek Basin, where browse vegetation is available and not covered by deep snow. Occasionally, even this becomes inaccessible to the deer because of unusually deep snow cover, causing large numbers of them to die from starvation.

There is insufficient information at present to indicate precisely which portions of the Central Piceance Creek Basin are most critical to deer herd survival, or what compensation measures might be effective in minimizing the impact of reduction in available range. However, it is clear that any significant reduction in the amount or quality of available

Category 2 - Areas where very limited air quality degradation will be permitted,

Category 3 - Urban areas where existing pollution is less than the secondary standard, allowing emissions up to the standard.

At present, the Colorado oil shale region is likely to be placed in category 2, if that designation will not impede oil shale development. If this category proves too restrictive, the region will probably be classified into category 3. These categories will be concerned only with particulates and oxides levels.

Of most importance to the mining, considered in this report, is the limitation on particulate matter, since this constraint covers atmospheric dust which may be expected from shale hauling and handling. The only other pollutant categories pertinent to mining are nitrogen oxides and hydrocarbons which may be produced by blasting. Although probably insufficient to constitute violations, the gases produced by underground blasting should be considered in the design of mine ventilation systems. Nitrous oxides and hydrocarbons could also be a problem along haul roads if truck hauling or other heavy vehicular traffic is anticipated.

The possibility of natural gas release by mining must also be considered. Depending on the amounts released, such gaseous hydrocarbons could be subject to control under federal or state emissions regulations.

4.7.2.2 Water Quality Regulations

The Colorado Water Quality Control Commission, by authority of the Colorado Water Quality Control Act of 1973, has issued standards for water quality and classifications of state water bodies according to the specific historic uses and future desired uses of each classified stream or lake. The Commission's regulations also carry federal authority by virtue of EPA approval of Colorado's water quality program, under the Federal Water Pollution Control Amendments of 1972.

The Commission has not classified Piceance Creek, the main drainage of the study area considered in this report; however, it has classified the White River into which Piceance Creek flows. Above the White River confluence with Piceance Creek it is classified as B_1, below as B_2. Both classes B_1 and B_2 designate waters as "suitable for all purposes for which raw water is customarily used, except primary contact recreation, such as swimming and water skiing." The only differences between classes B_1 and B_2 are in the water temperature changes resulting from effluent discharge and in a slightly lower dissolved oxygen content required for B_2. In addition, there are basic standards applicable to all state waters, whether classified or not. Among these is the Commission's proposition "...to maintain salinity concentrations at or below present levels...," until such time as sufficient data and salinity-reduction technology are available to permit the establishment of quantitative standards.

In applying the Colorado regulations, the state has authority to issue discharge permits to private developments where the discharge is in compliance with both state and federal law. If permits are issued which do not comply with the law, they are subject to veto by the EPA. The EPA may also veto permits which will result in adverse effects on water quality in another state. In the case of Federal developments, only the EPA can issue permits.

4.7.2.3 National Environmental Policy Act

Any federal action of the magnitude of the proposed oil shale development will, of course, be subject to provisions of the National Environmental Policy Act (NEPA). Therefore, public disclosure of the general development plans and submission of an environmental impact statement will likely be required. Although NEPA alone does not constitute physical, statutory constraints on development, it does provide legal means by which other federal agencies, state agencies, environmental groups and the general public can object to or otherwise exert influence on project design, timing, and even its existence. Considering the magnitude of the mining effort envisioned and its potential for ecological disturbance, significant public opposition to its commencement should be expected.

4.7.2.4 Minerals Leasing

Whether the proposed mining operations are conducted by private industry alone, or in combination with government agencies, the land area involved is public and will therefore be subject to leasing by the Interior Department, under the 1920 Mineral Leasing Act. If the Prototype Oil Shale Leasing Program and announced U.S.D.I. intentions in regard to new coal leasing policies are indicative, substantial environmental stipulations will be a part of any such instrument. Therefore, such requirements should be anticipated.

4.7.3 Mining Impacts

Under the broad heading "environmental impacts" are included many diverse effects, both positive and negative. The degree of some impacts is directly related to the mining method while some will be of equal magnitude for all methods. Due to the preliminary nature of this report, only general categories of impact are identified. Quantification and detail must await more comprehensive evaluation of the particular mining method selected.

As far as environmental impacts are concerned, the major difference between the mining methods considered in this report is the amount of land subsidence involved and the time period over which it will occur. Since both these factors relate to mining method and technology, associated environmental impacts will be discussed in the category, "technology-dependent" impacts. "Technology-independent" impacts are those which will be essentially the same, regardless of the underground mining approach.

4.7.3.1 Mining Technology Dependent Impacts

Definitions and Assumptions. In discussing technology-dependent impacts, three categories of subsidence are considered, as defined below:

Light Subsidence -	involves maximum backfilling of mined-out areas; subsidence restricted to 15 to 20 percent of mined room vertical dimension,
Moderate subsidence -	less than maximum backfilling; intermediate between light and severe subsidence,
Severe subsidence -	no backfilling; cliff formation around perimeter; block-faulted to rubble center; subsidence depth 80 percent of room depth.

An important factor in regard to the degree of subsidence damage is the rate at which subsidence takes place. Although precise estimates of subsidence rates are not yet available, this discussion assumes no subsidence would be apparent for four to five years after the start of mine development. Furthermore, that once commenced, subsidence would be gradual, never catastrophic, over the life of the mine. Since the extent of horizontal mining area could be essentially unlimited except for broad constraints on resource distribution, a single subsidence unit of one square mile is used for purposes of discussion. This unit is a land area which could be mined out and subsided in its entirety. Most of the environmental impacts from larger or smaller units can be extrapolated, within reasonable limits.

To compute vertical displacement of the land surface, a mined cavity vertical dimension of 1000 feet is assumed. Under this assumption, an 800-foot (80%) surface drop would constitute severe subsidence as defined above. Likewise, a 150 to 200-foot (15 to 20%) drop would be considered as light subsidence per square mile, average winter carrying capacity, the amount of forage loss would be equivalent to 14 to 54 animals. Assuming the mined area falls in the most critical winter range area, the true amount of range loss would probably fall somewhere between these two extremes. For a total herd size estimated to be 30,000 to 100,000 head (Johnson and Alden, 1974), such potential losses should not be significant. Under conditions of severe subsidence, the loss of habitat could total almost 1,500 acres.

Under light subsidence, other terrestrial wildlife besides deer would probably not suffer seriously. Although some reduction in habitat would likely occur, it would not be potentially critical to population survival. Most species are expected to reestablish in areas adjacent to disturbed land where the habitat would remain available. Although individuals of some less mobile species may well be exterminated, such losses would not be detrimental to the species in general, except where rare or endangered species are threatened.

Severe subsidence would result in a general loss of wildlife habitat for most species which formerly inhabited the disturbed area.

Land Use. The impact of the proposed program on land use in the study area would be profound under conditions of severe subsidence. Not only would the recreational resources be destroyed, grazing could also be lost for a long period, well beyond the end of mining. However, under light subsidence, neither of these land uses would necessarily be lost in the long term.

4.7.3.2 Mining Technology Independent Impacts

Air Quality. The major detrimental influence on air quality will be dust. Since dust is an air contaminant regulated by law, mine and associated facilities should be designed to comply with state and federal regulations in this regard.

Hauling and dumping of development rock is a potential source for considerable dust generation. During mine operations, shale hauling and handling will require dust control procedures. If conveyor systems are used to move the shale, they should be enclosed for dust suppression as well as weather protection.

Gas produced by blasting and released natural gas may also require some degree of control. Depending on the amounts of gas produced or released, the exhaust from mine ventilation systems may require filtration before release to the atmosphere. A possible alternative in the case of natural gas is to draw it off in advance of mining for commerical production, if they are of adequate quality.

Water Quality. The estimated water requirement for the project is approximately 200 gpm, average demand. This water is used in mining operations for drilling, dust suppression, and perhaps for hydraulic backfilling. Of this amount, approximately 100 gpm is expected to be recyclable. The remaining 100 gpm is lost to leakage, evaporation, and binding in the spent shale slurry used for backfilling.

According to estimates all water requirements can be filled by ground water from mine dewatering, even though the ground water may be high in soluble salt concentration. Therefore, no water from external sources is expected to be required. On the contrary, it appears likely that a substantial excess of water from mine dewatering will be generated, which will require disposal. If this water is to be discharged to surface drainages, state and federal water pollution regulations must be considered. If, as estimated in this report, 2,000 to 5,000 gpm of saline water is produced by mine dewatering, 1,800 to 4,800 gpm will require disposal even if maximum use of the water is made in mining operations. A small portion of this excess may be usable in surface spent shale disposal. The impact of this amount of discharge on surface water quality is probably too great for such an alternative to receive serious consideration, if groundwater in the mine area does prove to be highly saline. Desalination is an alternative which could probably be environmentally satisfactory, perhaps even desirable since the process

Assuming adequate technology for establishing vegetation on spent shale disposal piles can be developed, the question of where such disposal areas should be located must be considered. At present, two opposite approaches are being considered by companies developing oil shale. One company envisions placing the material in a natural drainage, slowly filling it from one end, and establishing vegetation stepwise, in a longitudinal manner down the valley as it is filled. A shale/water retention dam is constructed at the foot of the gulch with permanent drainage tunnels constructed underneath the shale pile. This approach permits natural, uncontaminated runoff to pass under the disposal area into the natural drainage below the dam. Shale contaminated (salty) water draining over and through the shale pile will be held by the dam. The other approach, being considered for prototype lease tract C-a by the Rio Blanco Project, considers piling the spent shale on a mesa. The methods for controlling contaminated runoff and leach water have not yet been determined. Both project approaches may involve placement of soil over the spent shale to provide a better plant growth medium.

For the project discussed in this report, either of the approaches discussed above may be feasible. However, there is another possibility presented by the land subsidence which will occur: placement of spent shale in the subsided area. Depending on the degree of subsidence, a disposal cavity estimated at 200 to 800 feet deep over a one square mile area would be created by mining one subsidence unit, as previously defined. To return this disposal cavity to ground level would require 200 to 800 million cubic yards of material. For a daily spent shale production rate of 35,000 yd^3 assuming no backfilling underground, 16 years of spent shale production would be required to fill the 800-foot deep cavity. For a 200-foot depth, with maximum mine backfilling, and excess spent shale production of 10,500 yd^3 per day, approximately 52 years of production would fill the cavity. Although attractive at first consideration, such a scheme suffers some practical shortcomings. Unless a subsidence unit was prepared in advance by disposal of spent shale elsewhere, no disposal cavity would be ready as it was needed during mining. The problem might be solved by systematic programming of the mining effort to create small, sequentially subsided areas which could be used for disposal following subsidence of each previous mining unit. Another possibility would be to pile the spent shale directly on top of the land area which is expected to subside in subsequent years. Whether the weight of this material would adversely affect the planned subsidence rate would require study. Such a pile would also require substantial precipitation runoff control to prevent surface water contamination.

