CONTACT INFORMATION
Mining Records Curator
Arizona Geological Survey
416 W. Congress St., Suite 100
Tucson, Arizona 85701
602-771-1601
http://www.azgs.az.gov
inquiries@azgs.az.gov

The following file is part of the Grover Heinrichs Mining Collection

ACCESS STATEMENT

These digitized collections are accessible for purposes of education and research. We have indicated what we know about copyright and rights of privacy, publicity, or trademark. Due to the nature of archival collections, we are not always able to identify this information. We are eager to hear from any rights owners, so that we may obtain accurate information. Upon request, we will remove material from public view while we address a rights issue.

CONSTRAINTS STATEMENT

The Arizona Geological Survey does not claim to control all rights for all materials in its collection. These rights include, but are not limited to: copyright, privacy rights, and cultural protection rights. The User hereby assumes all responsibility for obtaining any rights to use the material in excess of "fair use."

The Survey makes no intellectual property claims to the products created by individual authors in the manuscript collections, except when the author deeded those rights to the Survey or when those authors were employed by the State of Arizona and created intellectual products as a function of their official duties. The Survey does maintain property rights to the physical and digital representations of the works.

QUALITY STATEMENT

The Arizona Geological Survey is not responsible for the accuracy of the records, information, or opinions that may be contained in the files. The Survey collects, catalogs, and archives data on mineral properties regardless of its views of the veracity or accuracy of those data.

DRAFT 4/22/74

REPORT ON THE FEASIBILITY OF COPPER PRODUCTION IN THE LISBON VALLEY, SAN JUAN COUNTY, UTAH

ERRATA SHEET

1. Page II-3, first line

   Reads: "...would be apprlximately $1000/acre or $20,000/claim which would...."

   Should Read: "...would be approximately $1000/claim which would...."

2. Appendix B, GROUP: Climas, MORTGAGE PAYMENTS:

   Reads: "$4500 final payment due 6/17/74"

   Should Read: "None"

3. Appendix B, GROUP: Gamma, ROYALTIES: first sentence

   Reads: $.15 per wet ton B & K. .70 per wet ton to...."

   Should Read: $0.15 per wet ton B & K; $0.70 per wet ton to...."

4. Appendix B, GROUP: Anderson, ROYALTIES:

   Reads: "State royalties of 3% on $0.99 NSR 6% on $10.99 NSR with milling costs deducted. $0.50-.90 per day ton on soluable copper, 5% NSR on sulfide ore."

   Should Read: "State royalties of 3% on $0.99 NSR, 6% on $10.99 NSR with milling costs deducted. $0.50 - $0.90 per dry ton on soluble copper, 5% NSR on sulfide ore."

5. Appendix B, GROUP: Dakota, ROYALTIES:

   Reads: "...on 0.2% Cu ore. 6% on all other minerals except 7½% on yellow cake."

   Should Read: "...on 2.0% Cu ore. 6% on all other minerals except yellow cake which is 7½%."

6. Appendix B, GROUP: Big Wheel, ROYALTIES:

   Reads: "$0.02½ per pound ore recovered"

   Should Read: "$0.02½ per pound copper recovered"

7. Appendix B, GROUP: Big Indian, ROYALTIES:

   Reads: "1% NSR on EMJ price to Big Indian Uranium. $0.01 per pound for milling on premises."

   Should Read: "5% NSR on EMJ price to Big Indian Uranium. (1% NSR or $0.01 per pound of copper (?) for milling on premises, not of further consequence to project.)"

8. Appendix B, GROUP:Quatro, ASSEESMENT WORK:

Reads: "Amount $ 2100 ...."

Should Read: "Amount $ 400 ...."

9. Appendix B, GROUP: P.B.L., ANNUAL LEASE PAYMENT:

Reads: "...(Included with PBL Payment.)"

Should Read: "...(Included with Red payment.)"

10. Appendix B, GROUP: Security, ROYALTIES:

Reads: "$1.00/wet ton to Costanza"

Should Read: "$1.00/wet ton to Costanza toward payout"

11. Appendix B, GROUP: Cooley, ROYALTIES:

Reads: "$0.20/pound on contained copper 5% of value less milling and refining."

Should Read: "$0.02/pound on contained copper; 5% of value less milling and refining for other minerals."

12. Appendix B, UNPATENTED CLAIMS

Reads:

....	
CCC Fractions	4
Patented	
Sharon-Sherrie Groups	25
Red	47
PBL-Hot-Cool-Hop-Cap	387
Security	43

Should Read:

...	
CCC Fractions	4
<u>South Area</u>	
PBL-Hot-Cool-Hop-Cap	387
Security	43
<u>Centennial Area</u>	
Sharon-Sherrie Groups	25
Red	47
Climax	2
B&K	19
...	

REPORT ON THE

FEASIBILITY OF COPPER PRODUCTION

IN THE

LISBON VALLEY, SAN JUAN COUNTY, UTAH

Prepared by

CENTENNIAL DEVELOPMENT COMPANY
34 Century Park-Way
Salt Lake City, Utah 84115

March, 1974

INDEX OF REPORT

Introduction and Summary Conclusion . . . . . Section I

Property Description . . . . . Section II

Geology . . . . . Section III

Ore Reserves . . . . . Section IV

Mining . . . . . Section V

Metallurgy . . . . . Section VI

Water and Power . . . . . Section VII

Organization and Administration . . . . . Section VIII

Marketing . . . . . Section IX

Financial Analysis . . . . . Section X

INDEX OF APPENDICES

Reconciliation of Lisbon Valley Program Expenditures . . . Appendix A

Detailed Description of Property Holdings . . . . . Appendix B

Metallurgical Addenda . . . . . Appendix C

Smelting Metallurgy . . . . . Appendix D

Computer Mine Plan Summaries . . . . . Appendix E

INDEX OF PLATES

FOR

LISBON VALLEY COPPER MAP PORTFOLIO

Property Map (Overlay) 1.1A

Regional Geologic Map 1.1

General Mine-Plant Layout (Overlay) 2.1A

Local Geologic Map 2.1

Regional Geologic Cross Sections 2.2

Drill Hole and Local Geologic Cross Sections Index 3.1

Local Drill Hole-Assay Cross Sections (Overlays)
2100 N - 420 S (60 ft. intervals) 4.1A-4.43A

Local Geologic Cross Sections
2100 N - 420 S (60 ft. intervals) 4.1-4.43

Computer Plan Maps (10 ft. bench height) showing Stratigraphy
6515 ft. elevation - 6095 ft. elevation (20 ft. intervals) . . 5.1-5.22

Pincock, Allen & Holt, Inc. Pit Planning Level Maps (Overlays)
6495 ft. elevation - 6115 ft. elevation (20 ft. intervals) . . 6.1A-6.20A

Computer Plan Maps (20 ft. bench height) showing total Cu (%)
6495 ft. elevation - 6115 ft. elevation (20 ft. intervals) . . 6.1-6.20

Conformed Geologic Ore Plan, D. G. Bryant
6395 ft. elevation 6.5B
6295 ft. elevation 6.11B

Geologic Ore Cross Sections, D. G. Bryant 7.1

Pincock, Allen & Holt, Inc. Pit Mining Plans 8.1-8.7

Pincock, Allen & Holt, Inc. Pit Planning Cross Sections 9.1-9.7

LISBON VALLEY COPPER

1. Centennial Pit Area
2. Planned Mill Site
3. Tailings Area
4. GTO Pit
5. Sentinal Area

Figure 1

JOSEPH C. BENNETT
CHAIRMAN OF THE BOARD

34 CENTURY PARK-WAY
SALT LAKE CITY, UTAH 84115
TELEPHONE 801/487-5651

CENTENNIAL DEVELOPMENT COMPANY

Keystone Wallace Resources
5 Denver Technological Center
Englewood, Colorado 80110

Attention: George M. Wallace

Gentlemen:

Lisbon Valley Feasibility Report
Introduction and Summary Conclusion

In accordance with the terms of the Option Agreement of December 13, 1972, as subsequently amended on September 24 and December 24, 1973, Centennial Development Company, as optionee, has completed its investigation of copper ore reserves and mining costs on the Lisbon Valley properties of Keystone Wallace Resources (KWR), the optionor, and herewith submits its report on the feasibility of copper production. The accompanying map portfolio is an integral part of this report.

Centennial's investigations, which began in January, 1973, and the preparation of this report have been under the general supervision of Raymond E. Grant, Manager-Mine Project Evaluation, reporting to the undersigned. Mr. Grant divided his time between field and office and provided valuable geological interpretations which led to ore development. Ably assisting throughout the project has been James Quigley, Vice President, who provided geologic guidance as well as acting as

liaison with consulting metallurgists and plant designers. Centennial's engineer, Robert N. Breckenridge, did much of the early drill hole evaluations and later contributed to ore reserve determinations. Geologist R. H. Godbe was directly in charge of field drilling operations and he and Geologist Alan G. Seelos contributed to geologic interpretations of drill hole data. Mr. Seelos was also responsible for drafting and preparation of the map portfolio. Robert W. Gilmore, Centennial's property superintendent at Lisbon Valley, assisted in various phases of the project.

Donald G. Bryant, consulting geologist of Littleton, Colorado, has participated in the project from inception. He greatly contributed to the geologic conception underlying the exploration drilling and in developing guides to finding new ore along the Lisbon Valley fault trend. In the later stages of the project, Dr. Bryant performed valued services in validating computer calculated ore reserves by hand checking selected ore sections.

Early in the study the consulting firm of Pincock, Allen & Holt, Inc. of Tucson, Arizona, was retained to undertake mine design and costing studies, to oversee metallurgical test work at Mountain States Engineering in Tucson and to translate test results into mill flow sheet design and operating cost estimates. W. K. Pincock, P. F. Allen, Francis L. Holderreed and Thomas R. Couzens were, at various stages, deeply involved in the project and their reports comprise the mining and a portion of the metallurgy sections herein. The engineering firm

of Roberts & Schaefer Company of Chicago, Illinois, and more particularly this firm's resident manager in Salt Lake City, Bernard Bernstrom, was assigned the task of preliminary plant design and capital cost estimates.

Throughout the study Centennial relied heavily on the use of the University of Utah's 1108 Univac computer, utilizing computer programs developed by Mintec, Inc. of Tucson whose principals, A. Frederick Banfield, Jr. and William L. Meyer, contributed much of their time in the early stages of the office work.

On behalf of Centennial directors and shareholders, I wish to acknowledge with grateful appreciation the valued contributions and hard work of all employees and consultants, mentioned and unmentioned, to this effort. It should also be acknowledged that at all times full cooperation was received from the KWR staff both in Moab and in Denver.

As Appendix A to this report, there appears a reconciliation of expenditures incurred by Centennial on the investigation through February, 1974, including an estimate of those to be incurred in the current month.

In Centennial's view, a mining project cannot be considered economically viable unless a study such as this offers ample evidence that the equity investor can achieve a minimum 15% discounted cash flow rate of return on his investment on a no-loan basis. I am pleased to report that, the terms contemplated in the December 18, 1972 Option Agreement aside, the Lisbon Valley project, <u>on a project basis</u>, meets this test under the following economic and technical parameters:

1. Copper Price - 68¢ per pound
2. Capital Investment ± $10,500,000
3. Operating Costs - $3.61 per ton
4. Ore Reserves @ 0.4% Cu cut-off - 6,348,000 tons @ 0.80% Cu plus low grade stockpile for reclaim - 1,997,000 tons @ 0.35% Cu
5. Stripping Ratio - 3.61:1
6. Daily Mining Rate - 2,500 tons
7. Open Pit Mine Life - 7 years plus 2 years for low grade stockpile reclaim

Under the terms of the Limited Partnership Agreement contemplated by the above-referenced Option Agreement, Centennial, as general and managing partner responsible for all development capital, would not achieve its desired 15% rate of return unless the producer price of copper could be sustained at 82¢ per pound, all other parameters being equal. On a sensibly financed basis (see discussion Section X), such return might be achieved at a 75¢ price. On the other hand, at the present producer price, the investment criterion can be approximated if the financing partner were permitted full depletion allowance and return of capital from 100% of cash flow and the limited partner thereafter received 15% of net profits as defined in the Agreement.

Respectfully submitted,

J. C. Bennett

Joseph C. Bennett

March 22, 1974

SECTION II

PROPERTY DESCRIPTION

Introduction

Under agreement dated December 13, 1972, Centennial Development Company holds under option from Keystone Wallace Resources (KWR) unpatented and patented mining claims and a state lease covering approximately 17,000 acres in the Lisbon Valley area of San Juan County, Utah. The Lisbon Valley area is identified in the index map, Figure 2. The Big Indian-Lisbon Valley area came into prominence during the uranium rush on the Colorado Plateau in the early 1950's. The now famous Mi Vida Mine led to the rapid discovery of other large uranium deposits, often with associated copper, in the Lisbon Valley area. The Mi Vida property is located about seven and one-half airline miles northwest of the area which is the subject of this report. The optioned properties are distributed along 15 miles of the Lisbon Valley mineralized belt and are believed to cover approximately 75% of all the lands in the Lisbon Valley area which might be considered as having potential for commercial copper deposits.

The location of the claim groups and lease are displayed on an overlay covering the geological map---Portfolio Plate 1-A. Terms under which KWR holds the mineral and surface rights to the properties are detailed in Appendix B. The focus of this feasibility report is on approximately 280 acres (within the SW¼, Section 25 and the NW¼ Section 36, T30S, R25E, Salt Lake Base and Meridian) or 1.6% of the

duplicate page

total property position, which overlie the so-called Centennial ore body and proposed concentrator, tailings and waste dump location (see p.II-7).

For assessment purposes, the property holdings are divided into four noncontiguous groups. By locating claim fractions and acquiring other properties, it is anticipated that the number of assessment groups can be reduced to three.

Claim documents, leases and titles have variously been examined by John W. Horsley of the firm of Moyle and Draper, Salt Lake City, and Randy L. Parcell of the firm of Holland and Hart, Denver, during the cource of property acquisition and renegotiation. No title opinions have been requested of attorneys as of the date of this report, however it is Centennial's belief that such opinions, if requested, will verify that the claims are properly located and recorded and all other leases and agreements clearly give to KWR, and its successors, the right to mine copper mineral and utilize the surface in connection therewith.

A claimholder has full rights to mine ore from unpatented claims while patented claims have added security of ownership, by occupying those unpatented claims Centennial does not forsee any problems that patenting might alleviate.

In the opinion of Centennial, it would be in the economic interest of the partnership to maintain claims in an unpatented status. Such status costs $100 per claim as assessment toward which operating costs can be applied. This would fully alleviate assessment burdens on claims contiguous to mining operations. The cost of patenting claims, however,

would be approximately $1000/acre or $20,000/claim which would then subject them to property taxes of $5/claim each year.

Centennial Pit Area

The area overlying the recently discovered reserves and their possible extensions has been designated the Centennial Pit area and was formerly known as the Micro-Costanza Pit. It comprises the Section 36 state lease, the Patterson Ranch and the following unpatented claim groups: Climax, Alpha, Animal, Gamma, Sentinal, Globe, Dakota, Big Wheel and KWR. In addition, claims CW1-18 have been located over the Climax and Alpha claims to insure against any improper location of the Climax and Alpha claims. The schedule on the following pages details the terms of the property agreements covering the Centennial ore body and known or projected mineral outside the proven ore reserve area. All cash payments as royalties or purchase installments owing to lessees and owners are reflected fully in the cash flow statements in Section X (see Table I).

The balance of the property is covered by claim groups which are being gradually surveyed, reposted and developed by assessment work. The northern claims, covering the Big Indian operations of KWR, lie on the border of active uranium mining. The validity of these claims, some of which are patented, have been well established over a period of 20 years. In the central area of Lisbon Valley there are claim conflicts which will have to be resolved if and when active exploration begins. By and large these conflicts exist between two claim groups both of which are under lease to KWR.