Obviously, these approaches, modifications, and combinations of them, or others yet to be devised must be studied in much more detail before feasibilities can be assessed. Serious ground water pollution by the soluble salts in the spent shale would be difficult to prevent if a large subsidence unit, in which the material was placed, penetrated significant water-bearing strata. Depending on the retorting process, the spent shale may be of a texture which can be rendered almost impermeable by compaction, thereby reducing the hazard of percolation. But if a moderately permeable aquifer was cut by mining and bordered the spent shale, the interface between soil or rock and spent shale would present large contact across which soluble materials could move. One possible solution to this potential problem would be to remove the salts from the spent shale before disposal. This removal may be accomplished if recovery of other minerals, particularly nahcolite and dawsonite was accomplished during shale processing. Recovery of other minerals would also reduce the volume of spent shale requiring disposal.

4.7.3.3 Impacts of Mine-Associated Facilities

Surface construction and development necessary to mining operation contribute to the environmental impact of the total project, regardless of the mining technique used. Not considering the retort plant and associated construction, several mine-associated facilities have major impacts.

Transportation and Utility Corridors. The land area required for power transmission, pipeline, and road corridors will be at least temporarily removed from production as wildlife habitat. However, with the exception of roads, these corridors can be rehabilitated following construction and pose no long-term impact threat. Electric power transmission lines built according to BLM specifications for minimizing electrocution hazard to birds will not create an impact on eagles or other large birds.

Water Retention and Transport Structures. If water-holding or tailings ponds which contain toxic materials are constructed, they may constitute hazards to waterfowl which seasonally frequent the Piceance Creek Basin. Such ponds will also displace a small area of terrestrial wildlife habitat but the loss probably will not be significant. Construction of all such structures should be adequate to virtually eliminate any reasonable possibility of failure, whether or not the retained material is toxic.

Raw Shale Stockpile. Raw oil shale stored prior to retorting will require a certain amount of land. However, the crushed shale will not create a hazard to water quality through erosion as would retorted shale. Therefore, it should not be necessary to stabilize its surface with vegetation or covering of any kind from an environmental standpoint. After mining is finished, the area can be revegetated without serious difficulty, since no soil will have been removed.

Development Rock Disposal. The volume of rock extracted from the development of four mine shafts is estimated at 274,000 yd^3. Since this material will not have been subjected to weathering like exposed surface rock, it may constitute a source of pollution to surface waters, depending on its composition. Pre-development core drilling should provide data on this question. However, if it is found to bear unstable compounds which could break down to create acid drainage or other hazards to the environment, it may be necessary to build some type of containment structure. The amount of material involved is not so large as to preclude such an approach. Other alternatives include using the rock as concrete aggregate, road base, or fill material.

Surface Machinery and Construction. At least three types of environmental impact will be created by construction of head frames, crushers, conveyors, and related facilities. Noise generated in the head frame assembly and by crushers can be a significant impact, depending on the design of these facilities. If the primary crusher is located underground as expected, its surface noise contribution will be nonexistent. However, secondary crushing, movement of shale to the surface, dumping, and transport to the stockpile would generate considerable noise if uncontrolled. High noise levels would be expected to have significant effects on wildlife especially those species known to be relatively intolerant of human activity such as bear, mountain lion, and eagles. In effect, human activity decreases the amount of habitat available to these species because they tend to move out of the area as human activity increases.

Another potentially significant impact is degradation of air quality from dust in the immediate area of these installations. As discussed previously, dust suppression systems and procedures will probably be required.

Finally, the impact on aesthetics should be considered. Although it probably is impossible to completely eliminate this impact, it is quite possible to reduce it significantly. Some controls and restrictions on development related to aesthetics will probably be required by law, as they are in the Prototype Oil Shale Leasing Program. Examples of such measures are restricting road and corridor widths to a minimum, aesthetic consideration of sites selected for construction, and the compatible designs, including color, of all buildings.

4.7.4 Cost Estimates for Environmental Programs

The following cost estimates are preliminary and are intended only as broad guidelines. All estimates are based on the previous experiences of similar work, except for the environmental damage mitigation category. This category includes too many variables to be precise. The figure of $6,000,000 is intended to include only the costs of structures and facilities accessory to the mine, and not basic modification or design changes which may be necessary.

4.7.4.1 Baseline Data Collection

Program objective: Conduct field and laboratory studies in all environmental disciplines pertaining to ecology, meteorology, and hydrology specifically of the selected tract of land.

Program duration: Two years (minimum)

Program cost: $2,000,000

4.7.4.2 Environmental Monitoring

Program objective: Conduct continuous monitoring of air and water quality (surface and ground) and selected ecological parameters.

Program duration: Life of project

Program cost: $300,000 per year

4.7.4.3 Environmental Damage Mitigation

Program objective: Minimize serious environmental impacts during both construction and operation of mine by incorporating certain design modifications and operating procedures.

Program duration: Irregular - design inputs early; intermittent specific measures for life of project.

Program cost: $6,000,000

4.7.4.4 Research on Spent Shale Disposal

Program objective: Develop the technology for establishing permanent, beneficial vegetation on spent shale disposal area.

Program duration: Five years

Program cost: $1,500,000

4.8 HEALTH AND SAFETY

In the evaluation and design of the candidate mining systems, the health and safety of all underground personnel were of primary concern. All areas where personnel are required to work, even if on an intermittent basis, are adequately supported and designed using conservative factors of safety. The decision to design the mining systems considering the presence of methane was based on some evidence of small amounts of gas found in drill holes in the proposed mine site area. Ventilation requirements are considerably higher, due to dilution requirements for gas and dust, than for non gassy mines. A general discussion of the safety features are included for ventilation, fire prevention, working areas and equipment is presented followed by the estimated cost of the health and safety program.

4.8.1 Ventilation and Dust Control

Ventilation circuits are designed so that air is not reused and return and intake air drifts are isolated from each other. Systems of automatic air doors and regulators, possibly computer controlled, are used to ensure proper air quantities and velocities in working areas. All mine designs provide a sufficient quantity of air to dilute and carry away harmful gases and dusts. Control of dust created by drilling, blasting, and haulage is done by: (1) drilling with water, (2) keeping muck piles wet, and (3) watering haulage ways used by rubber tired loading equipment.

4.8.2 Fire Prevention and Protection

Drilling jumbos, LHD's, FEL's and all conveyor belt drives have built in fire suppression systems. Portable fire extinguishers are mandatory on all underground equipment. Fire chemical trucks are located throughout the mines for emergency control of large fires. Water lines and outlets are provided along all track and conveyor belt systems. Water trucks, used for dust suppression during production, are also available in case of fires.

4.8.3 Working Area Safety

The labor structure is set up so that wherever possible, no one man is working alone in a production or development area. This feature is generally standard in highly mechanized operations and helps to reduce serious or fatal injuries. Page phones are also located near working areas and belt drives. As mining advances, permanent phones are installed in strategic locations for emergency use and outside communications. First aid kits and stretcher sets are provided at all phone stations. All underground employees are required to carry MESA approved self rescuers and dust respirators. Methane detectors are issued to all underground supervisors. Emergency escape ways are provided in all shafts along with safety rooms at most phone locations.

4.8.4 Equipment Safety Features

The equipment used in the candidate mining systems are all classified as permissible and have roll over and impact resistant protective canopies. All diesel equipment have exhaust scrubbers and mufflers to reduce hydrocarbon emissions and noise levels. Adequate standby equipment is provided so that preventative maintenance can be performed regularly, thus reducing the possibility of equipment fires, explosions, and injuries.

4.8.5 Cost of Health and Safety Program

Table 4.53 is a breakdown of costs for various phases of the health and safety program for each mining system. The largest item contributing to increased production costs is the additional manpower required for working area safety. The cost of having permissible equipment is less than 20% of the total additional cost. The total health and safety program cost is estimated to range from $0.02 to $0.04 per ton of production (30.175 MM tpy), depending on the mining system.

Table 4.53 Estimated Health and Safety Costs for Candidate Mining Systems

Mining System	Safety Equipment Cost ($)	*Permissible Equipment Cost ($)	Additional Labor Cost ($)	**Additional Ventilation Costs ($)	Total Cost ($)	Cost Per Ton ($)
Sublevel stoping with full subsidence	83,000	160,000	602,000	4,000	849,000	0.03
Sublevel stoping with spent shale backfill	83,000	160,000	602,000	4,000	849,000	0.03
Advance entry and pillar	119,000	182,000	490,000	6,000	797,000	0.03
Chamber and pillar	90,000	160,000	490,000	6,000	746,000	0.02
Block caving with slushers	93,000	103,000	980,000	4,000	1,180,000	0.04
Block caving with LHD's	94,000	155,000	1,008,000	4,000	1,261,000	0.04

* 10% of yearly depreciation for underground equipment that must be permissible by law (includes canopies, fire suppression system)

** Cost of increasing ventilation requirements for gas dilution.

4.9 COST SUMMARY OF MINING SYSTEMS

A summary of all cost data for each mining design is presented in Table 4.54. This table includes net capital investment, preproduction cost, value of development ore, annual costs and total annual production cost. Included in operating costs are operating supplies necessary for mine operation, contingencies, and yearly interest cost on capital items (9%). Payroll overhead is assumed to be 35%. Indirect costs are 10% of labor, supervision, and operating supplies. Fixed costs include taxes, insurance and depreciation.

Included in depreciation costs are preproduction costs that would not generally appear if a corporation had other income against which development costs could be charged. For example, if mine development were not charged to net capital investment the annual production cost of sublevel stoping with full subsidence would decrease by $0.08 per ton.

Not included in the analyses are royalty payments, welfare payments, and surface transportation costs. All data were computed from costs collected during the first quarter, 1975.

Table 4.54 Summary of Candidate Mining System Costs

Mining Method*	Net Capital Investment ($)	Labor & Super-vision ($/ton)	Annual Costs: Operating ($/ton)	Power ($/ton)	Reclama-tion ($/ton)	Payroll Overhead ($/ton)	Explora-tion ($/ton)	Environ-mental ($/ton)	Indirect ($/ton)	Fixed ($/ton)	Total Annual Cost ($/ton)	Preprod. Cost ($)	Est. Value of Devel. Ore ($)
1	113,434,100	0.26	0.33	0.13	0.03	0.09	0.02	0.01	0.04	0.23	1.15	30,067,000	53,512,000
2	95,797,300	0.27	0.32	0.13	0.03	0.10	0.02	0.01	0.04	0.20	1.12	15,430,000	17,087,000
3	84,148,000	0.32	0.39	0.14	0.03	0.11	0.02	0.01	0.06	0.19	1.27	8,975,000	31,785,000
4	77,200,000	0.27	0.24	0.12	0.03	0.10	0.02	0.01	0.04	0.18	1.04	3,469,000	13,570,000
5	125,710,000	0.44	0.27	0.14	0.03	0.16	0.02	0.01	0.05	0.23	1.35	41,572,000	72,024,000
6	122,872,000	0.39	0.31	0.12	0.03	0.14	0.02	0.01	0.05	0.24	1.31	36,262,000	54,576,000

* 1 Sublevel stoping with full subsidence
2 Sublevel stoping with spent shale backfill
3 Advance entry and pillar
4 Chamber and pillar
5 Block caving with slushers
6 Block caving with LHD's

SECTION 5

RANKING OF MINING SYSTEMS

Selection of a technically and economically suitable mining method considering the many alternatives, becomes a complex problem because of the interaction of the many variables. The problem would be simple if the decision was based upon only one criteria or factor. For example, if the decision criteria selected is production cost, the mining method with the lowest production cost is selected. The complexity of the problem may increase considerably with an increase in decision criteria. In reality ranking analyses generally involve many criteria having different weights.

In ranking analyses the most difficult part is to determine the weight of each criteria affecting the overall selection. A method called DARE (Decision Alternative Ration Evaluation) is used for the ranking analysis of the candidate mining systems (Klee, 1970).

5.1 DESCRIPTION OF METHOD

The DARE method was developed by the U.S. Bureau of Solid Waste Management as an offshoot of the complicated problems encountered in solid waste research. It is used to provide a rational choice among alternatives (mining methods) based on carefully selected criteria. Criteria are determined by a group of professionals, who generate a weighted index of factors and then score the alternatives using the weighted factors. The evaluation factors are clearly expressed and the values assigned by each professional are readily identifiable. The method incorporates both qualitative data and quantitative factors. The use of qualitative data introduces subjectivity into the final score. This subjectivity, however, is only as positive in the results as is the technical ability of the professional performing the ranking analysis. Quantitative data, where available, are incorporated in the form of ratios which automatically produce the required weights.

In general, a list of major factors and subfactors which can be used to evaluate alterantives is established. The evaluators then make a comparison between factors taking them two at a time and assign a value as to the relative importance of one over the other. The list is then rearranged by a random selection and re-evaluated. By doing this several times biased responses can be reduced. Mean values are then used for the weights and scores.

Obviously, the effectiveness of the DARE method is a function of the proper selection of the evaluation factors, whether or not they are qualitative or quantitative, and the qualifications of the evaluators.

5.2 SELECTION AND WEIGHING OF FACTORS AND SUBFACTORS

Factors and their subfactors affecting the mining systems are identified and listed in Table 5.1. Minor factors are either excluded from the analysis or included indirectly in other factors. Weighing of the factors and subfactors has been done by four of the principle investigators by the methods previously discussed. Each investigator conducted pairwise comparison and weighing four separate times. An average for all four investigators was obtained for each factor resulting in reasonably unbiased factor weights. The weights obtained for each factor and subfactor are also listed in Table 5.1.

5.3 FACTOR SUBSCORES

The next step in the ranking analysis is to obtain subscores for each factor for each mining method under consideration. Tables 5.2 through 5.7 include computations for the factor subscores. The order of listing of the mining methods is arbitrary and is the same in all subsequent tables. Values assigned to the mining methods for each factor are based on either the investigators judgement, in cases where factors have no fixed value, or linear interpolation, for cases where factors have known values. For example, in health and safety, where no fixed value is available for a particular mining method, investigators used their own judgement. On the other hand, for production cost, which varied from $1.04 to $1.35 per ton, the best score was assigned to the method having the lowest cost.

Table 5.1 Factors and Weights Used in Ranking Analysis

Factor		Weights	
Technical Feasibility			24
A_1	Equipment development	23	
A_2	Preproduction time	27	
A_3	Stage of method development	19	
A_4	Productivity	31	
Mining Cost			22
B_1	Capital investment	40	
B_2	Labor and supervision	28	
B_3	Operating cost	33	
Resource Recovery			20
C_1	Mining selectivity	38	
C_2	Percent extraction	62	
Reclamation (D_1)			7
Environmental Impact			11
E_1	Socio-economic	34	
E_2	Wildlife	23	
E_3	Land use	25	
E_4	Underground spent shale disposal	18	
Health and Safety (F_1)			16
	Total		100

Table 5.2 Subfactor Scores and Weights for Technical Feasibility

Subfactors	A_1	A_2	A_3	A_4		Analysis		
Subfactor Weight	23	27	19	31	Total	Ratio	K	Subscores S
Mining* Method								
1	11	19	11	0	41	1.07	1.32	0.18
2	18	12	13	1	44	0.89	1.23	0.16
3	23	5	6	5	39	0.90	1.39	0.19
4	18	2	14	1	35	1.51	1.54	0.21
5	10	27	4	12	53	1.02	1.02	0.14
6	10	27	7	10	54	-	1.00	0.13
							7.50	1.01

* Method 1 - Sublevel stoping with full subsidence
Method 2 - Sublevel stoping with spent shale backfill
Method 3 - Advance entry and pillar
Method 4 - Chamber and pillar
Method 5 - Block caving with slushers
Method 6 - Block caving with LHD's

Table 5.3 Subfactor Scores and Weights for Mining Costs

Subfactors	B_1	B_2	B_3		Analysis		
Subfactor Weight	40	28	33	Total	Ratio	K	Subscores S
Mining Method							
1	36	17	28	82	0.94	1.05	0.16
2	31	18	27	76	1.07	1.13	1.18
3	27	21	33	81	0.81	1.06	0.16
4	25	18	23	66	1.35	1.31	0.20
5	40	28	21	89	0.97	0.97	0.14
6	37	23	26	86	-	1.00	0.16
						6.52	1.00

Table 5.4 Subfactor Scores and Weights for Resource Recovery

Subfactors	C_1	C_2		Analysis		
Subfactor Weight	38	62	Total	Ratio	K	Subscores S
Mining Method						
1	20	5	25	2.00	1.40	0.24
2	20	30	50	1.00	0.70	0.12
3	10	40	50	0.76	0.70	0.12
4	13	25	38	0.92	0.92	0.16
5	30	5	35	1.00	1.00	0.17
6	30	5	35	-	1.00	0.17
					5.72	0.98

Table 5.5 Subfactor Scores and Weights for Reclamation

Subfactor	D_1		Analysis		
Subfactor Weight	7	Total	Ratio	K	Subscores S
Mining Method					
1	6	6	0.17	1.01	0.06
2	1	1	3.00	5.94	0.35
3	3	3	0.33	1.98	0.12
4	1	1	6.00	6.00	0.35
5	6	6	1.00	1.00	0.06
6	6	6	--	1.00	0.06
				16.93	1.00

Table 5.6 **Subfactor** Scores and Weights for Environmental Impact

Subfactor	E_1	E_2	E_3	E_4		Analysis		
Subfactor Weights	34	23	25	18	Total	Ratio	K	Subscores S
Mining Method								
1	21	20	25	18	84	0.40	0.93	0.11
2	18	10	5	1	34	1.82	2.33	0.26
3	17	18	18	9	62	0.55	1.28	0.14
4	20	8	3	3	34	2.38	2.33	0.26
5	15	23	25	18	81	0.98	0.98	0.11
6	16	20	25	18	79	--	1.00	0.11
							8.85	0.99

Table 5.7 Subfactor Scores and Weights for Health and Safety

Subfactor	F_1		Analysis		
Subfactor Weight	16	Total	Ratio	K	Subscores S
Mining Method					
1	7	7	0.86	0.56	0.15
2	6	6	2.17	0.65	0.17
3	13	13	0.92	0.30	0.08
4	12	12	0.33	0.33	0.09
5	4	4	1.00	1.00	0.26
6	4	4	--	1.00	0.26
				3.15	1.01

5.4 RANKING

Weights for each factor are multiplied by the subscores for each mining method. Final scores for each mining method are obtained by summing the weighed scores for all the factors. The mining method having the highest score is considered to be the best mining method. Table 5.8 is a summary of the ranking analysis. Listed below are the candidate mining systems with the highest ranking first and lowest ranking last.

- Chamber and pillar
- Sublevel stoping with spent shale backfill
- Sublevel stoping with full subsidence
- Block caving using LHD's
- Block caving using slushers
- Advance entry and pillar

Table 5.8 Summary of Ranking Analysis Scores

Factor	Factor Weights	Sublevel Stoping Full Subsidence: Fact. Subsco.	Sublevel Stoping Full Subsidence: Weighted Scores	Sublevel Stoping Backfill: Fact. Subsco.	Sublevel Stoping Backfill: Weighted Scores	Advance Entry and Pillar: Fact. Subsco.	Advance Entry and Pillar: Weighted Scores	Chamber and Pillar: Fact. Subsco.	Chamber and Pillar: Weighted Scores	Block Caving with Slushers: Fact. Subsco.	Block Caving with Slushers: Weighted Scores	Block Caving with LHD's: Fact. Subsco.	Block Caving with LHD's: Weighted Scores
1. Technical Fea.	24	0.18	4.32	0.16	3.84	0.19	4.56	0.21	5.04	0.14	3.36	0.13	3.12
2. Mining Costs	22	0.16	3.50	0.17	3.74	0.16	3.50	0.20	4.40	0.14	3.08	0.16	3.52
3. Resource Recovery	20	0.24	4.80	0.12	2.40	0.12	2.40	0.16	3.20	0.17	3.40	0.17	3.40
4. Reclamation	7	0.06	0.42	0.35	2.45	0.12	0.84	0.35	2.45	0.06	0.42	0.06	0.42
5. Environmental Impact	11	0.11	1.21	0.26	2.86	0.14	1.54	0.26	2.86	0.11	1.21	0.11	1.21
6. Health and Safety	16	0.15	2.40	0.17	2.72	0.08	1.28	0.19	3.04	0.26	4.16	0.26	4.16
Total Score	100	--	16.65	--	18.01	--	14.12	--	20.99	--	15.63	--	15.83
Rank		--	3	--	2	--	6	--	1	--	5	--	4

SECTION 6

CONCLUSIONS AND RECOMMENDATIONS

An analysis of the technical feasibility and cost evaluation of candidate mining systems for mining the deep oil shale deposits of the Piceance Creek Basin has been conducted. The following conclusions and recommendations are made.