South of the Centennial ore body, most claims under lease are poorly defined with a great many claim corners lacking. Nevertheless,

Exxon 5 deep holes $35,000

Sept

South Group

Harding $17,500 June 30th

Anderson $1000 June 30
$1000 July 31

Wilcox $200 June
$200 July

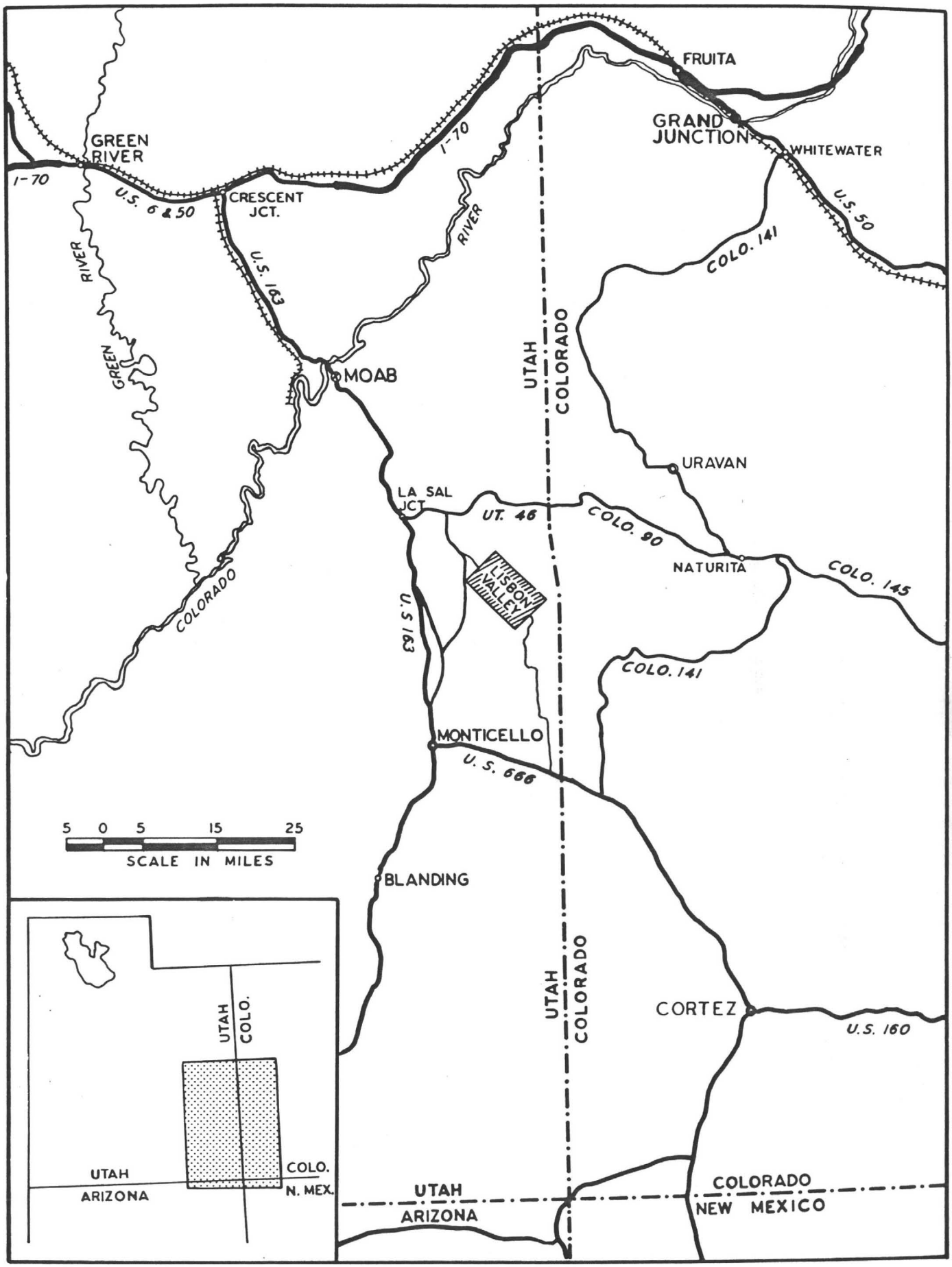
No. Big Indian $100 June
100 July

$20,100

TABLE II-1

SCHEDULE OF QUARTERLY LAND PAYMENTS TO PRODUCTION

	1974				1975				1976					Yearly Sustaining
BY OWNER	1st	2nd	3rd	4th	1st	2nd	3rd	4th	1st	2nd	3rd	4th		
Climax		4,500											Purchase	-
Harding		17,500				17,500				7,500			Purchase	-
Costanza						15,000				15,000			Purchase	-
Anderson	3,000	3,000	3,000	3,000	4,500	4,500	4,500	4,500	6,000	6,000	6,000	6,000	Purchase	30,000
Tintic Uranium							12,000	500	750	750	750	3,750	Work Req.&Roy.	3,000
Stuart, Fisher, etc.				1,000				1,000				1,000	Min. Roy.	1,000
Costanza, Patterson, et.al.						15,000				15,000			Min. Roy.	15,000
Costanza, Lema, Coates, et.al.						2,000				2,000			Min. Roy.	2,000
Cooley					1,000								Purchase	-
Wilcox Surface		600	600	600	1,600	600	600	600	600	600	600	600	Rental	2,400
Micro Copper										2,500			Purchase	2,500
BY CLASSIFICATION FOR TAXES														
Capital Minumum Land Payments	3,000	25,000	3,000	3,000	5,500	37,000	4,500	4,500	6,000	31,000	6,000	6,000	Purchase	30,000
Minimum Royalties				1,000		17,000		1,500	750	17,750	750	1,750	Min. Roy.	21,000
Rental Big Indian		600	600	600	1,600	600	600	600	600	600	600	600	Rental	2,400
Work Commitment							12,000					3,000	Work Com.	-
TOTAL LAND EXPENDITURES	3,000	25,600	3,600	4,600	7,100	54,600	17,100	6,600	7,350	49,350	7,350	11,350		$53,400
						85,400				75,400				



Index map showing location of Lisbon Valley

Figure 2

DRAFT 4/22/74

REPORT ON THE
FEASIBILITY OF COPPER PRODUCTION
IN THE
LISBON VALLEY, SAN JUAN COUNTY, UTAH

ERRATA SHEET

1. Page II-3, first line

   Reads: "...would be apprlximately $1000/acre or $20,000/claim which would...."

   Should Read: "...would be approximately $1000/claim which would...."

2. Appendix B, GROUP: Climas, MORTGAGE PAYMENTS:

   Reads: "$4500 final payment due 6/17/74"

   Should Read: "None"

3. Appendix B, GROUP: Gamma, ROYALTIES: first sentence

   Reads: $.15 per wet ton B & K. .70 per wet ton to...."

   Should Read: $0.15 per wet ton B & K; $0.70 per wet ton to...."

4. Appendix B, GROUP: Anderson, ROYALTIES:

   Reads: "State royalties of 3% on $0.99 NSR 6% on $10.99 NSR with milling costs deducted. $0.50-.90 per day ton on soluable copper, 5% NSR on sulfide ore."

   Should Read: "State royalties of 3% on $0.99 NSR, 6% on $10.99 NSR with milling costs deducted. $0.50 - $0.90 per dry ton on soluble copper, 5% NSR on sulfide ore."

5. Appendix B, GROUP: Dakota, ROYALTIES:

   Reads: "...on 0.2% Cu ore. 6% on all other minerals except 7½% on yellow cake."

   Should Read: "...on 2.0% Cu ore. 6% on all other minerals except yellow cake which is 7½%."

6. Appendix B, GROUP: Big Wheel, ROYALTIES:

   Reads: "$0.02½ per pound ore recovered"

   Should Read: "$0.02½ per pound copper recovered"

7. Appendix B, GROUP: Big Indian, ROYALTIES:

   Reads: "1% NSR on EMJ price to Big Indian Uranium. $0.01 per pound for milling on premises."

   Should Read: "5% NSR on EMJ price to Big Indian Uranium. (1% NSR or $0.01 per pound of copper (?) for milling on premises, not of further consequence to project.)"

8. Appendix B, GROUP:Quatro, ASSESSMENT WORK:

Reads: "Amount $ 2100 ...."

Should Read: "Amount $ 400 ...."

9. Appendix B, GROUP: P.B.L., ANNUAL LEASE PAYMENT:

Reads: "...(Included with PBL Payment.)"

Should Read: "...(Included with Red payment.)"

10. Appendix B, GROUP: Security, ROYALTIES:

Reads: "$1.00/wet ton to Costanza"

Should Read: "$1.00/wet ton to Costanza toward payout"

11. Appendix B, GROUP: Cooley, ROYALTIES:

Reads: "$0.20/pound on contained copper 5% of value less milling and refining."

Should Read: "$0.02/pound on contained copper; 5% of value less milling and refining for other minerals."

12. Appendix B, UNPATENTED CLAIMS

Reads:

....	
CCC Fractions	4
Patented	
Sharon-Sherrie Groups	25
Red	47
PBL-Hot-Cool-Hop-Cap	387
Security	43

Should Read:

...	
CCC Fractions	4
<u>South Area</u>	
PBL-Hot-Cool-Hop-Cap	387
Security	43
<u>Centennial Area</u>	
Sharon-Sherrie Groups	25
Red	47
Climax	2
B&K	19
...	

Essex

REPORT ON THE FEASIBILITY OF COPPER PRODUCTION IN THE LISBON VALLEY, SAN JUAN COUNTY, UTAH

Prepared by:
CENTENNIAL DEVELOPMENT COMPANY
34 CENTURY PARK WAY
SALT LAKE CITY, UTAH 84115

MARCH, 1974

TITLES - LEASES - ROYALTIES FOR KNOWN AND PROJECTED RESERVES

Mineral Property	Owner	Lessee	Royalty	Mortgage or Annual Payment
A) CENTENNIAL OREBODY (Planned Production)				
Climax 1 & 2	KWR	-	-	Final Payment of $4500 due 6-17-74.
Alpha 1-8	KWR	-	-	Payments Due: $17,500, 6-30-74; $17,500, 6-30-75; final payment $7500, 6-30-76.
Sentinal 1	CD-KWR	-	-	-
Animal Group	Brinton & Knowles	Costanza* (KWR)	$0.05/wet ton milled	*Costanza $0.70/wet ton of ore milled.
Section 36, E½, NW¼	State of Utah	Costanza* (KWR)	Graduated; Estimate on average grade: $0.055/ton milled @$0.55/lb. Cu $0.20/ton milled @ $0.85/lb. Cu	*Costanza $1.00/wet ton of ore milled.
B) PROJECTED RESERVES (Known Mineralization - anticipated production)				
Sentinal 2-10	Brinton & Knowles	Costanza* (KWR)	$0.15/wet ton milled	*Costanza $0.30/wet ton of ore milled.
GTO, Section 36, SW¼	State of Utah	Anderson (KWR)	5% NSR to Anderson; graduated as above for State after buy out.	NSR royalty paid to Anderson toward a $600,000 buy out, then to Utah as Section 36 above. Minimum payments due; $1000/mo-1974, $1500/mo-1975, $2000/mo-1976, $2500/mo-1977 and thereafter

*J.J. Costanza holds mineral leases on several properties. The lease rights to copper have been subleased by KWR with a $300,000 buyout provision. A present balance of $68,000 toward buy out should be met by $15,000 minimum annual payments in 1974, 1975 and royalties on ore produced in the first year from the Animal claims and Section 36.

Mineral Property	Owner	Lessee	Royalty	Mortgage or Annual Payment
B) PROJECTED RESERVES (Continued)				
Gamma 1 & 2	Brinton-Knowles	Costanza* (KWR)	$0.15/wet ton milled	*Costanza $0.85/wet ton of ore milled
Globe 1-10	Brown	KWR	2.5% NSR	-
Dakota	Stewart	KWR	5.0% Gross Value (E&MJ Prices)	Minimum Annual Royalty; $1000 beginning 3/1/74
Big Wheel	Micro Copper	CD-KWR	$0.25/lb. Cu	$75,000 Buy Out. Minimum Annual Royalties; $2500 - 1976-1977 $5000 - 1978-1982 Balance of buy out due May 31, 1982
KWR 1-13	KWR	-	-	-
C) PROJECTED RESERVES (Mineralization geologically projected)				
Patterson Ranch	Patterson	Tintic Uranium Co.	6% on NSR value less Milling Costs	Minimum Royalty; $250/mo starting 10/75

*J.J. Costanza holds mineral leases on several properties. The lease rights to copper have been subleased by KWR with a $300,000 buy out provision. A present balance of $68,000 toward buy out should be met by $15,000 minimum annual payments in 1974, 1975 and royalties on ore produced in the first year from the Animal claims and Section 36.

these claims have been shown on district maps for a great number of years, the annual assessment work has been continuously performed and the county records show continuous entries.

The area between the Centennial ore body and the claims in the Big Indian area to the north present a similar situation. Defects may be found, but they will be minor. The claims can be defended against any encroachment by operators seeking a substantial position. This is not to warrant that fractions do not exist and that others will not locate them in advance of Centennial's staff.

Property Not Under Option or Lease

Some distance from the Centennial ore body but within the area of interest, three claim groups: Hi Lime (8 claims), Profit (8 claims) and Texas Uranium (80 plus or minus 50 claims) are not under lease or option. In view of their exploration potential, steps are being taken to acquire these lands.

WHAT IS STEPS

Surface Use

All of the mineral claims and leases permit full use of the surface for exploration, mining and milling purposes. Projected mine waste dumps will occupy the Sentinal 4 and a portion of the Animal group, and the mill and tailings area will occupy portions of Section 36, Gammas 1 and 2 and the Coyote Claim.

In the case of Section 36 and the Patterson Ranch, the surface is owned by Max Wilcox of Moab who, for $200 per month, has agreed to full

use of his surface rights on these fee lands as well as use of whatever rights he might have as the holder of grazing permits in lower Lisbon Valley. Additionally, negotiations are under way for the Wood Ranch surface (320 acres) and mineral plus the SW¼-NW¼ and NW¼-SW¼ (80 acres) of Section 6, T31S, R26E. It is anticipated that this land can be purchased, including mineral rights for $125,000 in ten equal annual installments. The surface would be useful, although the Wood's ranch land has a low mineral potential. The 80 acres in Section 6 are adjacent to reportedly successful drilling by Atlas Corporation. At present, they are leased to Atlas and bear 12½% gross royalty to Wood.

SUMMARY OF ACREAGE TO BE OCCUPIED

	Acres
Pit Area 1000 x 2500 = 2,500,000	
Climax, Alpha, Animal and Section 36	59
Dump Area	
Sentinal 4, Animal	80
Mill Site Area	
Section 36, Gamma 1 and 2, Coyote	13
Tailings Area	
Section 36 (40 ft. at low end)	132
	284

Environmental Considerations

The properties lie within a recognized mining district of long history with a number of mines still active. The new Utah land planning law places a great deal of responsibility on local governments for land use planning. The county commission is sympathetic to mining. No adverse policies are anticipated.

Under present laws and regulations, the owners of patented and unpatented mining claims and lessees of state and fee lands have rights to full use of the surface for mining purposes. The ore reserve lies on unpatented claims which were located prior to 1955 after which the use of surface was restricted somewhat with regard to timber and water. The bulk of the proprty beyond the ore reserve area consists of unpatented claims located subsequent to the 1955 law. These claims fall under the Multiple Use law which gives the BLM rights to timber, water and surface uses not required for mining purposes.

There are presently no federal or state statutes requiring mined land reclamation for western open pit metal mines. Accordingly, no costs for restoration of contours, etc. have been factored into mine operating costs in this study. Should reclamation legislation be adopted, it is presumed attendent costs will be reflected in and covered by higher copper prices.

Discharge of water into streams and into the water table and of effluents into the atmosphere are subject to a host of regulations, all of which will be complied with in Lisbon Valley. Actually, there are

no flowing streams in the area or wells for any purposes other than stock watering. Water for stock watering is planned to be made available to the BLM. Containing tanks or ponds would be provided by stockmen.

The plant, mine waste dumps and tailings basin have been laid out to utilize natural contours and basins thus reducing environmental impact. There will be no water discharged into the water table except by potential seepage from the bottom of the tailings basin. The dust from trucks and equipment on the haul roads and pits will be controlled by watering. There is no way to control dust from blasting. It is probable that this will be minimal as the pit passes from dry surface rock to the levels below water table.

The BLM does have environmental responsibility on the lands crossed by the 11 miles UP&L power line from the Lisbon substation to the plant site. Utah Power & Light have been authorized to service this right of way and to make proper application for a right-of-way permit. In the event the project does not go ahead the cost of this application, which should not exceed $3000, will be charged to Centennial. The right-of-way application lies under the jurisdiction of the Monticello office of the BLM who have indicated their report will permit construction of the line with no further environmental study required.

SECTION III

GEOLOGY

A. General Geology*

The Lisbon Valley anticline is the structural environment of copper mineralization and is the most southern of five salt structures identified in the Paradox Basin of southeastern Utah and southwestern Colorado. These anticlinal structures are localized within the northwest-southeast trending belt, and consist of collapsed cores of salt which had been introduced into intermixed gypsum and clastic sediments. The evaporites of Pennsylvanian age are of an unknown thickness, however, wells drilled in the salt cores of some anticlines have reportedly penetrated as much as 11,000 feet of salt without reaching the base of the evaporite section. The original undisturbed thickness of the evaporite sequence is thought to be in excess of 5000 feet.