6.1 CONCLUSIONS

1. Large scale, underground mining of the central Piceance Creek oil shale deposits is technically feasible and of low cost. Four of the six mining systems evaluated were selected as being the most promising for further evaluation. These mining systems are:

- Chamber and pillar
- Sublevel stoping with spent shale backfill
- Sublevel stoping with full subsidence
- Block caving using LHD's

The first two methods have the lowest production cost and environmental impact. Mining selectivity is greatest in chamber and pillar and lowest in block caving. However, if mining is conducted in the leached zone, block caving may be the most desirable method. Resource recovery is highest in block caving and sublevel stoping with full subsidence and lowest in sublevel stoping with spent shale backfill.

2. Adequate oil shale resources, averaging at least 20 gpt, are present in proposed mine site 2 and mineable by either selective or bulk mining methods. Enough recoverable reserves of dawsonite and nahcolite are also present that may significantly add to the total resource value and increase the final product value.

3. Projecting from available hydrologic data, water inflow from the three aquifer systems will not significantly impede oil shale mining on site 2. An expected flow rate of 10,000 gpm can be adequately handled with present technology.

4. The rock mechanics data on Green River oil shale in the central portion of the Piceance Creek Basin is insufficient to determine design safety factors or failure probabilities. A physical testing program is needed to provide more accurate and reliable design data.

5. The mining costs, as calculated in section four, assume no corporate structure and no product value. In a complete economic evaluation, the mining costs may be lower because development costs can be charged against corporate cash flow. In addition, some early cash flow is generated by the sale of significant quantities of development ore. A major factor that was not considered in the cost evaluations were the government lease or royalty payments.

6. The environmental impact of underground oil shale mining, from both a socio-economic and ecological standpoint, will be significant. However, an oil shale mining industry can create long term, good paying jobs and serve to create strong and long lasting communities. Significant areas of land will be affected by spent shale disposal and land subsidence; however, the ecological impact can be reduced through land reclamation and controlled subsidence.

6.2 RECOMMENDATIONS

1. It is recommended that more refined design and cost evaluation studies on the four most promising mining methods be initiated. These methods are:

- Chamber and pillar
- Sublevel stoping with spent shale backfill
- Sublevel stoping with full subsidence
- Block caving using LHD's

The preliminary program outline is presented in Section 7.

2. Evaluation of the structural geology, hydrology, and resource of selected site 2 should be updated and re-evaluated. It is also recommended that the more detailed mine designs be performed considering site 2 as a possible prototype mine site.

3. An estimate of the socio-economic and ecological impact of each mining method should be conducted.

4. Additional rock mechanics data on the Green River Formation oil shale near the central portion of Piceance Creek should be made available or obtained through a drilling program.

5. An economic evaluation of oil shale mining by underground methods should be conducted that includes the estimated cost of processing and a market value of the shale oil and associated recoverable minerals.

SECTION 7

PHASE II REPORT OUTLINE

As required in section 1-3, Article I in the contract award, an outline of the phase II report is presented.

Cover
Frontispiece
Foreword
Abstract
Executive Summary
Contents
List of Figures
List of Tables
List of Symbols and Abbreviations

1. Introduction
   1.1 Purpose
   1.2 Approach
   1.3 Content

2. Summary of Phase I Evaluation

3. Refinement of Selected Candidate Mining Systems
   3.1 Refinement of Resource Evaluation
   3.2 Refinement of Selected Mine Designs
      3.2.1 Mine Access
      3.2.2 Mine Ventilation
      3.2.3 Equipment Performance and Selection
      3.2.4 Rock Mechanics Analysis
      3.2.5 Refinement of Cost Data
   3.3 Refinement of Environmental Impact
   3.4 Mining System Costing

4. Economic Evaluation

5. Conclusions

6. Recommendations

Appendices

References

APPENDIX A

SUBLEVEL STOPING METHODS

A-1

A.1 HEADING ROUND DESIGN

The following heading round designs have been determined by reviewing pertinent literaterature (East and Gardner, 1964 and Zambas et al, 1972) and visiting the Anvil Points mine (Paraho Project). The rounds are basic V-cuts, loaded and blasted with ANFO. Figures A.1 and A.2 are the designs used in this analysis to estimate drilling cycle times and heading round advance per shift. Holes are drilled 2-1/2 to three inches in diameter, 17 feet deep, giving an estimated average advance of 15 feet per blast. Zambas et al (1972) used a powder factor between 0.6 and 0.7 pounds of ANFO per ton of oil shale to produce satisfactory fragmentation in a 40 by 60-foot, 25-foot deep round. For this analysis a powder factor of 0.55 was considered to be satisfactory.

A two-boom drill jumbo, assuming an average drilling rate of five feet per minute, can drill a 30 by 20-foot round in about 32 minutes from one set up. Actual drilling rates for rotary hydraulic drills are eight to ten feet per minute; however, to include down time, bit change, and unexpected stoppages a rate of five feet per minute is used. A 20 by 20-foot round is drilled in 28 minutes. When moving and set-up times are included, a round (20 by 30-foot and 20 by 20-foot) can be drilled in approximately two hours.

A.2 PILLAR DRILLING AND BLASTING

Once full open stope production has been completed and the stope area is drawn empty the three exterior pillars are blasted into the stope. During the period of open stope production the three pillars are drilled and prepared for blasting a total of 62,650 feet of fan drilling must be completed before the pillars are ready for blasting. With an average drilling rate of five feet per minute over a period of 37 days it will take approximately three hours per shift for one two-boom drilling jumbo to prepare two stopes. Zambas et al (1972) estimates that a powder factor (PF) of 0.30 pounds of ANFO per ton of ore broken will produce satisfactory fragmentation. In the summary of calculations presented in Table A.1 a PF of 0.35 has been used.

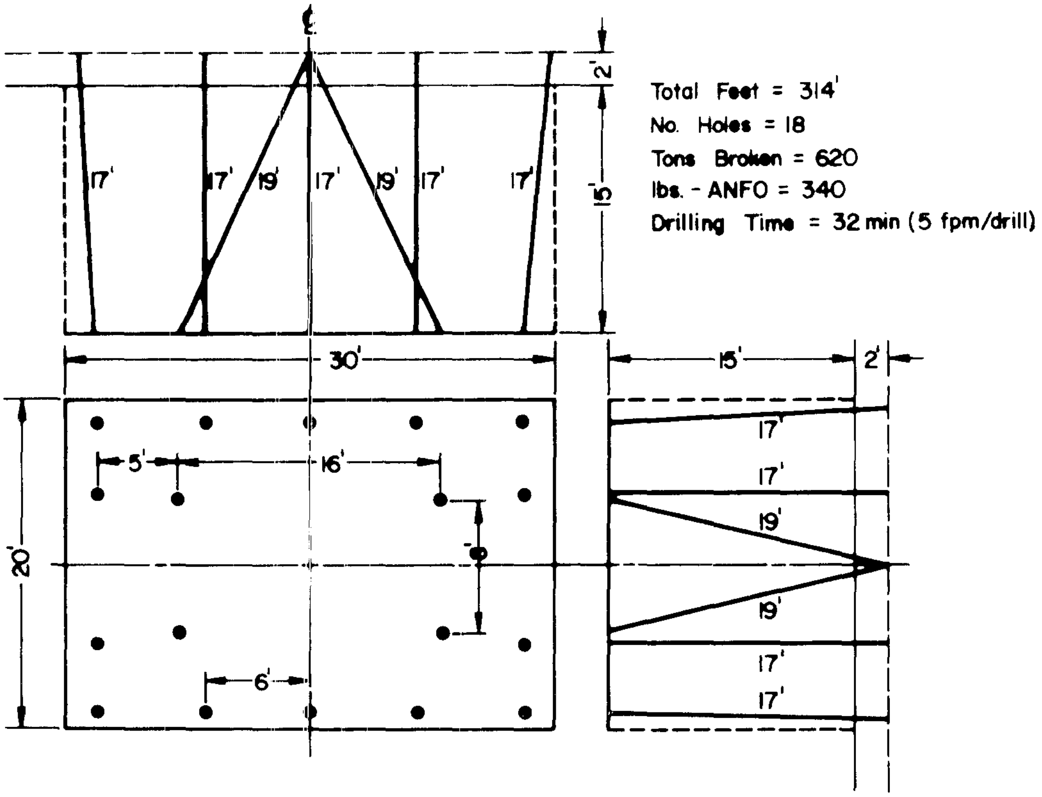


Figure A.1 Heading Round Design for 30 by 20-foot Drifts

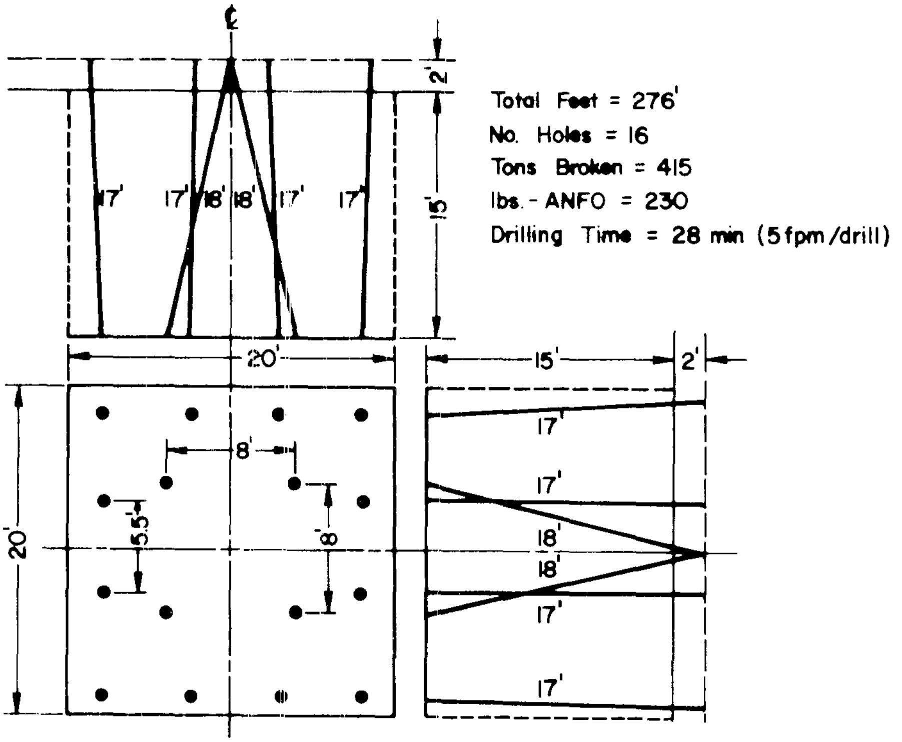


Figure A.2 Heading Round Design for 20 by 20-foot Drifts

Figure A.3 through A.5 are scale drawings of proposed fan drill layouts used in this analysis. A burden of ten feet has been assumed to be the most reasonable to use in this initial analysis. In reality it is believed this value is conservative; however, more research needs to be done in this area.