The Lisbon Valley anticline lies about four miles southwest of LaSal, Utah (see Index Map Figure 2) and south of laccolith intrusions in the LaSal mountains. The anticline extends in a southeasterly direction for approximately 16 miles (see Figure 3).

A prominent normal fault transcends the anticline throughout its length. The downthrown block lies to the northeast of the steep-to-moderately inclined, northeast dipping fault plane. On the west side of the anticline this "Lisbon Valley" fault has a stratigraphic throw in excess of 3800 feet. Lesser associated faults also pass

*Condensed from Guidebook No. 9 to the Geology of Utah entitled "Uranium Deposits and General Geology of Southeastern Utah."

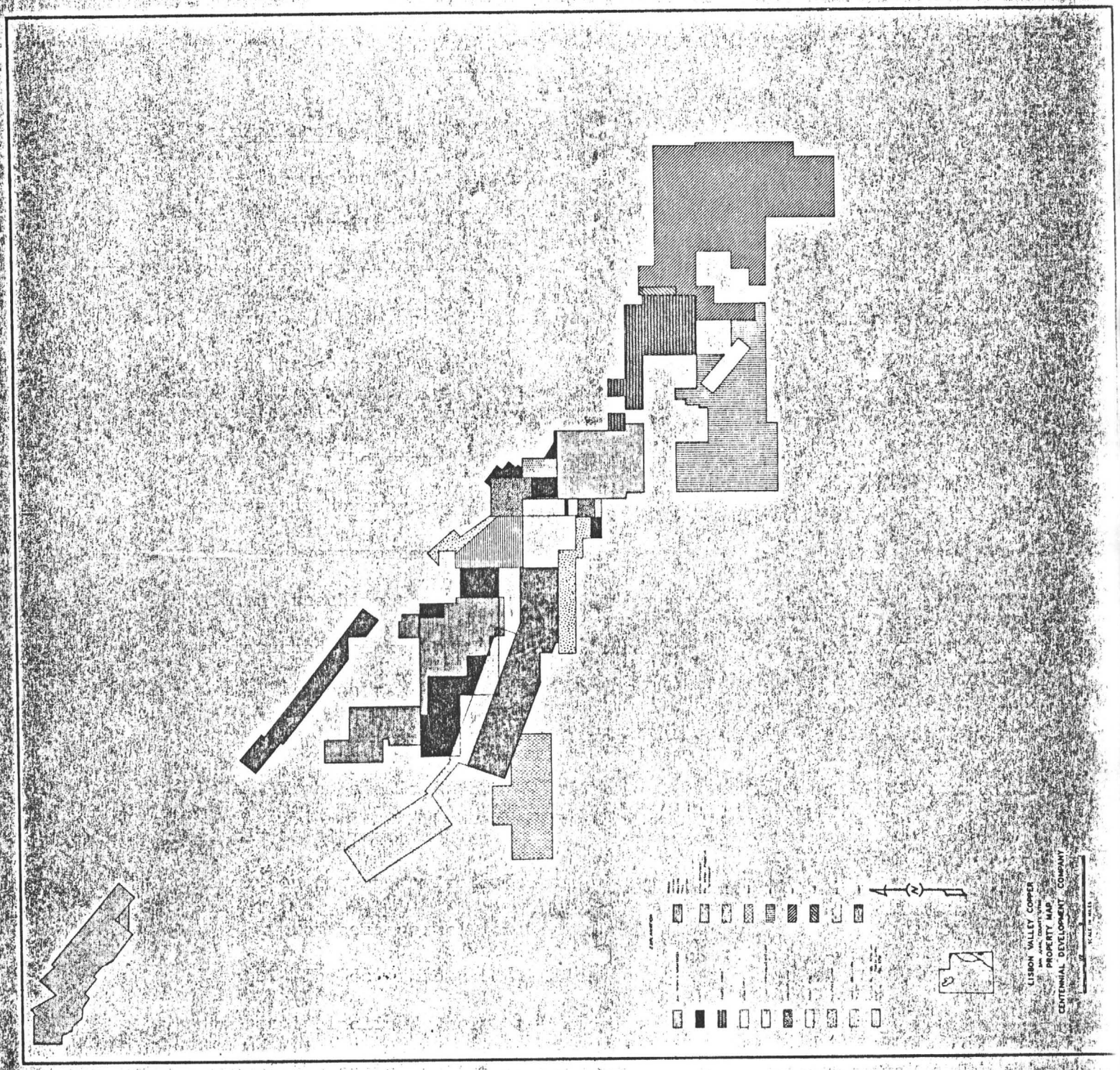

LISBON VALLEY COPPER
PROPERTY MAP
CENTENNIAL DEVELOPMENT COMPANY

Figure 3-A

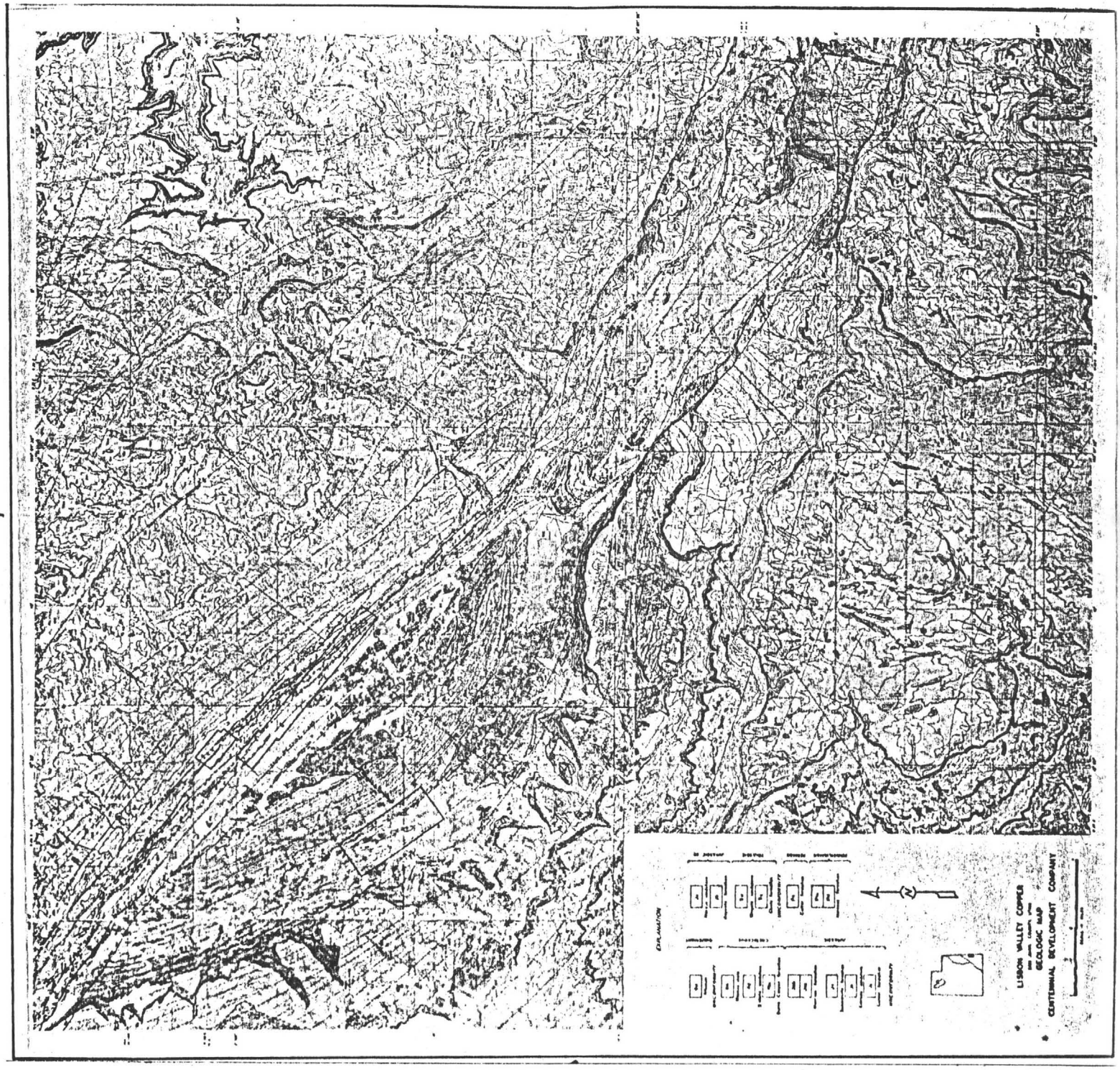

EXPLANATION
LISBON VALLEY COPPER
GEOLOGIC MAP
CENTENNIAL DEVELOPMENT COMPANY

Figure 3

through the structure. The faulting in this structure is regarded as resulting from a differential uplift, rather than a collapse, based upon the fact that the upthrown limb is more domed and steeper dipping than the opposite limb.

Along the surface trace of the Lisbon Valley fault, the downthrown block consists of the Morrison formation of Upper Jurrassic age and the overlying Burro Canyon, Dakota and Mancos shale formations of Cretaceous age.

The Morrison formation is divided into the Salt Wash member (lower portion), consisting of pink or gray sandstone interbedded with red or green mudstone, and the Brushy Basin member (upper portion), consisting of variegated shales with a few conglomerate sandstone layers.

The Burro Canyon formation overlies the Brushy Basin and consists of fine to medium grained sandstones of white and gray to variegated colors. It is overlain by the Dakota formation composed of beds of gray to buff sandstone interlayered with carbonaceous shale, coal and mudstones. A green shale, often sandy and from 5 to 25 foot in thickness, provides an excellent marker bed between the Dakota and Burro Canyon formations. These formations host the majority of commercial copper deposits in Lisbon Valley.

The Mancos shale in this area consists primarily of dark gray to greenish thin-bedded fissile shale.

Older formations outcrop within the upthrown or footwall block of the Lisbon Valley fault. These sediments encompass a geologic

time frame including Pennsylvanian, Permian, Triassic and Jurassic ages and harbor the major uranium host rocks described later. Copper mineralization is known to transcend this entire section as well. The greatest offset along the Lisbon Valley fault occurs where the Hermosa formation of Pennsylvanian age is in fault contrast with the Morrison formation. Here a stratigraphic throw in excess of 3800 feet is demonstrated.

Uranium in the Big Indian and lower Lisbon Valley districts has been mined from arkose lenses in the Cutler formation, the lower Moss Back member of the Chinle, the Moenkopi, and the Salt Wash member of the Morrison formation. Uranium deposits lie adjacent to the Lisbon Valley copper prospects in the footwall of the Lisbon Valley fault system.

Copper deposits occur in the hanging wall block of the Lisbon Valley fault within favorable sandstone beds of the Dakota-Burro Canyon formations.

The earliest known mining operation in the area was at the Big Indian copper mine, located in the N½, Section 34, T29S, R24E, Salt Lake Base and Meridian. This property was first operated sometime before 1913. Copper ore was mined by surface methods during World War II and further exploration of the deposit area has been conducted by Keystone-Wallace Resources in recent years. An oxide plant utilizing the conventional acid leach process (precipitation of cement copper from pregnant solution with de-tinned cans) has proven

successful in treating oxide copper ores of this locale and from tributary outlying deposits.

Additional productive copper deposits in the area of a similar geologic environment include the following:

DESCRIPTION	PROPERTY OWNER	LOCATION
GTO Pit	KWR-CDC	SW¼, Section 36, T30S, R25E
Globe Pit	KWR-CDC	NW¼, Section 26, T30S, R25E
Sentinal Pit	KWR-CDC	NE¼, Section 26, T30S, R25E
Micro Pit*	KWR-CDC	SW¼, Section 25, T30S, R25E and NW¼, Section 36, T30S, R25E
Blue Jay Pit	KWR-CDC	N½, Section 34, T29S, R24E
Three Step Mine	Wardle	W½, Section 8, T30S, R26E
Alice Mine	Homestake	W½, Section 28, T29S, R24E
Hi Lime	Nielsen	SW¼, Section 17,T30S, R25E

B. Centennial Pit Area Geology

The area of main economic interest to Centennial and the area of concentration of its investigation and resulting feasibility study lies at the southeastern end of the Lisbon Valley anticline. The axis of the Lisbon Valley anticline parallels the Lisbon Valley fault which trends 49° northwest. The Lisbon Valley fault horsetails in the Centennial Pit area making it structurally complex and thus favorably prepared for mineralization emplacement and/or movement.

*Located in the present Centennial Pit area.

The main structure of the Centennial Pit area is the Lisbon Valley fault itself, which separates the Cretaceous sediments from the Triassic and Jurassic on the upthrown or footwall side of the fault. The fault dips about 39° to the northeast and underlies most of the Centennial reserve. The Lisbon Valley fault and a number of parallel hinge faults have structurally separated the mineral deposit into four or five main blocks. These blocks generally strike in a northwesterly direction and dip to the southwest. The beds are parallel to the Lisbon Valley fault and its complimentary hinge branches. The blocks themselves generally dip more steeply on the southwest side of the deposit than on the northeast side where they project out into the valley.

The beds in the block on the southwest side of the deposit have a dip of 40° southwest. The beds in the blocks proceeding from southwest to northeast (see Plates 4.1-4.43) progressively decrease in dip to about 6° southwest. On the northwest side of the deposit, one block was projected to have a northeasterly dip instead of a southwesterly dip. It is now felt that this projection was in error. It caused some ore to be projected into the upper levels whereas we now expect it to down-dip southwesterly into the fault. These projections will not make any large differences in total tonnages of ore except, perhaps, in the early sequence of mining operations. These differences will be accounted for in final mine planning.

A number of cross faults project across the Centennial Pit area. These cross faults are felt to have a very significant role in the emplacement and reconcentration of copper mineralization. Throughout the Lisbon Valley, where Cretaceous mineralization is found, the relationship of this mineralization to cross faulting is very apparent. Copper mineralization in areas such as the Sentinal, Globe, GTO, Three Step and Hi Lime claims have been shown to be associated with these northeasterly trending cross faults.

The regional geology of the Centennial Pit area can be observed in Plates 1.1, 2.1 and in cross and longitudinal sections A-A, B-B and C-C indexed on Plate 2.1 and shown on Plate 2.2. More detailed geology of the deposit is shown on Plates 4.1 through 4.43. Plate overlays show the individual 5 ft. total copper assays. The observable magnitude of total copper assays in individual beds points to the favorable stratigraphic controls for mineralization. The reader can also observe the trend of mineralization toward and away from faults.

The stratigraphy of the Centennial deposit is comprised of the Dakota and Burro Canyon formations. The Dakota consists of sandstones, mudstones, shales and coal beds and is distinguished from the overlying Mancos shales by its carbon content. The Mancos is composed mainly of olive drab fissile shales whereas the Dakota contains more carboniferous shales, mudstones and coal beds which

are intermixed with both fine and coarse grained sandstones. It is the coarser grained sandstone and pebble conglomerates in the Dakota formation that host the higher copper grades. Close to faults, however, the entire Dakota formation can be mineralized to ore grade. The stratigraphy, thickness, sequences and numerical nomenclature of beds are shown in Table III-1.

The Dakota formation is separated from the underlying Burro Canyon by green bentonitic shales which, in turn, are underlain by fine to coarse sandstone which is well mineralized. Below this is a sequence of maroon and gray sandstones, limestones and shales which are mainly barren. At the bottom of the Burro Canyon formation is a sequence of fine to medium grained sandstones which contain a large tonnage of lower grade mineralization. This sequence, which is upthrown in the middle block of the Centennial Pit area (see cross sections 4.1-4.43), is highly silicified and contains varying amounts of mineralization depending on its proximity to faults. In and near the faults this bed contains very high grade mineralization.