Loading of the drill holes is done with pneumatic ANFO loading machines capable of loading up to 10,000 lbs of ANFO per shift. Primers consist of conventional sticks of 60% strength dynamite initiated by electric blasting caps.

The blasting sequence consists of initiating the end pillar first, then the back end of the crown pillar and back half of the rib pillar, and the front halves of rib and crown pillars last. As blasting is completed the caved waste material above the stoping level will subside into the void area. The volume available for the blasted ore is approximately 889,500 yd^3 and the volume of in place pillars is approximately 587,000 yd^3. With a swell factor of 1.33 the volume of broken ore becomes 780,800 yd^3, leaving a void area of about 695,000 yd^3 or about 47% of the total volume.

A.3 VENTILATION REQUIREMENTS

The ventilation requirements for both full subsidence and spent shale backfill stoping methods are the same. Summarized here are the air requirements necessary for diesel equipment and men, for gassy mining conditions under present Colorado mining laws (Colorado Bureau of Mines, 1971 and State of Colorado, 1966).

A total of 2,000,000 cfm is used and will be supplied by four axial-vane, 12-foot diameter fans exhausting from two 16-foot diameter ventilation shafts.

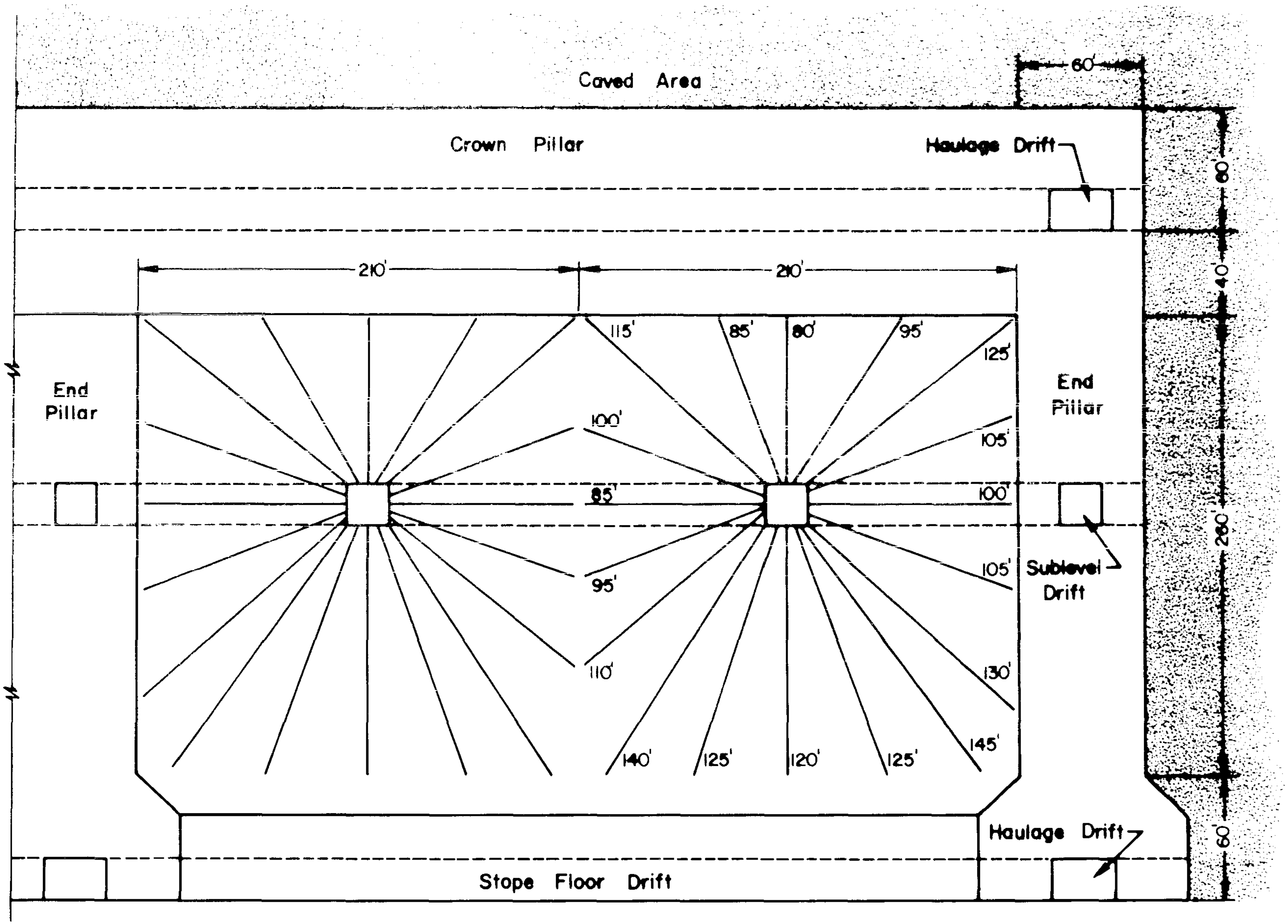


Figure A.3 Rib Pillar Fan Drill Design for Sublevel Stoping with Full Subsidence

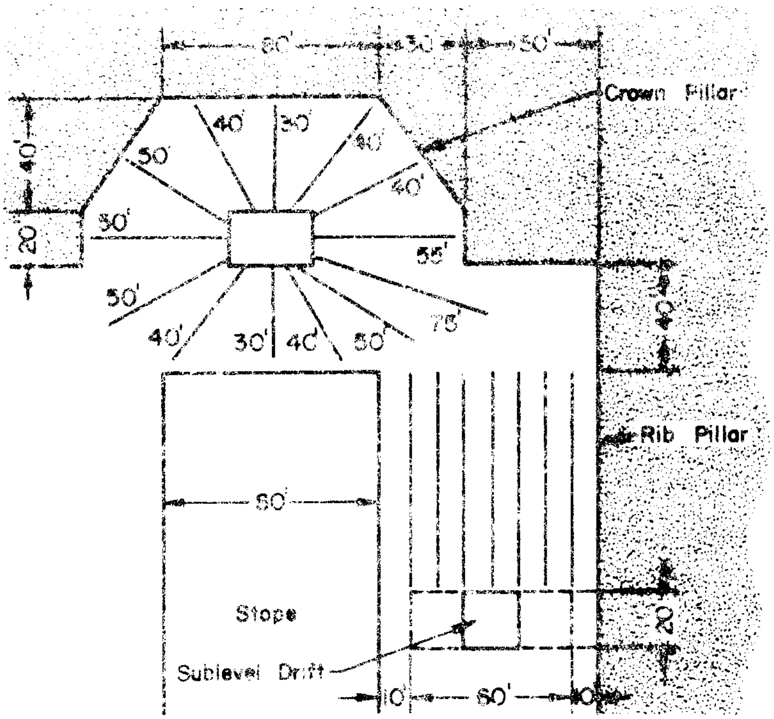


Figure A.4 Crown Pillar Fan Drill Design for Sublevel Stoping with Full Subsidence

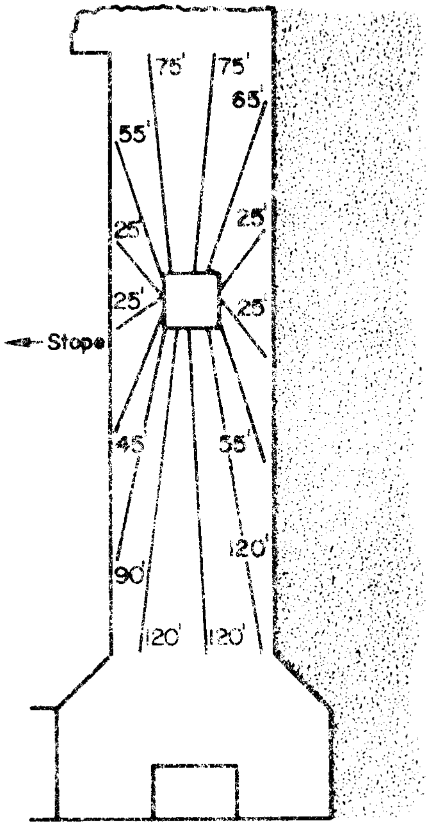


Figure A.5 End Pillar Fan Drill Design for Sublevel Stoping with Full Subsidence

Table A.1 Summary of Fan Drilling Designs

Description	No. Fans	Footage	Feet Day	No. Holes	Tonnage	Drilling time (hrs)	ANFO (lbs)
Rib Pillar Fan Drills	14	27,790	751	252	566,670	93	198,340
Crown Pillar Fan Drills	46	27,370	740	598	456,000	92	160,000
End Pillar Fan Drills	7	6,440	175	98	70,580	22	24,700
TOTAL	67	61,600	1,666	948	1,093,250	207	383,040

Description	Quantity	HP/Unit	Total HP	Air Req'd (cfm)
FEL (12 yd^3)	21	612	12,900	964,000
Two-boom Drill Jumbo	7	100	700	53,000
Roof Bolting Machine	4	100	400	30,000
Fan Drill Machine	2	100	200	15,000
Trolley Locomotive and Misc.			1,000	75,000
Subtotal.				1,137,000
Contingency (15%)				171,000
Subtotal.				1,308,000
350 men/shift @ 100 cfm each.				35,000
Contingency (15%)				6,000
Subtotal.				1,349,000
Leakage (45%)				608,000
Total.				1,957,000

A.4 MAJOR EQUIPMENT SELECTION

The following equipment selection determinations for major items are based on best estimates of equipment availability, operator efficiency, and equipment performance. Availability of equipment varies depending on the type of work and handling it receives during duty cycles. In general, availabilities between 65 and 85% have been applied to various types of equipment in order to determine the number of standby machines needed.