Below the Burro Canyon is the Brushy Basin member of the Morrison formation which has been shown to be generally barren. In the deposit as a whole, the drill holes bottomed in either the Brushy Basin or in Triassic or Jurassic sediments which have been grouped into one footwall type (footwall of main Lisbon Valley fault) for the purpose of this feasibility study. These beds are generally not mineralized except at the fault contact. Relationships between

TABLE III-1

STRATIGRAPHY

Unit	Formation	Stratigraphic Description	Thickness	Unit
1	Overburden		0 ft. - 40 ft.	1
2	Mancos		0 ft. - 70 ft.	2
3	Dakota	Carbonaceous Shale to Mudstone	10 ft. - 20 ft.	3
4		Buff Medium Grained Sandstone to Mudstone	15 ft. - 25 ft.	4
5		Mudstone to Sandstone	15 ft. - 25 ft.	5
6		Coal to Mudstone and Carbonaceous Sandstone	5 ft. - 20 ft.	6
7		Mudstone to Sandstone	10 ft. - 20 ft.	7
8		Coal to Mudstone & Carbonaceous Sandstone	5 ft. - 20 ft.	8
9		Mudstone	10 ft. - 25 ft.	9
10		Fine to Medium Grained Sandstone to Mudstone	15 ft. - 20 ft.	10
11		Medium Grained Sandstone	20 ft. - 60 ft.	11
12	Burro Canyon	Green Shale - Sandy	5 ft. - 25 ft.	12
13		Medium Grained Sandstone	20 ft. - 40 ft.	13
14		Variagated Red Sandstone	50 ft. - 120 ft.	14
15		Silica Fine Grained Sandstone	40 ft. - 100 ft.	15
16		Medium Grained Sandstone	5 ft. - 30 ft.	16
17	Jmb-Jms-Js-Je-Jc-Jn-Jk-Tw-Tc-Pc	(Jurassic, Triassic and Permian - see Plate 1.1)		17

mineralization and stratigraphy can be noted on the cross sections and also on the 10 ft. level maps which show the geology that was input to the computer (Plates 5.1 through 5.22).

Geology will play an important part in mining and milling this deposit. The main Lisbon Valley fault plane stands as a potential slide hazard in the footwall (southwest side) of the pit. Pit planning has considered that wedges left on this fault might slide; benches were thus carried back into the fault on the southwest side. Strikes and dips of the beds change radically in crossing from one fault block to another making it important that mine plans always account for changes in ore loci.

Ten to 20 ft. beds of ore are often separated by 10 ft. to 20 ft. of barren waste. A higher grade may be mined if this waste could be selectively rejected. Ore rock types will range from coals and highly carbonaceous sandstones and mudstones to very hard quartzite which is found in the Burro Canyon. The difference in these rock types will affect mill throughput on a day-to-day basis. Under present mining plans, extended periods of these two extremes will be experienced. A certain amount of blending will be needed to provide a more uniform hardness and oxide content in the feed.

C. Mineralization

Both sulfide and oxide mineralization exist within the Centennial Pit area. Micro Copper and KWR have mined the main oxide portion of the deposit. The ratio of sulfides to oxides remaining is

approximately 80% to 20%. The amount of sulfide and oxide expected in the mill feed will vary considerably depending on which benches are being mined. Sulfide mineralization consists primarily of chalcocite with minor percentages of chalcopyrite, bornite and covellite. The main oxide minerals are malachite and azurite. A more detailed description of the mineral distribution, with polished and thin section photographs, can be found in the mineralogical report by Laszlo Dudas in Appendix C. Other copper minerals that may be found are cuprite, native copper and digenite.

Quite frequently it is difficult to distinguish chalcocite mineralization from carbon. Even after a great deal of field experience, this identification often led to misinterpretation of the degree of copper mineralization in drill hole samples. Because of the similar appearance of these constituents, it may be difficult to visually separate ore and waste zones in the pit.

The main ore mineral, chalcocite, has been found to occur in primary and secondary forms. Primary mineralization occurs mainly as balls or as small flecks within carbon layers. Secondary chalcocite has been identified in fractures around carbon lenses, within clays, and within coal lenses. Copper mineralization occurs erratically in the coal beds which are generally more barren away from the faults. The oxide minerals are believed to be products of oxidation of emplaced sulfide minerals. There has been some movement of mineralized solutions along fracture systems and within

porous sandstones. The degree of oxidation, movement and enrichment should be the subject of further study.

D. Alteration Features Within The Deposit

The main alteration noted is the silicification of one ore bed in the central, uplifted block of the deposit. The silicification is locally intense and the bed has metamorphosed into a fine to medium grained quartzite which can be seen in its original state on the deposit's extremities. An envelope of fine, white, hydrothermal clays (dickite) is found in lieu of chalcocite filling the favorable porous sandstone beds on the extremities. The coarse, interstitial form of copper mineralization plus the lack of intergrowth of copper minerals with other gangue minerals permits very good metallurgical recovery.

SECTION IV

ORE RESERVES

The process of evaluating and determining the mineral inventory and ore reserves resulting from the exploration program is described in detail below. This exploration program lasted approximately one year and involved additional land acquisition, setting up a survey network, drilling, sampling, assaying and metallurgical testing followed by office engineering analysis. The collected data was used in determining quantity and quality of mineralization and the economic parameters of its removal and treatment. After the mineral inventory had been calculated, it was subjected to a verification procedure as shown in Section B below and on geologic maps and cross sections appearing as Plates 6.5B, 6.11B and 7.1. The verification procedure established the accuracy of the computer model predictions.

A. Summary of Mineral Inventories and Ore Reserves

Mineral inventory is distinguished herein from an ore reserve in that a mineral inventory identifies expected quantities and qualities of mineral in place within a given area while an ore reserve is that material which can be extracted profitably from the mineral inventory. Calculations of Centennial Pit area mineral inventories progressed sequentially from hand cross section calculations to 25 ft. bench height computer calculated inventories, to 10 ft. bench height computer inventories, and then to a composite (averaged) 20 ft. bench height computer inventory. Minable reserves

were calculated on 25 ft. and 20 ft. bench height assumptions only as lesser bench heights were considered uneconomic from a mining standpoint. Summaries of mineral inventories and ore reserves are shown below in Table IV-1. The assumptions and methods for calculating these mineral inventories are described in following subsections.

TABLE IV-1

SUMMARY ANALYSIS OF MINERAL INVENTORY AND ORE RESERVE

CALCULATIONS THROUGH FEBRUARY 4, 1974

		Tonnage	Grade % Cu	Cutoff % Cu	Stripping Ratio	lb Cu
I.	Hand Calculated Mineral Inventory	6,412,000	0.912	0.40	N/A	116,955,000
II.	Computer Calculated Mineral Inventory (Assuming 25 ft. bench levels; all tonnage not necessarily minable.)	5,567,000	0.820	0.40	N/A	
III.	Computer Calculated - Pit Trial Run #1 (Assuming 25 ft. bench levels.)	4,151,000 5,514,000	0.746 0.645	0.40 0.30	5.36:1 3.78:1	
IV.	Computer Calculated - Pit Trial Run #2 (Assuming 25 ft. bench levels.)	4,790,000 6,605,000	0.717 0.615	0.40 0.30	6.89:1 4.72:1	
V.	Computer Calculated Mineral Inventory (Assuming 10 ft. bench levels.)	8,419,000 12,278,000	0.779 0.6448	0.40 0.30	N/A N/A	131,168,0 158,386,0
VI.	Computer Calculated Mineral Inventory (Assuming 20 ft. bench levels.)	8,463,000 12,321,000	0.748 0.623	0.40 0.30	N/A N/A	126,606,0 153,520,0
VII.	Computer Calculated Pit Trial #2 (Assuming 20 ft. bench levels.)	6,319,000 8,394,000	0.800 0.680	0.40 0.30	3:54 2:42	101,102,0 114,158,0

B. Verification of Ore Reserves

In a meeting in the offices of Pincock, Allen & Holt in Tucson on December 6, 1973, at which it was decided to reevaluate the Centennial Pit mineral inventory using a 10 ft. bench height composite to give a 20 ft. bench height analysis, Donald G. Bryant was asked to verify the results of the computer calculation of the mineral inventory by hand check and his report follows.

I. Hand Calculated Inventory - Grant & Brockenridge [illegible]
Explained IV-17 10 ft [illegible] mineral [illegible]
method, [illegible] mineral

II, III, IV [illegible] trend in bench elevation
rock type [illegible] grade [illegible] benches, does not
project along beds [illegible] rock type
Ore grade weighted by ft - [illegible] beds &
[illegible]

V Assume [illegible] samples into 10 composites interpolate
along strike & dip of bed to the same
rock type [illegible] with [illegible] block
Grade is weighted by ft

VI + VII [illegible] 10 [illegible] in V into 20 foot
benches

DONALD G. BRYANT

MINING AND MINERAL EXPLORATION CONSULTANT

7060 SO. GRAY COURT
LITTLETON, COLORADO 80123
(303) 798-3192

February 21, 1974

To: J. C. Bennett, Chairman
Centennial Development Corporation

From: Donald G. Bryant

Item: Validation of the Centennial pit copper reserves, Lisbon Valley, Utah as generated by the CDC computer program (Dec. 18, 1973)

CONCLUSIONS AND RECOMMENDATIONS

After intensive study, I have concluded that the reserves and ore distribution maps generated by the CDC computer program for the Lisbon Valley copper program are an adequate approximation of the results that could be calculated by hand. Because of the complex nature of the ore controls, I did not attempt to check the numerical grade distribution. The manual methods available - polygon or averaging in geologic units - could not or would not be more accurate than the computer with its ability to calculate the mathematical distribution between holes while making allowances for lithological changes. A general impression gained from a comparison of the distribution of +0.4 and +0.15% Cu ore indicates that the computer might be slightly conservative. During this study, I observed a few minor discrepancies on the 6300, 6350 and 6400 levels that can't be attributed to differences in interpretation of the structure and stratigraphy used by the computer. Although these discrepancies are not sufficient to affect the present feasibility study, they should be removed in subsequent computer studies.

PURPOSE AND BACKGROUND DATA

The purpose of this report and the accompanying maps is to determine whether the results of the CDC computer program (Dec. 18, 1973) for the Lisbon Valley copper project were a reasonable or adequate approximation of results that might be obtained from manual methods with conformed geologic sections and plans. The decision to check the results

of the computer program was made in early December, 1973, to be included as part of the feasibility program. Later in the month after the completion of the December 18 computer run, four levels - 6450, 6400, 6350 and 6300 - were selected for complete analysis and comparison. In January and February, 14 days were used to conform the 6300 and 6400 levels. In addition to the stratigraphy and ore intervals, the geologic structure as mapped by R. E. Grant and from my study of colored aerial photographs were used in the conformation of the plans. The interaction of structures, stratigraphy and ore intercepts complicated the study requiring more time than anticipated. The final resolution of the 6300 and 6400 levels required such extensive vertical correction of the sections that additional levels are not needed to establish the adequateness of the computer reserves. With the use of my sections, additional levels should be conformed to check for more minor discrepancies that can be corrected in subsequent printouts.

On the verticl sections used for this study, the drill hole notation included only the stratigraphy and ore intercepts as +0.4% Cu (solid) and +0.15% Cu (cross-hatched). Under these conditions some of the computer leach ore (yellow blocks) is not indicated on my work sections and plans. The general distribution, however, conforms nicely with that indicated by the manual estimate.

METHOD OF COMPARISON

After conformations were completed on the 6300 and 6400 at a scale of 1:600, the +0.4 and +0.15% Cu outlines were drawn on the sections and transferred to the plans. These sections and plans were then reduced to 1:1200, the scale of the computer printout. The structure and stratigraphy were compared on the level plans with reasonable conformance considering the differences between my interpretation and that initially entered into the computer. The ore distribution was also checked on the plans but the comparison on the sections was considered more important. The section comparison was more appropriate because the reason for the differences were more easily ascertained. Most of the differences, with inspection, could be attributed to slight differences of strike and dip of beds or to the different fault interpretations. Unaccountable differences were

DONALD G. BRYANT 7060 So. GRAY COURT LITTLETON, COLORADO 80123

minor and can be corrected in later printouts.

REGISTERED GEOLOGIST
CERTIFICATE NO.
8018
DONALD
CLASSELL BRYANT
Date Signed 6/21/76
ARIZONA, U.S.A.

DONALD G. BRYANT 7060 S. GRAY COURT LITTLETON, COLORADO 80123

C. Description of Exploration Program

The exploration program began in January, 1973, with a detailed evaluation of KWR's data supporting evidence of copper mineralization in the Lisbon Valley area. The KWR drilling through 1972 indicated the old GTO Pit had the best potential and therefore initial efforts were concentrated in this area. Unfortunately, it was quickly determined the GTO reserves (see analysis Tables IV-2 and IV-3) carried uneconomically high stripping ratios and further work in that area was abandoned.

Mineralization shown by KWR shallow drilling in the Micro-Costanza area, now referred to as the Centennial Pit area, was the next center of attention as it also showed considerable promise. A drilling grid was established and the exploration that ensued is described in more detail below.

On first analysis the drill hole locations on KWR maps of the Micro area became immediately suspect. Collar locations in the field had not been surveyed nor elevations established. Also, some sections were not assayed. Therefore, a basic topographic map (scale 1 in. = 100 ft., contour interval 5 ft.) of the general pit and adjoining areas was completed by Centennial personnel using plane table-alidade methods. This map was used as a base for early geologic mapping and later for drill hole location. Ultimately, all drill holes were surveyed by transit-chain procedures for accurate field locations and collar elevations.

Olympus Aerial Surveys was retained to make an initial grid survey, referenced to known government markers, and also to fly the

area to obtain aerial photographs for preparation of topographic maps at scales of 1 in. = 200 ft. and 1 in. = 50 ft.

TABLE IV-2

PINCOCK, ALLEN & HOLT

GTO PIT - 1ST TRIAL RESERVE STATEMENT

AS OF 1/1/73

(Tonnages in Thousands)

Level	+1.00% Tons	+1.00% % Cu	0.50-0.99% Tons	0.50-0.99% % Cu	0.40-0.49% Tons	0.40-0.49% % Cu	0.30-0.39% Tons	0.30-0.39% % Cu	0.20-0.29% Tons	0.20-0.29% % Cu	0.10-0.19% Tons	0.10-0.19% % Cu	-0.10% Tons	Tons
6500	--	--	--	--	--	--	--	--	--	--	--	--	97	97
6475	--	--	--	--	--	--	--	--	--	--	--	--	503	503
6450	--	--	--	--	--	--	--	--	--	--	--	--	1,040	1,040
6425	--	--	--	--	--	--	--	--	--	--	--	--	1,054	1,054
6400	--	--	--	--	--	--	--	--	--	--	--	--	999	999
6375	--	--	--	--	--	--	--	--	--	--	--	--	969	969
6350	52	1.73	1	0.50	--	--	--	--	--	--	--	--	899	952
6325	5	1.04	127	0.64	4	0.41	13	0.39	10	0.21	9	0.17	959	1,127
6300	67	1.52	89	0.62	11	0.46	47	0.35	--	--	67	0.14	780	1,061
6275	43	1.24	22	0.54	79	0.43	44	0.38	60	0.25	26	0.11	648	922
6250	--	--	50	0.91	19	0.40	61	0.37	11	0.26	41	0.14	491	673
6225	53	1.57	53	0.63	13	0.40	--	--	20	0.22	31	0.14	377	547
6200	20	1.96	60	0.63	--	--	--	--	9	0.22	30	0.15	189	308
6175	30	1.65	45	0.59	--	--	--	--	--	--	13	0.15	101	189
6150	37	1.76	--	--	5	0.44	--	--	--	--	--	--	74	116
6125	10	1.82	--	--	--	--	--	--	5	0.26	--	--	51	66
6100	--	--	5	0.84	--	--	--	--	--	--	5	0.14	9	19
Total	317	1.59	452	0.66	131	0.42	165	0.37	115	0.24	222	0.14	9,240	10,642

Preproduction stripping (above 6350) = 5,561 M tons.

TABLE IV-2

TABLE IV-3

PINCOCK, ALLEN & HOLT

GTO PIT - 1ST TRIAL RESERVE ESTIMATE

AS OF 1/1/73

WASTE RATIO AND LBS. COPPER RELATED TO CUTOFF GRADE

(Tons and Lbs. Cu in Thousands)

Cutoff % Cu	Ore Tons	% Cu	Waste Tons	Total Tons	Waste-Ore Ratio	Total Lbs. Cu
0.10	1,402	0.70	9,240	10,642	6.6	19,628
0.20	1,180	0.80	9,462	10,642	8.0	18,880
0.30	1,065	0.86	9,577	10,642	9.0	18,318
0.40	900	0.95	9,742	10,642	10.8	17,100
0.50	769	1.04	9,873	10,642	12.8	15,995

Drilling

After completion of access roads and initial drill hole site preparation, exploratory drilling commenced February 2, 1973, under contract to A-1 Drilling Company of Blanding, Utah. Rotary drilling operations by this firm utilized a Failing CFD1B (heavy duty Failing 1000) rig on a continuing basis. A second rig (Mayhew 1000) was provided for supplemental drilling during the month of June. Tricone bits of 4 3/4 in. diameter were used throughout the program. Preliminary drilling was conducted on 200 ft. to 400 ft. centers over the entire area of interest. Forty-four holes were completed aggregating 11,805 ft.