Operator efficiency has been assumed constant for all equipment and is estimated by considering a five-hour work shift (75% efficiency with one hour lunch period). A work day consists of three, eight-hour shifts, seven days per week. Equipment performance data has been taken from vendor data publications. Where necessary, modifications have been made to take into account special mining conditions.

A.4.1 Drilling Jumbos

High penetration rates in oil shale can be achieved using rotary-hydraulic drills with drag bits. East and Gardner (1964) published the results of drilling tests at the Anvil Points Mine and report that rotary drilling rates were as much as three times that of percussion drilling. A major manufacturer of drilling jumbos has a prototype two boom jumbo currently being tested at the Occidental Oil Shale pilot mine near Grand Junction, Colorado. Performance data from this prototype machine has been used to determine heading cycle times.

The estimated drilling rate in 20 gpt oil shale is at least eight fpm per drill or 16 fpm per jumbo for two to three inch holes. A drilling efficiency of 65% has been assumed and includes bit changes, steel changes, and minor mechanical problems (10 fpm per jumbo). The heading rounds are designated so that an entire round can be drilled from one initial jumbo setup. The 30 by 20-foot drift has a total drill length of 314 feet, producing a total drilling time of about 32 minutes. A total drilling time of about 28 minutes is needed for the 20 by 20-foot FEL and sublevel drifts. Table A.2 shows the estimated role of the drilling jumbo in the full heading cycle.

Each jumbo can realistically drill out three headings per shift. During full production the total number of jumbos needed are:

1.	Level and main haulage development . . .	3
2.	Stope development.	3
	Subtotal.	6
3.	Standby (70% availability)	2
	Total	8

A.4.2 Fan Drills

At the present time a two-boom, rotary-hydraulic fan drill jumbo capable of drilling a 360 degree arc has not been developed. However, manufacturers of drilling equipment have indicated that this type of machine is feasible at a reasonable cost.

A total of 61,600 feet of ring drilling is needed for the pillar drilling of one stope and 70,110 feet for angle stope production ring drilling. Although a large footage is drilled from one fan drill set up and the moving distance is much shorter, drill steel changes are a much larger factor than for heading jumbos. A drilling rate of eight-fpm per jumbo is therefore assumed to determine the quantity of machines needed.

Table A.2 Cycle Time for a 30 by 20-foot Heading Round

	Description	Time (min)	Total Time
1.	Set up jumbo for drilling (30' x 20')	30	0-30
2.	Drill round	32	30-62
3.	Roof bolt previous round	70	0-70
4.	Breakdown jumbo for next set up	10	62-72
5.	Charging crew set up	20	72-92
6.	Move jumbo to next heading	45	72-117
7.	Charging crew load round	60	92-152
8.	Charging crew prepares for blast	10	152-162
9.	Blast and clear dust	30	162-192
10.	Muck round	60	192-252
11.	Advance track, pipe, ventilation, etc.	60	252-312

1.	Stope floor fan drills.	2
2.	Sublevel stope fan drills . . .	2
3.	Pillar fan drills	2
	Subtotal	6
4.	Standby (85% availability). . .	1
	Total.	7

A.4.3 Train Locomotives

The calculations used to determine the size and quantity of locomotives needed to adequately haul the required tonnages were simplified in this analysis. A maximum round trip length of 12,120 feet has been used and should be quite a conservative estimation. Track grade is estimated between 1/4 and 1/2% with the shaft area at the low point. Loaded trains will have the grade in their favor and haul ten cars per trip. The following data was used to estimate the weight of locomotive needed:

1. Trailing load . . . . .1000 tons
2. Rolling resistance of mine cars 8-lb/ton
3. Frictional resistance of locomotive 15-lb/ton
4. Grade resistance (0.35%). 7-lb/ton
5. Acceleration rate . . . . $0.15 mps^2$
6. Coefficient of adhesion 0.25
7. Locomotive weight for grades in favor of loads (loaded). . . . . 32 tons
8. Locomotive weight for grades in favor of loads (unloaded). . . . 13 tons
9. Locomotive weight for grades in favor of loads (braking) . . . . 37 tons
10. Select a 40-ton locomotive.

Two, ten-car trains must be dumped every 20 minutes to produce the required tonnage. Therefore, two 40-ton locomotives will be needed for full production plus one standby locomotive. For development work on additional levels, two 30-ton locomotives will be used.

A.4.4 Ore Cars

Stope production will be approximately 12,000 tpd per stope or 800 tons per shift. The broken volume of 800 tons of oil shale is approximately 570 yd^3. Rail cars with an inside volume of 60 yd^3 and 80-ton capacity will fit easily within the 30 by 20-foot drifts and where needed, can be double tracked. The approximate size of a car is nine feet wide by six feet high by 30 feet long. Cars are dumped by rotary dump machines at the shaft station All cars will have pneumatic air brakes.

Two stopes will be serviced every 20 minutes by two locomotives. While two trains are being hauled and dumped, all six stopes will be loading empty cars. A total of 20 extra cars will be needed to haul development ore. The total amount of rail cars needed are:

1.	Cars being loaded	60
2.	Cars being hauled	20
3.	Development haulage	20
	Subtotal	100
4.	Standby (80% availability)	20
	Total	120

A.4.5 Front End Loaders

FEL size is limited to the dimensions of the haulage drifts and the machine turning radius. The FEL used in these calculations has a payload capacity of 33,000 pounds (16.5 tons or 12 yd^3), a height of 16-1/2 feet, width of 13 feet, and a turning radius of 68 feet. It is also capable of negotiating 12 to 15% grades while loaded. To produce reasonable mucking times, two FEL's are used to load cars and will take approximately one hour to load a total of 800 tons. The total amount of FEL's needed for full production is:

1.	Mucking stopes	12
2.	Sublevel development	2
3.	Haulage level development	3
	Subtotal	17
4.	Standby (60% availability)	7
	Total	24

A time-motion analysis of a FEL mucking a 30 by 20-foot heading round is given below. The heading round produces approximately 450 yd^3 of broken ore and the haulage track will be a maximum of 50 feet from the face. The average haulage distance will be about 280 feet total.

1.	Fill bucket.	10 sec
2.	Back to rail car (average 4 mph)	24 sec
3.	Turn and crowd one car	10 sec
4.	Dump	4 sec
5.	Turn	6 sec
6.	Forward to face (average 5 mph).	20 sec
	Subtotal.	74 sec
7.	Time to muck round (40 trips).	50 min
8.	Clean up time.	10 min
	Total	60 min

The mucking cycle time of the FEL on a 20 by 20-foot heading will be approximately the same as that of 30 by 30-foot headings, due to a longer hauling distance. The mucking time of the FEL is included in Table A.2, and it is anticipated that three FEL's will support the development drill jumbos.

A.4.6 Roof Bolting and Scaling Machines

Combining roof bolting and scaling machines into one unit should produce a more efficient operation and heading cycle time. Roof bolts are spaced 5.4 feet center-to-center (side bolts 1-1/2 feet from ribs) with an average of 15 bolts installed per round. With an installation time of five minutes per bolt it will take two machines approximately 40 minutes to set the bolts. A total of 70 minutes is used in Table A.2 to include additional time for scaling although both operations are simultaneous.

Two roof bolting and scaling machines will be needed for each drilling jumbo in use or a subtotal of 12 machines. Availability is estimated at 85%; therefore, a total of 14 machines are needed.

A.4.7 Underground Crushers

Section 4.3.2 describes the selection and calculation of various sizes of gyratory crushers and their installation costs. For both sublevel stoping methods two, 60-inch gyratory crushers having a capacity of 2,900 tph are used. Two rotary dump mechanisms for unloading rail cars and two surge bins are also installed for each crusher installation.

A.4.8 ANFO Loading Trucks

Heading and fan drill rounds are loaded with 5,000 pound capacity, truck mounted, pneumatic charging machines. Bulk ANFO (prilled) is used to charge the holes and can be loaded at a rate of approximately 1600 pounds per hour. Three machines are required for development, three for fan drills, and two for standby for a total of eight.

APPENDIX B

ROOM AND PILLAR VARIATIONS

B.1 HEADING, BENCH AND FAN DRILLING DESIGNS

Heading round designs for both the entry and pillar and the chamber and pillar mining methods have been based on the work by East and Gardner, (1964) and Zambas et al (1972). The rounds are basic V-cuts, primed with electric caps and dynamite, and charged with ANFO. Figures B.1 and B.2 illustrate the round designs used in the analysis to determine drilling cycle times and heading round advance. The heading round for entry and pillar mining is drilled 22 feet deep with an expected average advance rate of 20 feet per blast. Bench blasting holes average 30 feet of vertical pull per blast with a 15 foot burden. Chamber and pillar heading rounds are designed to get an average advance rate of 30 feet per blast. Fan drilling in the chamber drifts averages 520 feet of three-inch diameter holes having ten feet of burden. A powder factor of 0.55 was used for the heading round designs and 0.35 for the fan drill rounds.

The 90 by 40-foot heading rounds are drilled from a single set up using a four-boom hydraulic drilling jumbo. The 30 by 20-foot heading rounds use a conventional two-boom hydraulic drill jumbo. The drilling rate is assumed to be approximately six fpm per drill.

B.2 VENTILATION REQUIREMENTS

The ventilation requirements for the advance entry and pillar and the chamber and pillar systems are different due to the variation in mining technique. Each system is analyzed separately; however, both consider gassy mining conditions and comply with Colorado mining laws (Colorado Bureau of Mines, 1971 and State of Colorado, 1966).

B.2.1 Advance Entry and Pillar Design

The law requires that 3,231,000 cfm of air be present to ventilate the mine. To insure the safety of men, control dust, and to maintain sufficient air to ventilate the working faces 3,500,000 cfm of air is regulated through the mine. Two large 12-foot diameter, axial-vane fans in two 16-foot diameter shafts are used to exhaust air out of the mine.