Favorable analysis of results from the preliminary drilling program warranted more closely spaced drilling for full evaluation of grade and tonnage. Accordingly, concentrated drilling operations on approximately 100 ft. centers were commenced April 24 and terminated August 23, 1973. During this four month period, an additional 179 rotary holes were completed for a total of 43,735 ft. In summary, a total of 223 rotary holes aggregating 55,540 ft. were drilled during the period February 2 through August 23, 1973.

Rotary drilling operations utilized compressed air to clear and retrieve cuttings from the hole bottom. Loss of circulation became a problem when damp or wet ground was penetrated and cessation of drilling on some holes was required. For this reason, it became necessary to deepen these holes by diamond

drilling to their projected depths. Several holes of relatively shallow depth were cored from surface. Each of these was located in the bottom of the old Micro Pit where the water table was at or near the surface. In addition, it was considered prudent to verify results of prior rotary drilling by coring alongside selected holes. Such offset holes were collared within five feet of a previously drilled rotary hole. Five holes were subjected to this test and results substantially verified earlier rotary drilling.

Diamond core drilling was contracted to B. F. Kissner Drilling Company of Cedaredge, Colorado. The varied requirements of the core drilling program precluded a direct-cost-per-completed-foot contract. Accordingly, drill rig time was contracted on an hourly basis with lump sum payments for mobilization and demobilization. Actual core drilling operations started May 23 and finished September 12, 1973. This program employed a modified Joy 22 truck-mounted rig (Sprague & Henwood drill works, etc.) and conventional (as opposed to wire line) BX drill bits. A total of 17 diamond drill holes were drilled from the surface and 31 holes were deepened. A total of 4736 ft. of coring was completed.

Prior to core drilling, a trial effort utilizing a Con-Cor rig was made to satisfy the needed check drilling. These services were contracted with Calvert Western Exploration Company of Tulsa, Oklahoma. Actual drilling operations commenced on May 12

but were terminated after only five days. In addition to unforeseen, difficult operating problems, it was determined that the coarse vs. fine split in the final sample, attained from a remix of material from coarse and fine sample splitters, was not accurately proportioned. This factor alone defeated the basic purpose of check sampling of rotary drilling operations. The rig was therefore dismissed in favor of core drilling.

Down-the-hole surveys were conducted by Scintilog Corporation of Casper, Wyoming. Deviation from vertical and azimuth readings were recorded in 50 ft. increments. A total of 135 holes were surveyed in this manner. The lack of available surveying units and caving of drill holes prevented all drill holes from being surveyed.

Sampling

Each type of drilling required its own sampling technique as discussed below.

Drill cuttings and retrieved dust from rotary drilling operations were sampled in 5 ft. increments in each hole. A work crew of three or four men was required at each drill site for proper execution of the sampling procedure. All cuttings and dust were repeatedly run through a Jones splitter until reduced to two samples of about two to three pounds each. After tagging and plastic bag identification, one sample was forwarded to Salt Lake for assay and the other cataloged and stored on the property for possible future reference. Concurrently a chip

card was made displaying cuttings representing each 5 ft. interval. These chip cards proved to be an invaluable aid toward identification of the 17 rock types and for resolution of faults.

Core recovery from diamond drilling operations was excellent, with nearly 100% attained throughout the program. The BX core was carefully split lengthwise with one-half retained for future reference. The other half was crushed in the field in a jaw crusher to approximately minus 3/8 in. and forwarded for assay in 5 ft. increments.

In view of the excellent core recovery, sludge samples were not recovered. Rotary holes deepened by diamond drilling were cased to the bottom to alleviate drill rod chatter. Upon penetrating new ground, the water circulation was often lost with no sludge return. The experienced driller encountered no difficulty in maintaining excellent core recovery under these adverse conditions.

Is the mineralogy such that some Cu mineralization could have been washed away by water circulating through some of the fractures (and occurrence)

As previously described, Con-Cor samples were not considered to be fully representative and drilling was terminated after only a few days. Con-Cor check drilling was performed adjacent to only two previously completed rotary drill holes. Assay results affirm the ore horizons of the rotary drilling, but the generally higher assays reported by the Con-Cor drilling have not been considered in ore reserve determination.

Assaying

All samples were sent to Salt Lake City to Chemical & Mineralogical Services (CMS) who then crushed, split and pulverized them to about one-quarter their original weight for assay by atomic absorption methods. Check assaying on approximately 10% of the samples was performed by the Union Assay office in Salt Lake City. Determinations were made for total copper, non-sulfide copper and composites were made of selected intervals (not exceeding 20 ft.) to assay for sulfur and silver. The Union check assays, performed by chemical methods, generally proved to be higher than CMS and it was thus felt the CMS assaying was, if anything, conservative.

In checking assay procedures it was realized that sample preparation could make a good deal of difference in assay results. Differences in preparation of rotary samples, as compared to split core samples, are generally thought to produce less severe anomalies. Because high grade nodules of chalcocite ("plumbs in a pudding") occur along the carboniferous sandstone beds, one can visually ascertain in the field how offset drilling by only a few feet will provide erratic assays for the same stratigraphic interval. Assay differences for 5 ft. intervals would be quite high; however such differences over larger intervals, say up to 20 ft., would be much lower. The check of diamond drilling against rotary drilling was hindered by this phenomenon because diamond drill holes were of smaller

Could be a serious problem in check drilling bulk sampling of multiple zones might be necessary

cross section, thus giving less sample and less chance of finding the "plumbs." Mineralization was also lost in cuttings making diamond drilling results more conservative. On the other hand, diamond drilling does not produce the salting that occurs when water is encountered in rotary drill holes and cuttings are trapped on the wall and later blown out irrespective of the depth of the drill hole.

D. Method of Reserve Compilation

1. Hand Calculated Mineral Inventory

The hand calculated mineral inventory utilized a vertical cross section estimating technique. Mineral above 0.4% total copper was classified as "ore" and between 0.15% - 0.399% total copper as "leach" material. Leach, waste and ore intervals were required to have a minimum thickness of 10 ft. to be placed in their respective categories. For intervals meeting the above classification requirements, polygons were drawn on each cross section to conform to the stratigraphy. The polygons were assumed to extend halfway (30 ft.) to neighboring cross sections.

Drill hole data was plotted on cross sections as shown in Portfolio Plates 4.1A - 4.43A. Assay intervals were composited such that assays above cut-off grades could be combined into ore and leach intervals. The constructing of polygons along section were limited by the lesser of a distance halfway to neighboring drill holes, fault boundaries or 75 ft. Geological

interpretations were not utilized in the hand calculated analysis to the extent they were in the computerized exercise.

Areas of polygons were planimetered on section and assigned a thickness of 60 ft. as explained above. The resulting volume was divided by a tonnage factor of 12.8 cu. ft. per ton to obtain a tonnage for each polygon. The derived tonnages and composite grades of each polygon were then accumulated to obtain a total tonnage and weighted average grade for the two classifications of mineralization.

was tonnage factor actually measured or estimated ???

Drilling was lacking on some cross sections. If geology and mineralization correlated well between cross sections, ore was projected to the deficient section using weighted averages of areas and grades from neighboring sections.

This method revealed a lack of continuity in ore from one section to the next. The problem of correlating geology and the trend of polygons from section to section introduced considerable complexity and uncertainty into the analysis.

Results of this analysis are shown in Table IV-1.

2. Computer Calculated Mineral Inventories

It was early determined that a geologic model was needed to more accurately depict the complex geology and mineral occurrence. A search was thus made to obtain an established computer program or service to meet this need. Ultimately a mineral evaluation and design system called "MEDS" was selected and leased from Mintec Inc. located in Tucson, Arizona.

The "MEDS" system was installed with considerable difficulty on the Univac 1108 Computer at the University of Utah Computer Center. A Texas Instrument "Silent 700" portable terminal was leased to communicate via telephone with the Univac 1108 for the purpose of altering and compiling programs, submitting data and starting run control programs. Considerable batch processing (feeding control and data cards into card readers) took place at the University Computer Center.

Program systems with basic data are stored on magnetic tapes at the University center and are loaded on "Fastran Drum" storage files for processing and alteration. Drum storage allows faster access to files and programs thus reducing time for processing data. Major repairs and revisions of computer supervisory systems at the University caused considerable delay, frustration and added expense to the overall program.

Computer models for mineral inventories were based upon three premises: (1) the quantity and quality of ore minerals varies as a function of distance from faults; (2) the mineral deposit consists of alternating beds of ore and waste; and (3) strikes, dips and location of beds change from one fault block to another.

The approach to (Figure 4) calculating the computer mineral inventories and/or reserves and some basic assumptions derived therefrom are as follows:

a) The geographic limits and dimensions of the three dimensional blocks to be used in defining the mineral inventory were

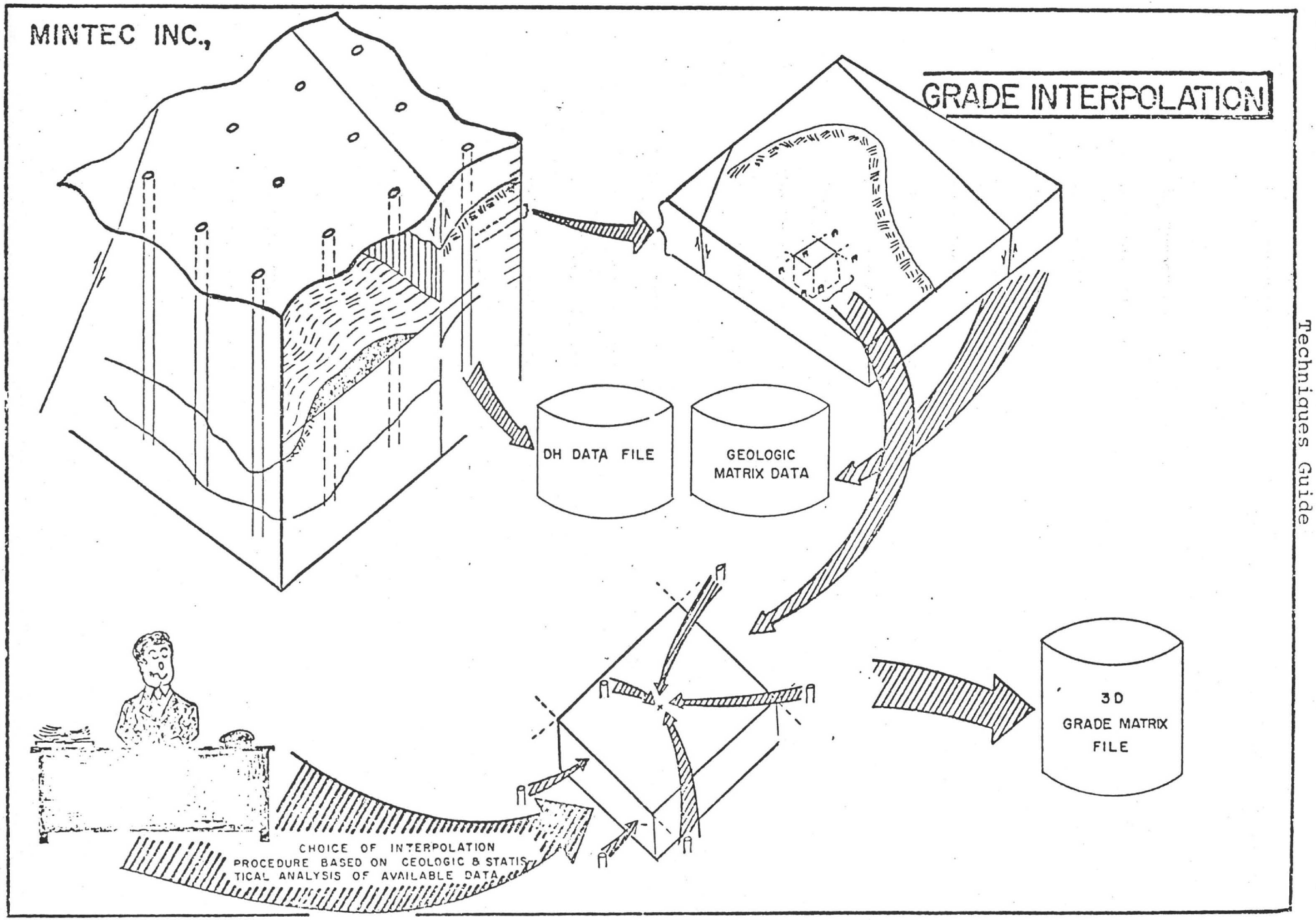

MINTEC INC.,
GRADE INTERPOLATION
DH DATA FILE
GEOLOGIC MATRIX DATA
3D GRADE MATRIX FILE
CHOICE OF INTERPOLATION PROCEDURE BASED ON GEOLOGIC & STATISTICAL ANALYSIS OF AVAILABLE DATA

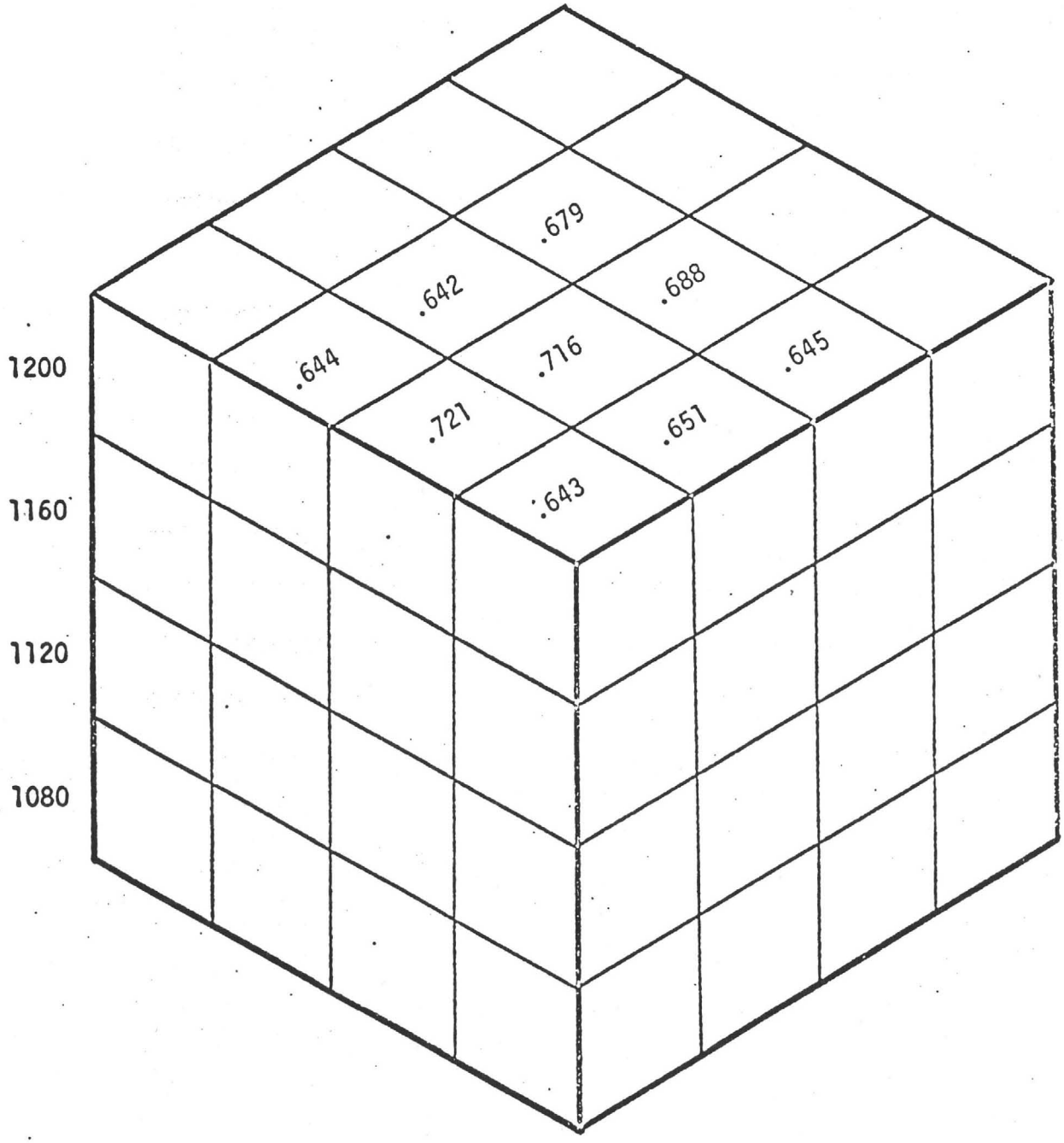
Figure 4

selected. The geographic limits incorporate all drilling in the Centennial Pit area (Plates 5.1 - 5.23). Vertical limits are from 6000 ft. to 6700 ft. above sea level. The size of individual blocks within this area (Figure 5) are 25 ft. square in horizontal cross section by the depth of selected bench heights. The blocks are oriented in a north-south direction to more accurately depict the northeast trending deposit.

b) Exploration program data was encoded on computer cards. This data includes: (1) assays coded and placed in a drill hole data file; (2) topography digitized in 50 ft. increments from which 25 ft. increments are interpolated; (3) surface and down hole surveys for each drill hole; (4) geology of each drill hole, including a code as to rock type and fault block identification; and (5) depths of fault intercepts in drill holes. These data permitted accurate calculations as to location and magnitude of mineralization in the deposit. Data encoding was then checked to eliminate any errors.

c) After data encoding, a bench height and elevation reference analysis was made. Assay composites of 5 ft. intervals were made from bench reference elevations and were averaged in 5, 10, 15, 20 and 25 ft. increments. The composite assays for all drill holes were examined to determine whether a specific bench reference elevation or a specific bench height would improve the grade and/or tons of ore to be mined. A change

Figure 9



3-DIMENSIONAL BLOCK REPRESENTATION OF AN ORE DEPOSIT

Figure 5

in elevation of the bench reference was found to have an affect on grade and the 6395 elevation was selected as optimum.