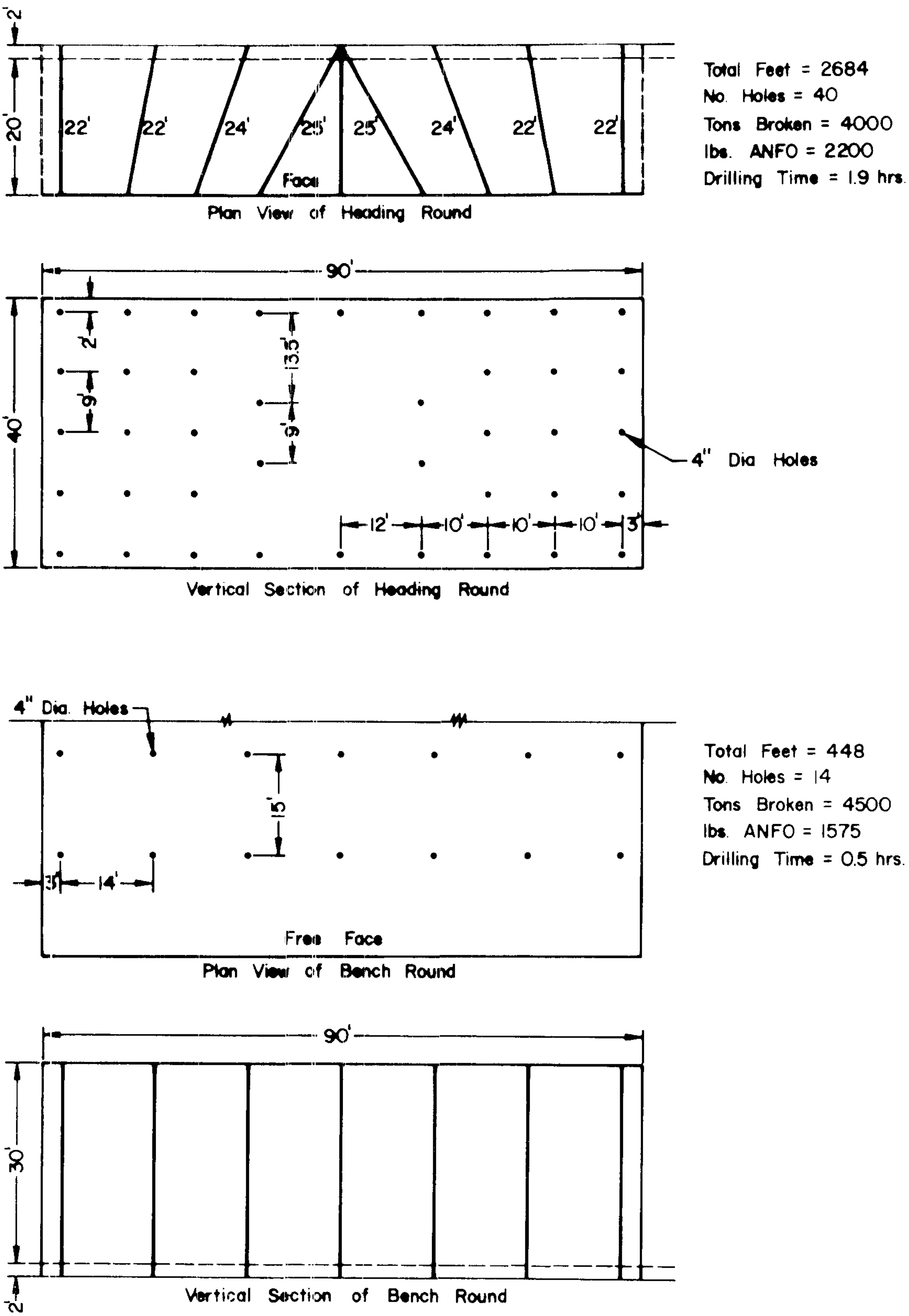
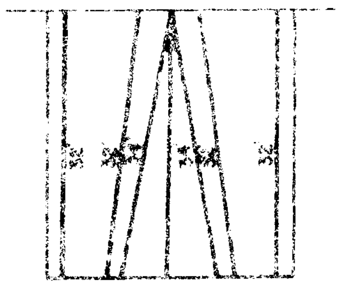
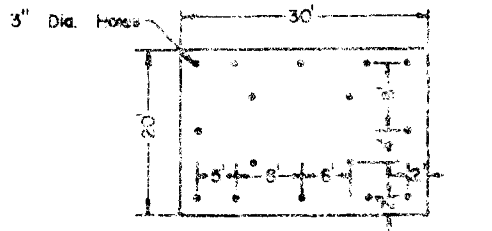


Figure B.1 Entry and Pillar Round Designs

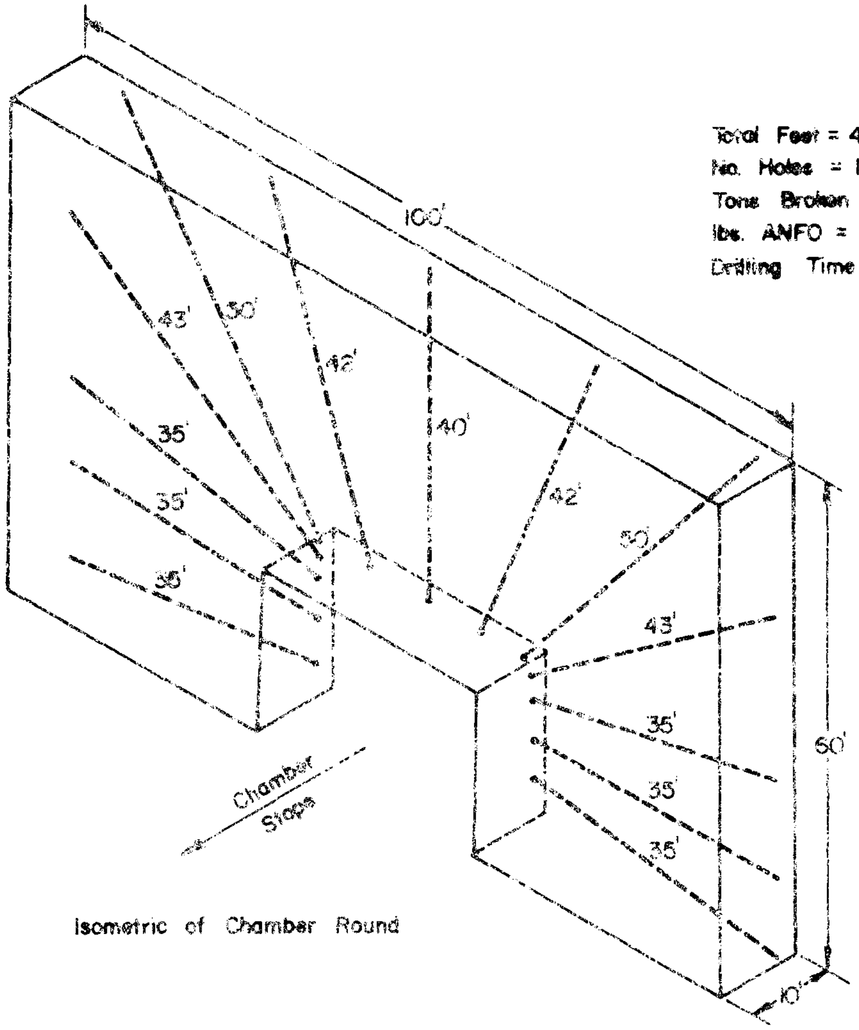


Plan View of Heading Round

Total Feet = 296
No. Holes = 16
Tons Broken = 1344
lbs. ANFO = 740
Drilling Time = 0.5 hrs.



Vertical Section of Heading Round



Isometric of Chamber Round

Total Feet = 485
No. Holes = 13
Tons Broken = 3800
lbs. ANFO = 2090
Drilling Time = 0.7 hrs.

Figure B.2 Chamber and Pillar Round Designs

Description	Quantity	HP/Unit	Total HP	Air Req'd (cfm)
LHD (12 yd^3)	38	394	15,000	1,125,000
Four-Boom Drill Jumbo	5	150	750	56,000
Two-Boom Vertical Drill	6	100	600	45,000
Bulldozer	8	500	4,000	300,000
Miscellaneous Equipment			5,000	375,000
Subtotal.				1,901,000
Contingency (15%)				286,000
Subtotal.				2,187,000
350 men/shift @ 100 cfm each.				35,000
Contingency (15%)				6,000
Subtotal.				2,228,000
Leakage (45%)				1,003,000
Total.				3,231,000

B.2.2 Chamber and Pillar Design

Description	Quantity	HP/Unit	Total HP	Air Req'd (cfm)
LHD (12 yd^3)	38	394	15,000	1,125,000
Two-Boom Jumbo	8	100	800	60,000
Two-Boom Rock Bolter	8	100	800	60,000
Miscellaneous Equipment			2,000	150,000
Subtotal.				1,395,000
Contingency (15%)				210,000
Subtotal.				1,605,000
350 men/shift @ 100 cfm each.				35,000
Contingency (15%)				6,000
Subtotal.				1,646,000
Leakage (45%)				741,000
Total.				2,387,000

The law requires that 2,387,000 cfm of air is present. To insure the safety of men, control dust, and to maintain a sufficient quantity of air, 2,400,000 cfm is regulated through the mine. Two large 12-foot axial-vane fans in two 16-foot diameter shafts are used to exhaust air out of the mine.

B.3 MAJOR EQUIPMENT SELECTION

The following equipment selection determinations for major items are based on best estimates of equipment availability, operator efficiency, and equipment performance. Availability of equipment varies depending on the type of work and handling it receives during duty cycles. In general, availabilities between 65 and 85% have been applied to various types of equipment in order to determine the number of standby machines needed.

Operator efficiency has been assumed constant for all equipment and is estimated by considering a five-hour work shift (75% efficiency with one hour lunch period). A workday consists of three, eight-hour shifts, seven days per week. Equipment performance data has been taken from vendor data publication. Where necessary, modifications have been made to take into account special mining conditions.

B.3.1 Drilling Equipment

For use in advance entry and pillar mining, a drilling jumbo had to be selected that could drill a 40 by 90-foot face with only one set up. After personal communication with a major manufacturer a prototype drilling jumbo was selected. This jumbo is an automatic four-boom rotary hydraulic drilling jumbo that can efficiently drill out the 40 by 90-foot round. Holes will be drilled using four-inch diameter drag bits. The estimated time to completely drill a round is 1.9 hours assuming an average penetration rate of six feet per minute. A total of five, four-boom drilling jumbos will be maintained with an estimated overall availability of 80% (four in use at any one time).

Benches drilled in the advance entry and pillar mining will be accomplished using two-boom vertical hydraulic drilling jumbos. This piece of equipment is presently available. Holes are drilled using four-inch diameter drag bits with a penetration rate of about eight fpm. The estimated time to drill out a 30-foot high by 90-foot wide bench is 0.5 hour. There are six vertical drill jumbos maintained with an estimated overall availability of 70% (four in use at any one time).