The bench height analysis also showed considerable differences in copper grade as shown below:

Bench Height	Composite Footage	Composite Grade %	Footage x Grade (analogous to tons x grade)
10	9010	0.94	8469
15	9405	0.89	8370
20	9460	0.85	8041
25	9425	0.83	7822
30	8760	0.83	7271

It is seen that lower bench heights permit waste to be left out of the composite intervals thus increasing both grade and pounds of metal contained. In most ore bodies when grade is increased, total metal content will drop. In this instance, metal content does not drop with increased grade because the decreased bench height not only eliminates waste but also increases the number of low grade ore intervals. A 20 ft. bench height was ultimately selected as a compromise between lower costs from mining higher benches and achieving more metal recovery from the use of lower benches.

d) Composites of assays conforming to selected bench height intervals were calculated starting from the optimum bench reference elevation. Total copper assays and oxide values

were averaged for these intervals. Oxide values were assayed only for assay intervals averaging more than 0.3% total copper. This meant that mineralization grading below 0.3% copper required an oxide value estimate. For marginal material the formula used in estimating the amount of oxide is as follows:

Let:

T = Total Copper Assay for Interval

O = Oxide Copper Assay for Interval

Then:

$$O = \left[0.6 - \left[\frac{\text{Depth of Assay}}{150} \right] \times .6 \right] \times T$$

By this formula the oxide grade would be 60% of the total copper grade at surface and would decrease down the drill hole to 0% of the total copper grade at 150 ft. The oxide was considered to be nil below 150 ft. After the composite grades had been determined for each bench interval the majority rock type and fault block type representing the composite were assigned to the same computer composite assay file.

e) The next step in the computer process was to calculate strikes and dips of faults and beds within each fault block. The fault blocks were defined after strikes and dips of faults were calculated. A "least squares" technique was used to define the fault or bedding plane that best represented

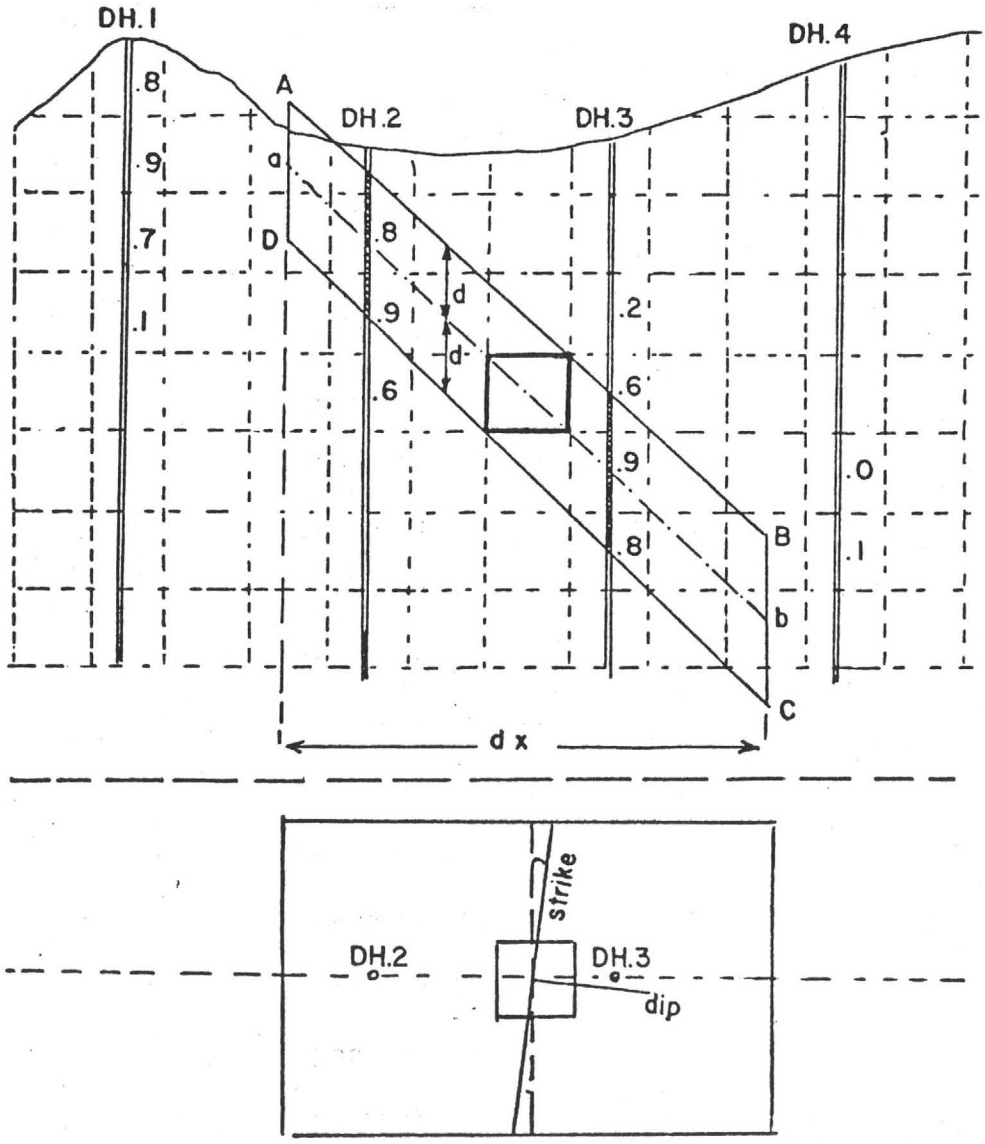
the data. In some cases fault blocks had to be revised because residuals from the "least squares" analysis were too high. A residual is the normal distance from the predicted plane to the data reference in the drill hole. Most residuals were less than 5 ft. to 10 ft. A few fault blocks contained only one or two drill holes which was insufficient for calculating strikes and dips of beds in these blocks. In such cases an arbitrary strike and dip was assigned. Where beds deviated a large amount from prediction, new fault blocks were arbitrarily defined and strikes and dips adjusted accordingly.

f) Individual model blocks were identified by code in relation to fault blocks. This was done by identifying each fault block by its bounding faults. All model blocks within the boundaries of a fault block were assigned the code of that fault block.

g) Once a model block was assigned a fault block code, a rock type code was assigned based on the closest drill hole interval along the projected strike and dip of the predominant bedding plane.

h) The fault block area was searched for other drill hole intercepts meeting limiting distance criteria and which had an affect on the interpolated value of a particular model block (Figures 6-A and 6-B). The limiting distance criteria was 150 ft. horizontally from the model block. This permitted the predicting of values for continuous areas of mineralization

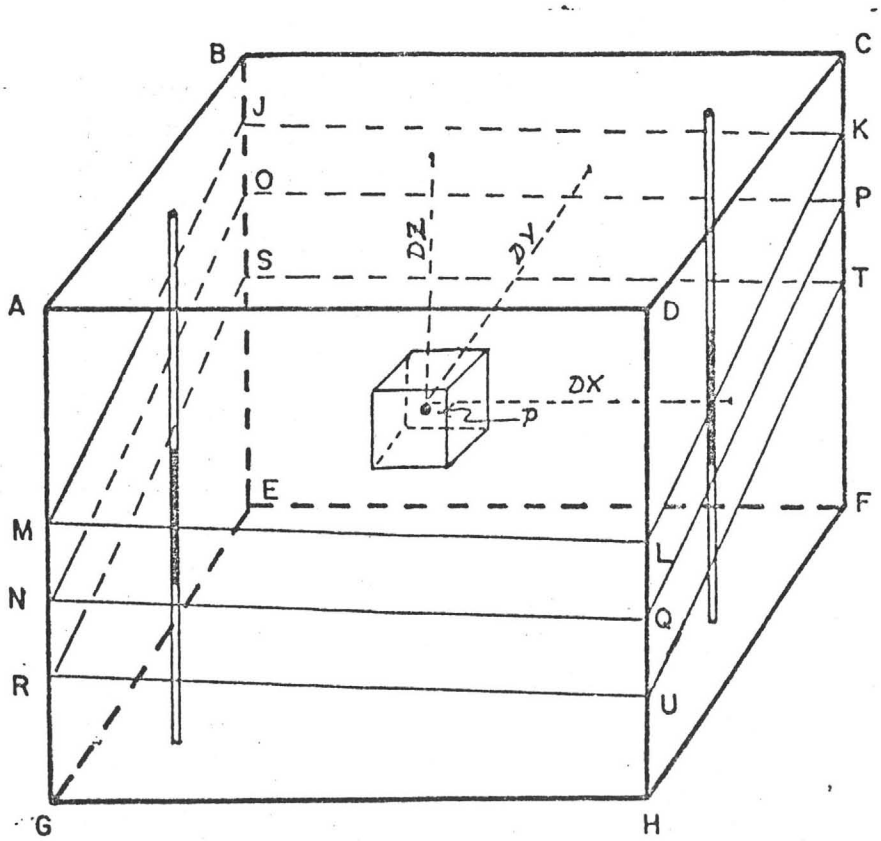
ILLUSTRATION OF 3-D SEARCH AROUND A TREND PLANE



NOTE THAT .2 COMPOSITE ASSAY IN HOLE 3. IS NOT WITHIN THE SEARCH AREA WHILE THE .9 COMPOSITE IN HOLE 3. WHICH IS THE SAME DISTANCE FROM THE BLOCK CENTER IS WITHIN THE SEARCH AREA.

Figure 6-A

ILLUSTRATION OF 3-D SEARCH AROUND A TREND PLANE



1. THE MAXIMUM AREA BOUNDED BY THE SIX OUTSIDE PLANES (A,B,C,D),(D,C,F,H),(E,F,H,G),(A,D,H,G), (A,B,E,G),(B,C,F,E).

2. FOR ANY BLOCK IN THE INTERPOLATION THE PLANE (N,O,P,Q) PASSING THROUGH THE MIDPOINT OF THE BLOCK WITH A GIVEN STRIKE AND DIP SPECIFIES THE SEARCH DIRECTION.

3. THE TWO PLANES (J,K,L,M), AND (R,S,T,U) THAT ARE PARALLEL TO AND AN EQUAL DISTANCE FROM (N,O,P,Q) FORM THE SEARCH VOLUME FOR THE BLOCK IN THE INTERPOLATION.

4. ALL DRILLHOLE VALUES THAT ARE WITHIN THE "SEARCH VOLUME" AND WITHIN ANY GEOLOGIC RESTRICTIONS WILL THEN ENTER INTO THE INTERPOLATION CALCULATIONS.

Figure 6-B

without unduly averaging a large number of assays. Vertically, a composite assay interval was allowed to be included if it was within two bench height intervals of the plane passing through the block. Only the closest composite from each drill hole was included in the interpolation formula.

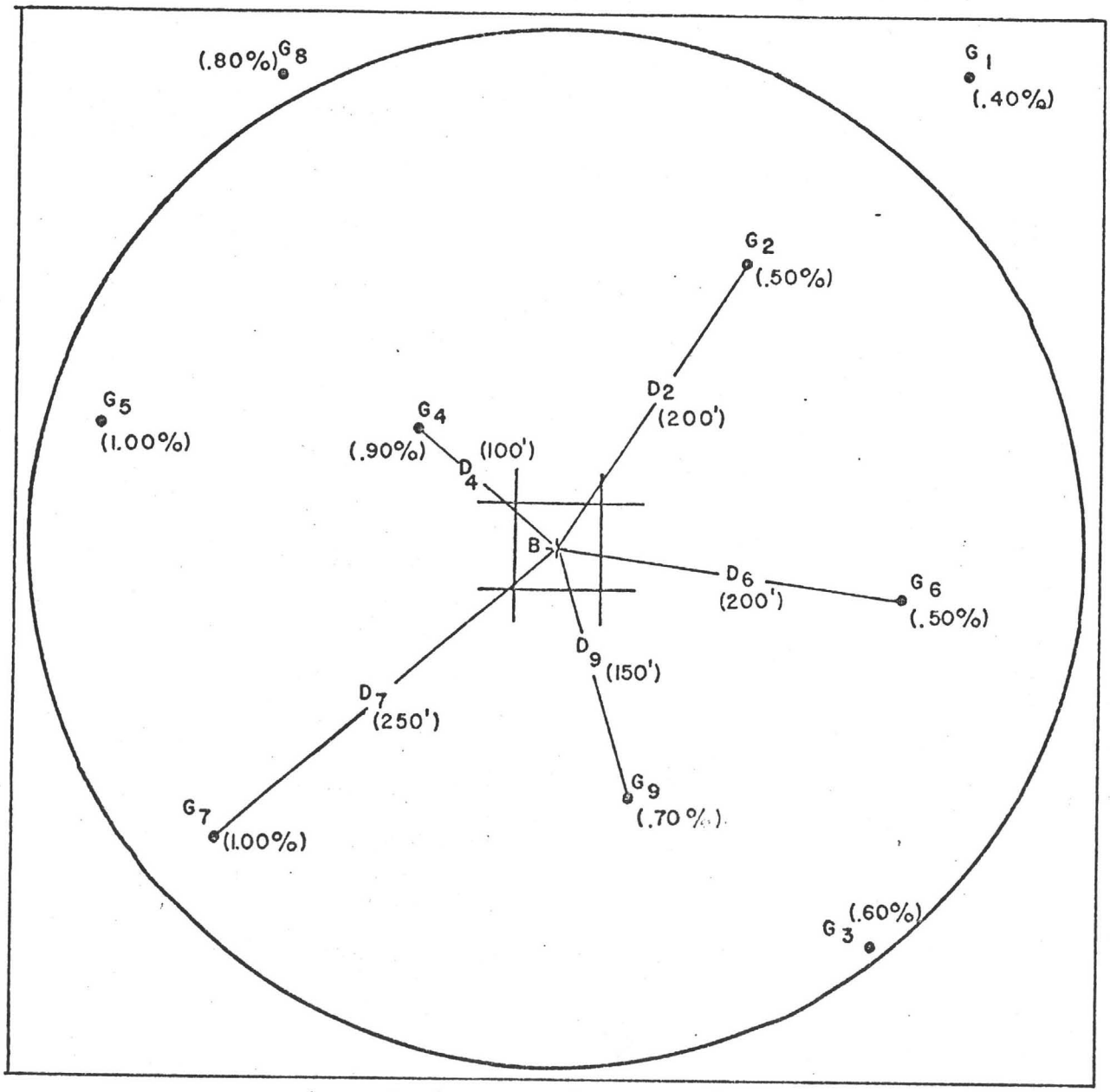
During the interpolation of the 25 ft. bench height inventory, the computer model masked the true identity of ore beds. Some drill holes contained minor thicknesses of high grade ore which could be minable but which were assigned a waste rock code. This masking thus served to reduce considerably the ore picture in the 25 ft. bench height analysis.

Matching drill hole intercepts were used in the interpolation calculations only if they were in the same fault block and met the limiting distance criteria.

It was found that true stratigraphy was best preserved in the model when 10 ft. composite intervals were the basis for rock type assignment.

i) An interpolation procedure using "$1/d^2$" weighting was used to determine total and oxide copper values in a given block. The interpolation formula, shown in Figure 7, is widely used at other mining properties and is believed to provide an accurate depiction of mineralization distribution.

The values for sulfur and silver were not carried in the mineral inventory. The results of metallurgical testing indicated their economic impact was not significant enough to warrant inclusion in the model.



$$B = \frac{G_2 \times \frac{1}{(D_2)^2} + G_6 \times \frac{1}{(D_6)^2} + G_9 \times \frac{1}{(D_9)^2} + G_7 \times \frac{1}{(D_7)^2} + G_4 \times \frac{1}{(D_4)^2}}{\frac{1}{(D_2)^2} + \frac{1}{(D_6)^2} + \frac{1}{(D_9)^2} + \frac{1}{(D_7)^2} + \frac{1}{(D_4)^2}}$$

$$B = \frac{.5 \times \frac{1}{(200)^2} + .5 \times \frac{1}{(200)^2} + .7 \times \frac{1}{(150)^2} + 1.0 \times \frac{1}{(250)^2} + .9 \times \frac{1}{(100)^2}}{\frac{1}{(200)^2} + \frac{1}{(200)^2} + \frac{1}{(150)^2} + \frac{1}{(250)^2} + \frac{1}{(100)^2}} = .77\%$$

ILLUSTRATION OF BASIC GRADE INTERPOLATION PROCEDURE

Figure 7

j) After the metal and rock type values had been assigned to each of the model blocks, maps showing total copper grade, rock type and fault block code were prepared as shown by Plates 5.1 - 5.22 and 6.1 - 6.20. The data on these maps were checked against drill log information and corrections were made where necessary to present a true model.

k) Following the prediction of values in the mineral inventory model, classifications of mineralization by level, grade and rock type were made and summarized (except for rock type) in Appendix E.

l) The next step was to code pit boundaries derived from mine plans prepared by Pincock, Allen & Holt.

m) Summaries of minable ore tonnage and grade by levels and rock type were made for two trial pits and the selected feasibility pit (Centennial Pit), all coded. These summaries provided a check of hand summation of blocks by Pincock, Allen & Holt.

n) Steps (a) through (i) above were performed for 10 ft. bench intervals. Starting with (j) the procedure was modified to permit a 20 ft. bench height analysis. Block values from the 10 ft. bench height model were averaged with those of the superimposed block immediately above to produce 20 ft. high model blocks. These were then used to develop a new set of mineral inventory and ore reserve summaries and, finally, mine production rates and costs used herein.

SECTION V

MINING

Pincock, Allen & Holt were responsible for mine design, production scheduling, equipment selection and development of operating costs. The discussion of their contributions in these areas follows in this section. The drawings and pit outlines supporting this work are shown in Plates 6.1A - 6.20A, 8.1 - 8.7 and 9.1 - 9.7.

SUMMARY

During the early phase of the project it was determined that additional drilling was required to accurately define the ore body. This drilling was done and all drill results were coded for processing by computer.

The resulting ore body model and level maps gave a realistic representation of the mineralized zones used in this study with respect to both tonnage and grade. Using this data an economic feasibility pit was designed and contained the following mineral inventory:

+0.40% Cu	6,348,000	tons at 0.80% Cu
0.35 - 0.39% Cu	887,000	tons at 0.38% Cu
0.30 - 0.34% Cu	1,110,000	tons at 0.32% Cu
	8,345,000	
Waste	20,914,000	tons

This pit design was based on a bench height of 20 feet, haul road width of 60 feet and varying slopes in the pit walls. Ore body geometry or geologic controls for the most part dictated the final wall slopes to be used.

The choice of the 20 foot bench height used in the pit design was based primarily upon limiting the amount of waste dilution of the slightly dipping ore horizons.

Inspection of the color coded level maps and sections indicated

that grade cut off was not an important criteria in establishing the final pit outline. The marginal material (0.30% to 0.39%Cu) tended to fill in gaps within the ore body, more than to extend the limits. As a result a change in cutoff grade would not significantly affect the pit design limits. However, if used, it would have the effect of increasing the tonnage of ore reserves at a reduced grade and give a lower overall waste-ore ratio.

Preliminary mining plans were developed by years to establish the amount of preproduction stripping required to prepare the mine for production. This amounted to 2,500,000 tons. These mining plans also established the amount of waste removal required to keep ore available on a continuing basis.

In view of the relatively low daily tonnages required (17,290 tons per day in years 1-3) together with the need for mobility of the loading equipment, we propose that the mine be developed with the use of front end loaders rather than conventional mining shovels.

Metallurgical test results indicated that the mixed oxide-sulfide ore carrying varying amounts of low grade coal was readily amenable to copper mineral recovery by recognized froth flotation techniques.

The demonstrated copper recovery of ore at an average grade of 0.80% Cu was 86% and produced a concentrate of 41.4% Cu.

A complete summary of the metallurgical results is enclosed under the section on Metallurgy.

PIT DESIGN

General

The geological mineral inventory of the ore body was based upon a deposit model developed by Centennial Development Company in Salt Lake City using the University of Utah computer. This model was based upon a series of 25' x 25' x 10' blocks. The computer output was checked by Dr. D. G. Bryant a geological consultant and gave a realistic representation of the ore body.

We received a set of level maps both at a 10 foot bench height and at a 20 foot bench height. The 20 foot bench maps were made from the computer model by using a program that added the two adjacent 10 foot levels, block by block.

A statement of geologic reserves by levels on the 20 foot interval was received from Centennial on January 12, 1974. This gave the following mineral inventory at the 20 foot bench height.

Cut off Grade	Tons	Average Grade
0.30	12,321,000	0.673
0.35	10,135,000	0.688
0.40	8,463,000	0.748

The choice of the 20 foot bench height in the pit design was a compromise between ore selectivity and mining efficiencies. The slight difference in grade and tonnage between the two bench heights was not significant enough to warrant the added mining costs for the lower bench height. In fact, it may be possible to do a selective job of grade control at a 25 foot bench height as experience is gained in the ore body.

Design Criteria

Haul roads are 60 feet wide and at a maximum grade of 8%. These roads will have to be surfaced with suitable rock in order that they can be serviceable for truck haulage in all seasons. This road material may have to be supplied from an outside source since the rocks within the pit area may not be suitable for this purpose.

Bench outlines are shown as the centerline of the bench face. Benches are labelled with the base elevation. In other words, the 6415 bench would extend from the 6415 elevation to the 6435 elevation and the 6415 bench centerline would be shown as the mid point of the bench face at 6425 elevation.

Ore body geometry or geologic controls for the most part dictated the final wall slopes to be used. Final wall

slopes were controlled by the Lisbon fault on the southwest and ore body geometry or maximum slope of 45° on the northeast wall. At both ends of the pit where the outlines were concave and were principally crosscutting the formations, a 50° slope was used. Cutoff grade was set at 0.40% Cu.

Design Procedure

The bench maps and sections were color coded based upon grade values of the blocks. The Lisbon Valley fault was also plotted on all sections and bench maps. Inspection of the resulting maps indicated that grade cutoff was not an important criteria in establishing a final pit outline. The marginal grade material (0.30% to 0.39% Cu) tended to fill in gaps within the ore body, more than to extend the ore body limits. As a result a change in cutoff grade would not significantly affect the pit design limits. It would have the effect of increasing the tonnage of ore reserves at a reduced grade together with a resultant drop in the waste-ore ratio.

Preliminary pit outlines were drawn on the sections and then transferred to plan maps. Since the ore body consists principally of two ore horizons separated by a waste zone, these preliminary pit outlines were first limited to the upper

zone and then another set was carried through to include the lower zone.

The small pit based upon the upper zone contained 3,396,000 tons of ore (+0.40% Cu) with a waste-ore ratio of 3.00 to 1. The total pit encompassing both zones contained 6,394,000 tons of ore with a waste-ore ratio of 3.63 to 1. The difference attributable to the lower zone was 2,998,000 tons with a waste-ore ratio of 4.34 to 1 and is economic.

There is one comparatively large ore section shown on the level maps and outside the feasibility pit outlines. This ore zone extends from the 6115 level to the 6275 level. We checked this area for possible inclusion in the feasibility pit. Resulting calculations indicated that it would have a waste-ore ratio in excess of 11 to 1 with an average grade of 0.58. This is considered to be uneconomic at this time.

MINEABLE RESERVES

Once the pit outlines were finalized a tabulation was made manually of the ore blocks within the pit limits on a level by level basis. Cutoff grade was taken at 0.40% Cu, however, all material was tabulated down to 0.30% Cu.

Total mineable reserves within the feasibility pit are summarized below at the various grade ranges:

(tonnage in thousands)

+0.40%		0.35-0.39%		0.30-0.34%		Waste
Tons	% Cu	Tons	% Cu	Tons	Grade	Tons
6348	0.80	887	0.38	1110	0.32	20,914

Waste-Ore Ratios

Cutoff grade	Ore-tons	Grade	Waste tons	W/O Ratio
0.40%	6348	0.80	22,911	3.61
0.35%	7235	0.75	22,024	3.04
0.30%	8345	0.69	20,914	2.51

The principal difference between total geological reserves and mineable reserves is caused by the large section to the north of the pit and below the 6295 elevation which has an uneconomic stripping ratio at this time. There are several other small areas excluded for the same reason.

The grade of the economic mineable reserves, however, is 0.80% Cu as compared with 0.748% Cu for the total geologic reserves.

CENTENNIAL PIT

Mining Reserve Statement

Level	Ore +0.40% Tons	Ore +0.40% %Cu	Marginal Material 0.35-0.39% Tons	Marginal Material 0.35-0.39% %Cu	Marginal Material 0.30-0.34% Tons	Marginal Material 0.30-0.34% %Cu	Waste Tons	Total Tons	W/O Ratio
6535							9,400	9,400	
6515							12,200	12,200	
6495	3,300	.60	900	.38	1,600	.33	55,500	61,300	17.58
6475	92,600	.63	10,300	.38	12,000	.32	724,200	839,100	8.06
6455	74,400	.63	18,900	.38	30,700	.32	1,376,500	1,500,500	19.17
6435	120,000	.69	18,200	.38	39,300	.32	1,839,200	2,016,700	15.81
6415	173,700	.65	38,300	.38	54,500	.32	2,044,700	2,311,200	12.31
6395	268,900	.68	53,900	.37	50,700	.32	2,326,600	2,700,100	9.04
6375	400,400	.72	63,100	.38	78,300	.32	2,078,400	2,620,200	5.54
6355	484,100	.74	78,000	.38	111,800	.32	1,856,800	2,530,700	4.23
6335	624,100	.85	82,000	.37	112,400	.32	1,606,900	2,425,400	2.89
6315	692,700	.91	81,500	.38	125,600	.32	1,356,100	2,255,900	2.26
6295	660,500	.95	65,000	.37	111,600	.32	1,223,100	2,060,200	2.12
6275	605,400	.91	77,500	.38	89,000	.32	1,039,700	1,811,600	1.99
6255	481,100	.83	61,300	.38	75,600	.32	890,800	1,508,800	2.14
6235	392,000	.74	45,600	.38	46,100	.32	730,500	1,214,200	2.10
6215	364,000	.75	47,700	.38	42,100	.33	570,600	1,024,400	1.81
6195	322,700	.74	32,400	.38	37,600	.32	449,200	841,900	1.61
6175	246,500	.65	39,100	.37	38,400	.32	338,500	662,500	1.69
6155	156,400	.70	38,500	.38	28,300	.32	247,400	470,600	2.01
6135	110,400	.77	24,600	.37	20,900	.32	113,500	269,400	1.44
6115	74,800	.76	10,300	.38	3,300	.34	24,100	112,500	.50
Total	6,348,000	.80	887,100	.38	1,109,800	.32	20,913,900	29,258,800	3.61

8,344,900 @ 0.69% Cu

Waste/ore ratio based on 0.40% Cu cutoff grade.

101,568
6,742
7,103
115,413

PINCOCK, ALLEN & HOLT, INC.

MINING PLAN

Preliminary mining plans were made to establish the amount of preproduction stripping required to place the mine in production. They were also made to establish the waste movement required to keep ore available on a continuous basis.

The basic criteria used in developing these plans were as follows:

1.	Cutoff grade	0.40% Copper
2.	Annual ore tonnage to concentrator	
	2500 tons/day x 360 days =	900,000 tons

Preproduction stripping - several approaches were tried. The best approach seemed to be to concentrate mining in the early stages along the Lisbon Valley fault in the west and northwest part of the pit. This allowed exploitation of the higher grade material as early as possible with no appreciable change in the amount of preproduction stripping required. Total preproduction amounted to 2,500,000 tons. A small amount of ore (170,000 tons) was removed to a stockpile during this period and would be sent through the plant in the start-up phase.

Mining plans were continued on a yearly basis. In developing these plans the annual ore requirements were blocked out and

the necessary waste was taken to leave the pit in a good operating situation and the haul roads were placed in their final position as soon as practicable.

We detailed four yearly increments and at this point the heavy stripping period was over. Also at this stage the pit was completed to its final limits down through the 6335 level. This made it possible to project a valid waste-ore ratio and yearly mining grade from this point to the end life of the mine.

These plans are feasibility study plans and are not to be considered as the final operating plans which will need to be detailed to a greater extent. However, they should serve as the guide in making the final operating plans with respect to properly sequencing the operation to achieve the yearly average grades shown. These mine plans have been sufficiently detailed and developed with realistic operating conditions in mind that they can be used in making a valid projection of operating costs and mill head grade over the life of the operation.

The accompanying tabulations outline the proposed mining plans by levels and years.

MINING SCHEDULE BY YEARS

Year	Ore +0.40% Tons	Ore +0.40% %Cu	Marginal Ore 0.35-0.39% Tons	Marginal Ore 0.35-0.39% %Cu	Marginal Ore 0.30-0.34% Tons	Marginal Ore 0.30-0.34% %Cu	Waste Tons	Total Tons	W/O Ratio
Pre-Prod.	170,300	.63	30,100	.38	44,300	.32	2,255,300	2,500,000	13.68
1	729,700	.78	76,600	.37	117,700	.32	3,398,500	4,322,500	4.92
2	900,000	.89	95,100	.37	144,400	.32	3,183,000	4,322,500	3.80
3	900,000	.93	99,800	.38	164,200	.32	3,504,800	4,668,800	4.19
4	900,000	.85	148,600	.37	164,700	.32	2,873,900	4,087,200	3.54
5	900,000	.72	175,600	.38	229,300	.32	2,548,600	3,853,500	3.28
6	900,000	.76	111,500	.38	112,400	.32	1,918,800	3,042,700	2.38
7	900,000	.71	143,200	.38	130,700	.32	1,215,500	2,389,400	1.66
Remainder	48,000	.76	6,600	.38	2,100	.34	15,500	72,200	0.50
Total	6,348,000	.80	887,100	.38	1,109,800	.32	20,913,900	29,258,800	3.61

PINCOCK, ALLEN & HOLT, INC.

CENTENNIAL PIT MINING PLAN

Pre-Production Period

Level	Ore +0.40% Tons	Ore +0.40% %Cu	Marginal Material 0.35-0.39% Tons	Marginal Material 0.35-0.39% %Cu	Marginal Material 0.30-0.34% Tons	Marginal Material 0.30-0.34% %Cu	Waste Tons	Total Tons	W/O Ratio
6535							9,400	9,400	--
6515							12,200	12,200	--
6495	3,300	.60	900	.38	1,600	.33	55,500	61,300	17.58
6475	92,600	.63	10,300	.38	12,000	.32	724,200	839,100	8.06
6455	74,400	.63	18,900	.38	30,700	.32	1,376,500	1,500,500*	19.17
6435							77,500	77,500	--
Total	170,300**	.63	30,100	.38	44,300	.32	2,255,300	2,500,000	13.68

* 6455 is the lowest level finished to reserve pit limit.

** +0.40% material to be stockpiled for milling in Ore Year 1.

Mine production requires 150 working days at 16,625 tons/day.

Waste/ore ratio based on 0.40% Cu cutoff grade.

PINCOCK, ALLEN & HOLT, INC.

CENTENNIAL PIT MINING PLAN

Ore Year 1

Level	Ore +0.40% Tons	Ore +0.40% %Cu	Marginal Material 0.35-0.39% Tons	Marginal Material 0.35-0.39% %Cu	Marginal Material 0.30-0.34% Tons	Marginal Material 0.30-0.34% %Cu	Waste Tons	Total Tons	W/O Ratio
6435	83,200	.75	7,800	.37	23,700	.32	987,000	1,101,700	12.24
6415	106,300	.71	11,400	.37	27,700	.32	1,044,900	1,190,300	10.20
6395	163,100	.74	22,800	.38	22,600	.32	695,100	903,600	4.54
6375	215,200	.81	20,700	.37	26,400	.32	525,800	788,100	2.66
6355	161,900	.85	13,900	.37	17,300	.32	145,700	338,800	1.09
Total	729,700	.78	76,600	.37	117,700	.32	3,398,500	4,322,500	4.92
Add ore from stock-pile	170,300	.63							
Total Mill Feed	900,000	.75							

Mine production requires 260 working days at 16,625 tons/day.