The chamber and pillar mine design will make use of a two-boom rotary hydraulic drilling jumbo similar to the one currently operating in a test mine rear Grand Junction, Colorado. The drilling jumbo is used for heading round drilling and fan drilling of the chambers. Holes are drilled using three-inch diameter drag bits. The estimated time to drill a 30 by 20-foot heading round is 0.5 hour assuming an average penetration rate of six feet per minute. A chamber crosscut 30 by 20 by 450 feet long, is estimated to be completely fan drilled in four days. A total of eight drilling jumbos will be maintained with an estimated overall availability of 80% (six in use at any one time).

B.3.2 Haulage Equipment Selection

In selecting a primary underground haulage system to be used in both the advance entry and pillar and chamber and pillar mining, two systems were investigated: truck and conveyor belt. To help decide which of these systems would be most efficient, the advantages, disadvantages and cost were compared. As a result of the initial evaluation, a 60-inch floor mounted conveyor system was chosen because of the production requirements and the ability to easily handle large tonnages.

A secondary haulage system then was selected. The equipment evaluated were front end loaders and load-haul-dump machines (LHD). Because the maximum haulage distance was estimated to be more than 1000 feet in both mining systems, the LHD's were chosen. LHD's size is limited to the dimensions of the haulage entries and the travel speed. An LHD with a payload capacity of 33,000 pounds (12 yd^3), 6.5 feet high and ten feet wide was chosen. The 12 yd^3 LHD can haul up to 15 mph with a fully loaded bucket. To protect the operator from any material that may come lose from the roof, canopies are built on all LHD's. As another safety feature all LHDs are permissible and contain a fire suppression system.

The LHD cycle time for mucking a 90 by 100-foot heading round is shown below:

1. Loading. . . . . 20 sec
2. Turn . . . . 10 sec
3. Travel (1400 feet) . . . . .190 sec
4. Dump . . . . 20 sec
5. Turn . . . . 10 sec

Total . . . . .250 sec

6. Time to muck round (267 trips using three LHD's) . . . . .6.2 hrs

A total of eight heading rounds are being mucked at one time requiring a total of 24 LHD's. A total of four additional LHD's are used to continuously muck the bench rounds. Assuming an availability of 65% a total of 38 LHD's are needed.

LHD cycle times for mucking chamber rounds are estimated as:

1. Loading. . . . . 20 sec
2. Turn . . . . 9 sec
3. Travel (1244 feet) . . . . .170 sec
4. Dump . . . . 18 sec
5. Turn . . . . 9 sec

Total . . . . .225 sec

6. Time to muck fan drill round (253 trips using three LHD's) . . . . . . .5.3 hrs

A total of eight chambers are being mucked at one time requiring a total of 24 LHD's. Four LHD's are used for development. Using an overall availability of 65% a total of 38 LHD's are needed.

B.3.3 Mobile Crushers

Work done at the Anvil Points mine near Rifle, Colorado indicates that blasting fragment size, using powder factors from 0.35 to 0.70, varies in size up to four cubic feet. In addition, blasted oil shale tends to be quite abrasive, breaking into pieces with sharp edges. Since both mining methods use conveyor belts it was necessary to select a crushing system that was portable and had a capacity of approximately 300 tph. Experimental work done in determining the most effective way to crush oil shale showed that an impact type crusher was reasonably efficient. A portable impact crusher, capable of handling pieces as large as 50 by 60 inches and producing a minus six-inch product at 800 to 1200 tph, was selected. Eight units are used for full production and with an availability of 75% a total of ten are required.

B.3.4 Rock Bolting and Scaling Machines

A two-boom electric-hydraulic rock bolting machine is used in both mining systems. In chamber and pillar mining, eight-foot bolts on seven-foot centers are installed in all development entries and it is estimated that it will take 25 minutes to scale and bolt a round. In the chamber drifts, five-foot bolts are installed on five-foot centers. Six machines are needed at full production, and using an estimated 75% availability, a total of eight are required.

The advance entry and pillar mine design has eight-foot bolts on seven-foot centers throughout the mine. It is estimated that it will take approximately two hours to scale and rock bolt a round. Four rock bolting machines are required at full production, for a total of five with a 75% availability.

APPENDIX C

BLOCK CAVING METHODS

C.1 VENTILATION REQUIREMENTS

The ventilation requirements for block caving with slushers and block caving with LHD's differ due to the type of equipment used underground. Each system is analyzed separately considering gassy mining conditions and complying with Colorado mining laws (Colorado Bureau of Mines, 1971 and State of Colorado, 1966).

C.1.1 Block Caving with Slushers

Description	Quantity	HP/Unit	Total HP	Air Req'd (cfm)
Two-boom hydraulic drill jumbo	4	135	540	41,000
Two-boom medium drill jumbo	4	44	176	13,000
Two-boom fan drill	4	135	540	41,000
FEL 8 yd^3	4	380	1520	114,000
LHD 8 yd^3	8	250	2000	150,000
Miscellaneous	-	-	4000	300,000
Subtotal				773,000
Contingency (15%)				116,000
Subtotal				889,000
280 men/shift @ 100 cfm each				28,000
Contingency (15%)				5,000
Subtotal				992,000
Leakage (45%)				447,000
Total				1,439,000

C.1.2 Block Caving with LHD's

Totals of 1,500,000 and 3,000,000 cfm are used for block caving using slushers and block caving with LHD's, respectively.

Description	Quantity	HP/Unit	Total HP	Air Req'd (cfm)
Two-boom hydraulic drill jumbo	4	135	540	41,000
Two-boom medium drill jumbo	4	44	176	13,000
Two-boom fan drill	4	135	540	40,000
FEL (8 yd^3)	4	380	1,520	114,000
LDH's (5 yd^3)	4	300	1,200	90,000
LHD's (8 yd^3)	31	380	11,780	884,000
Miscellaneous	-	-	4,000	300,000
Subtotal.				1,482,000
Contingency (15%)				222,000
Subtotal.				1,704,000
300 men/shift @ 100 cfm each.				30,000
Contingency (15%)				5,000
Subtotal.				1,739,000
Leakage (45%)				783,000
Total.				2,522,000

C.2 MAJOR EQUIPMENT SELECTION

The following equipment determinations for major pieces of equipment are based on best estimates of equipment availability, operator efficiency, and equipment performance. Availability of equipment varies depending on the type of work and handling it receives during duty cycles. In general, availabilities between 65 and 90% have been applied to various types of equipment in order to determine the number of standby machines needed.

Operator efficiency has been assumed constant for all equipment and is estimated by considering a five-hour work shift (75% efficiency with a one hour lunch period). A work day consists of three eight-hour shifts, seven days per week. Equipment performance data has been taken from vendor supplies information and publications. Where necessary, modifications have been made to take into account special mining conditions.

The following calculations were used to determine the production area required and the number of active blocks needed for full production.

C.2.1 Production Area Required

Block size.	300' X 300' X 550'
Draw rate	0.75'/day
Tonnage factor.	14.3 ft^3/ton
30 year reserves	
85,000 X 355 X 30	905,250,000 tons
Ore in block	
(300 X 300 X 550) ÷ 14.3	3,461,500 tons
Required number of blocks	
905,250,000 ÷ 3,461,500	262
Blocks per level	
262 ÷ 3	90

A total of ten panels having nine blocks each are used for each level. If production is increased to 170,000 tpd, ten panels will be developed on the opposite side of the shaft.

C.2.2 Number of Active Blocks

Block production	
(300 X 300 X 0.75) ÷ 14.3.	4,720 tons/day
Full production blocks	
85,000 ÷ 4,720	18
Total blocks required	18

C.2.1 Slushers and Scrapers

Low profile, hinged blade scrapers were chosen to collect the muck from the draw points and load directly into 80-ton rail cars. The hinged blade gives maximum flexibility during the retreat cycle by enabling the scraper blade to easily glide over broken muck.

A 84-inch scraper is used with a 150 hp slusher. The following calculations show the number of units required to meet total production of about 85,000 tpd.

Rope speed.	200 feet/minute
Total length of haul.	300 feet (round trip)
Total time taken for one load	1.5 minutes
Delay time.	0.5 minutes/trip
Total cycle time.	2.0 minutes

Tonnage hauled per load

100.0 ÷ (14.3 X 1.3) . . . . 5.3 tons

Tonnage per hour (5.3 X 30) . . . . 160 tph

Number of slushers required

85,000 ÷ 816 104

One-hundred-four slushers can meet the production which is less than the number of slusher lines selected (108).

C.2.2 Block Caving with LHD's

LHD size is limited to the dimensions of the haulage drifts and the machine turning radius. The LHD's chosen in these calculations have a payload capacity of 24,000 lbs (12 tons or 8 yd^3) a height of 6.3 feet, width of 8.2 feet, length of 33 feet, and a turning radius of 26 feet. Loaded the LHD's are capable of negotiating a 52% grade. The number of LHD's required for 85,000 tpd are calculated assuming:

Average haul.	250 feet
Average LHD speed	4 mph
Load time	60 sec
Haul time	45 sec
Dump time	30 sec
	135 sec

Number of tons per trip. 9.28

Number of trips per day. 15

Tonnage required from each block 9,440

Number of LHD's per block

9,440 ÷ (400 X 9.28) 2.5

Therefore use three LHD's per block or 27 LHD's for the nine blocks in production.

Selection of LHD's to muck haulage drifts panel drifts, production lines and production crosscuts during preproduction is based on the number of available faces rather than maximum output per LHD. Using this method of analysis and considering an 80% availability, four 8 yd^3 LHD's are needed.

Selection of LHD's to muck the undercut drifts and crosscuts is done taking into account available faces and working space. Using this method and selecting a 65% availability, six five-yd^3 LHD's are needed.

C.2.3 FEL's

FEL's are used only on the haulage and ventilation levels. FEL size is limited to the dimensions of the drifts and the machine turning radius. Eight cubic yard FEL's are used in both levels and load the ore directly into rail cars. The number of units required is based on faces available rather than maximum output.

Ventilation drifts.	2
Haulage drift	2
Standby (65% availability).	2
Total.	6 - 8 yd^3 FEL's

C.2.4 Train Locomotives

Calculations for trolley locomotives for the block caving are similar to those of appendix A.4.3. The only difference lies in the number of cars the locomotives may be pulling and the braking requirements. In the block caving design the locomotives may have occasion to pull more than ten cars over a much longer distance (two miles). A 50-ton trolley locomotive was selected. Assuming a two mile haul and an average speed of eight miles per hour, one train has a cycle time of 15 minutes. At 640 tons/train and 15 minutes per train, it will take five locomotives to effectively handle the required 85,000 tpd. With one locomotive as a standby a total of six 50-ton locomotives are required.

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