Waste/ore ratio based on 0.40$ Cu cutoff grade.

PINCOCK, ALLEN & HOLT, INC.

CENTENNIAL PIT MINING PLAN

Ore Year 2

Level	Ore +0.40% Tons	Ore +0.40% %Cu	Marginal Material 0.35-0.39% Tons	Marginal Material 0.35-0.39% %Cu	Marginal Material 0.30-0.34% Tons	Marginal Material 0.30-0.34% %Cu	Waste Tons	Total Tons	Ratio
6435	36,800	.55	10,400	.38	15,600	.32	774,700	837,500*	21.76
6415	7,100	.61	2,200	.38	2,500	.31	249,100	260,900	35.75
6395	22,800	.60	7,300	.37	6,700	.32	569,900	606,700	25.61
6375	82,400	.67	12,800	.37	14,100	.33	479,500	588,800	6.15
6355	171,500	.80	15,800	.38	32,300	.31	654,700	874,300	4.10
6335	446,200	.94	37,900	.37	60,900	.32	439,800	984,800	1.21
6315	133,200	1.11	8,700	.37	12,300	.33	15,300	169,500	.27
Total	900,000	0.89	95,100	.37	144,400	.32	3,183,000	4,322,500	3.80

*6435 is the lowest level finished to reserve pit limit.

Mine production requires 260 working days at 16,625 tons/day.

Waste/ore ratio based on 0.40% Cu cutoff grade.

PINCOCK, ALLEN & HOLT, INC.

CENTENNIAL PIT MINING PLAN

Ore Year 3

Level	Ore +0.40% Tons	Ore +0.40% %Cu	Marginal Material 0.35-0.39% Tons	Marginal Material 0.35-0.39% %Cu	Marginal Material 0.30-0.34% Tons	Marginal Material 0.30-0.34% %Cu	Waste Tons	Total Tons	W/O Ratio
6415	60,300	.55	24,700	.38	24,300	.32	750,700	860,000	13.26
6395	83,000	.59	23,800	.37	21,400	.33	1,061,600	1,189,800*	13.33
6375	2,200	.55	3,800	.38	12,800	.32	415,600	434,400	196.46
6355	19,700	.54	10,300	.37	15,000	.33	383,800	428,800	20.77
6335	6,600	.54	2,300	.36	5,600	.32	401,100	415,600	63.97
6315	338,600	.97	19,700	.38	44,900	.32	333,100	736,300	1.18
6295	389,600	1.06	15,200	.37	40,200	.32	158,900	603,900	0.55
Total	900,000	.93	99,800	.38	164,200	.32	3,504,800	4,668,800	4.19

* 6395 is the lowest level finished to reserve pit limit.

Mine production requires 281 working days at 16,625 tons/day.

Waste/ore ratio based on 0.40% Cu cutoff grade.

PINCOCK, ALLEN & HOLT, INC.

CENTENNIAL PIT MINING PLAN

Ore Year 4

Level	Ore +0.40% Tons	Ore +0.40% %Cu	Marginal Material 0.35-0.39% Tons	Marginal Material 0.35-0.39% %Cu	Marginal Material 0.30-0.34% Tons	Marginal Material 0.30-0.34% %Cu	Waste Tons	Total Tons	W/O Ratio
6375	100,600	.58	25,800	.38	25,000	.32	657,500	808,900	7.04
6355	131,000	.58	38,000	.38	47,200	.33	672,600	888,800	5.78
6335	171,300	.62	41,800	.37	45,900	.32	766,000	1,025,000*	4.98
6315	128,500	.74	12,400	.37	14,700	.33	324,700	480,300	2.74
6295	63,700	1.18	6,900	.37	11,200	.32	340,100	421,900	5.62
6275	304,900	1.15	23,700	.37	20,700	.32	113,000	462,300	0.52
Total	900,000	.85	148,600	.37	164,700	.33	2,873,900	4,087,200	3.54

*6335 is the lowest level finished to reserve pit limit.

Mine production requires 246 working days at 16,625 tons/day.

Waste/ore ratio based on 0.40% Cu cutoff grade.

PINCOCK, ALLEN & HOLT, INC.

CENTENNIAL PIT MINING PLAN

Ore Year 6

Level	Ore +0.40% Tons	Ore +0.40% %Cu	Marginal Material 0.35-0.39% Tons	Marginal Material 0.35-0.39% %Cu	Marginal Material 0.30-0.34% Tons	Marginal Material 0.30-0.34% %Cu	Waste Tons	Total Tons	W/O Ratio
6255	181,200	.83	23,100	.38	28,500	.32	676,000	908,800	4.02
6235	392,000	.74	45,600	.38	46,100	.32	730,500	1,214,200*	2.10
6215	326,800	.75	42,800	.38	37,800	.33	512,300	919,700	1.81
Total	900,000	.76	111,500	.38	112,400	.32	1,918,800	3,042,700	2.38

* 6235 is the lowest level finished to reserve pit limit.

Waste/ore ratio based on 0.40% Cu cutoff grade.

PINCOCK, ALLEN & HOLT, INC.

CENTENNIAL PIT MINING PLAN

Ore Year 5

Level	Ore +0.40% Tons	Ore +0.40% %Cu	Marginal Material 0.35-0.39% Tons	Marginal Material 0.35-0.39% %Cu	Marginal Material 0.30-0.34% Tons	Marginal Material 0.30-0.34% %Cu	Waste Tons	Total Tons	W/O Ratio
6315	92,400	.64	40,700	.38	53,700	.32	683,000	869,800	8.41
6295	207,200	.67	42,900	.37	60,200	.32	724,100	1,034,400	3.99
6275	300,500	.67*	53,800	.38	68,300	.32	926,700	1,349,300*	3.49
6255	299,900	.83	38,200	.38	47,100	.32	214,800	600,000	1.00
Total	900,000	.72	175,600	.38	229,300	.32	2,548,600	3,853,500	3.28

* 6275 is the lowest level finished to reserve pit limit.

Waste/ore ratio based on 0.40% Cu cutoff grade.

PINCOCK, ALLEN & HOLT, INC.

CENTENNIAL PIT MINING PLAN

Ore Year 7

Level	Ore +0.40% Tons	Ore +0.40% %Cu	Marginal Material 0.35-0.39% Tons	Marginal Material 0.35-0.39% %Cu	Marginal Material 0.30-0.34% Tons	Marginal Material 0.30-0.34% %Cu	Waste Tons	Total Tons	W/O Ratio
6215	37,200	.75	4,900	.38	4,300	.33	58,300	104,700	1.82
6195	322,700	.74	32,400	.38	37,600	.32	449,200	841,900	1.61
6175	246,500	.65	39,100	.37	38,400	.32	338,500	662,500	1.69
6155	156,400	.70	38,500	.38	28,300	.32	247,400	470,600	2.01
6135	110,400	.77	24,600	.37	20,900	.32	113,500	269,400	1.44
6115	26,800	.76	3,700	.38	1,200	.34	8,600	40,300	.51
Total	900,000	.71	143,200	.38	130,700	.32	1,215,500	2,389,400	1.66
Remainder 6115	48,000	.76	6,600	.38	2,100	.34	15,500	72,200	

Waste/ore ratio based on 0.40% Cu cutoff grade.

PINCOCK, ALLEN & HOLT, INC.

EQUIPMENT SELECTION

General

The nature of the deposit is such that drilling and blasting will have to be utilized at depth and can be favorable to attaining the lowest direct mining cost.

A small oxide operation has been conducted in this area in recent years utilizing scrapers. Mining has been in a relatively small area most of which was weathered oxide material. The present feasibility pit encompasses some oxide with a majority of the ore being sulfides. It also covers two main ore horizons separated by a waste zone.

Ore control procedures to minimize mining dilution will have to be adopted. It would be difficult to do this in a scraper operation. Therefore, since we will have to eventually drill and blast, it is felt that it is best to start with this system and utilize only one set of equipment throughout the life of the mine.

In view of the relatively low daily tonnages required together with the need for mobility of the loading equipment, we propose that the mine be developed with the use of front end loaders rather than using the conventional mining shovels.

The following data and calculations relate to the proposed equipment sizes and efficiencies:

1. Drilling

The 20 foot bench height selected for this project is near the minimum that can be utilized effectively by a rotary drill. Manpower requirements and total drilling cost are minimized also by using the rotary type drill.

A. Basic Data

1. Pattern size		18' x 18' x 24'
2. Hole size		9 7/8 inch
3. Yards per hole		240 BCY
4. Tons broken per hole		506.25 tons
5. Tons broken per foot drilled		21.09 tons
6. Waste ore ratio years 1-3	=	3.94
7. Average drill footage per drill shift		500 feet

B. Calculations

1. Total daily tonnage to be handled assuming a 5 day week in the mine and 7 day week in the plant.

$$2500 \times 4.94 \times 7/5 = 17{,}290 \text{ tons}$$

2. Drill shifts required per week

$$(17{,}290 \times 5) \div (21.09 \times 500) = 8.2 \text{ drill shifts}$$

use 9 drill shifts per week.

3. Utilization required to maintain production.

9 ÷ 15 = 60% on a 5 day basis allowing the off shifts and weekend for maintenance.

Use one diesel electric truck mounted rotary drill equipped with 30 foot rods for drilling holes in a single pass.

2. Loading

In order to minimize the capital cost of loading equipment and the need for mobility, we are proposing that 10 yard front end loaders be utilized as the primary loading units.

A. Basic Assumptions

1. Fill factor	85%
2. Cycle time per pass	60 seconds
3. Weight of material	2.11 tons/BCY
4. Swell factor	1 BCY = 1.33 broken
5. Weekly tonnage years 1-3	86,450

B. Calculations

1. Tons per loader shift

$$\frac{10 \times 85\% \times 60 \times 350 \times 2.11}{1.33 \times 60} = 4719 \quad \text{use } \underline{4700}$$

2. Loader shifts per day

17,290 ÷ 4700 = 3.68 use 4

3. Number of loaders required -

Four loader shifts per day would mean that two loaders would be working on one shift and a single loader be operating on the other shifts. The maximum availability to be expected is 50-60%. This would require three loaders to maintain four loader shifts per day on a continuous basis.

3. Hauling

1. The 50 ton truck unit was contemplated for this study.
2. Cycle times were calculated from the center of gravity of the ore body (6355 level) to the waste dump which is the longest haul and upon which the size of the truck fleet would have to be based.
3. Basic data

a. Loading cycle - average time per pass for loader to operate in a continuous production situation - 60 seconds.

b. Speeds

	mph	fpm
Level haul on bench < 750'	12	1056
Level haul on bench > 750'	15	1320
Ramp haul loaded +8%	8.5	748
Level haul surface	20	1760
Return down ramp	18	1584

c. Fixed time in haul cycle

Loading time

50 tons ÷ 13.5 tons/pass =	3.7 or 4 passes
4 x 60 seconds	4.0 minutes
turn and back in	0.5 minutes
turn and dump	0.9 minutes
Total	5.4 minutes

4. Cycle time calculations - Waste haul

Loaded		Return	
900' @ 1320 =	0.68 min.	2000' @ 1760 =	1.14
2500' @ 748 =	3.34	2500' @ 1584 =	1.58
2000' @ 1760 =	1.14	900' @ 1320 =	0.68
	5.16		3.40

Total cycle time = 5.40 + 5.16 + 3.40 = 13.96 min.

Tons per truck shift (350 ÷ 13.96) x 50 = 1250 tons

Ore haul cycle = 12.82 min.

Tons per truck shift = (350 ÷ 12.82) x 50 = 1350 tons

Truck requirements

Ore - 3500 tons ÷ 1350 = 2.6 use 3.0

Waste - 13790 tons ÷ 1250 = 11.0

Total truck shifts per day = 14.0

This would require 6 truck shifts on one shift while two loaders were operating at a reduced output and four trucks operating on each of the other shifts with one loader.

A minimum of 7 trucks would be required. This would give an average utilization factor of 67%. In the event the 6 trucks could not be maintained on the one shift it would force an extra loader shift, again at a reduced output per shift but would achieve the total production goals.

MINE COST SUMMARY

A. Mine Capital Cost

Major Equipment

1.	Drill - 1 Diesel Electric Rotary Drill 9 7/8 inch tools - 30 ft. rods	$ 190,000
2.	Loaders - Terex 7281 - 10 yd 3 @ $154,500	463,500
3.	Trucks - 7 Terex 3307 - 50 Ton	913,000
	1 Water Truck	83,000
4.	Dozers and Graders	
	2 - 8240 dozers with rippers	212,000
	2 - 12E cat graders	93,000
5.	Pickups and other auxiliary equipment and supplies	130,000
		$2,084,500
6.	Spare Parts Inventory @ 4.8%	100,000
	Total Mine Capital	$2,184,500

B. Direct Mine Operating Cost

		¢ per/ton
1.	Drilling	3.27
2.	Blasting	3.84
3.	Loading	6.48
4.	Hauling	11.34
5.	Roads and Dumps	4.57
6.	Pit Department (supervision and General)	4.38
	Sub total	33.88
7.	Payroll burdens	2.94
	Total	36.82
	Use	37.00

These costs are based upon the prevailing wage rates currently in effect for the 1973-74 south west area labor contracts.

Payroll burdens used were 31% for hourly employees and 22% for staff employees.

Direct Mine Operating Costs

1. Drilling

A. Rig cost (operating and repair) $40/oper.hr.

9 drill shifts per week

9 x $40 x 7 ÷ 86450 tons 2.91¢/ton

B. Bit cost $380/bit @ 5000' life

380 ÷ 5000 = 7.6¢ per foot

7.6 ÷ 21.09 = 0.36¢/ton

C. Total drilling cost per ton 3.27¢

2. Blasting - Based upon using Ammonium Nitrate

A. Basic data

1. Powder factor 0.7 lbs per yard
2. Pounds ANFO per hole (.7 x 240) 168 lbs.
3. ANFO price 7¢ lb.
4. Labor - 3 man crew

   1 lead powder man @ 4.37/hr.

   2 powder men @ 4.27/hr.

B. Cost calculations

1. Explosives 168 lbs x 0.07 = $ 11.76/hole
2. Supplies - cord, boosters, delays, etc. 2.90/hole

Subtotal $ 14.66

3. Explosives cost per ton

$14.66 ÷ 506.25 tons	=	2.90¢/ton

4. Labor & truck cost

(4.37 + 2 x 4.27) x 8	=	$103.28
truck @ $7.50/hr x 8	=	60.00
		$163.28

5. Labor & truck cost per ton	0.94¢/ton
6. Total Blasting cost	3.84¢/ton

3. Loading

10 yd loader - Oper. & Main. cost/hr	=	$ 40.00
4 x 40 x 7 ÷ 17,290	=	6.48¢/ton

4. Hauling

Operating and Maintenance cost		$140.00/shift
14 truck shifts per day		
14 x 140 ÷ 17,290	=	11.34¢/ton

5. Roads and Dumps

Equipment	Cost/Hr.	Cost/Shift	Shifts/Day	Total Cost
Dozers	18	126	3	$ 378.00
Graders	13	91	3	273.00
Water truck	10	70	2	140.00
			Total	$ 791.00

Cost per ton $791.00 ÷ 17,290 — 4.57¢/ton

6. Pit Department - Supervision & General

a. Supervision - Staff

Superintendent	1	$ 20,000
Shift Foremen	3	45,000
Chief Engineer	1	15,000
Junior Engineer	1 @ 1100/Mo.	13,200
Surveyors	2 @ 900/Mo.	21,600
Timekeeper	1 @ 800/Mo.	9,600
Clerk	1 @ 750/Mo.	9,000
Janitor	2 @ 700/Mo.	16,800
Mechanical Foreman	1 @ 1400/Mo.	16,800
	Total Staff	$167,000

b.	Supplies @ 300/month	3,600
c.	Light Vehicles - 5 @ 200/month	12,000
d.	Miscellaneous	12,000
	Annual Total	$194,600

Annual Tonnage = 4,446,000 tons

Cost per ton = 4.38¢/ton

7. Total Fuel Requirements (diesel) 16,350 gal./wk.

or 3,270 gal./day

PERSONNEL SCHEDULE

I.	**Staff**	
	Pit Superintendent	1
	Mechanical Foreman	1
	Pit Shift Foremen	3
	Chief Engineer	1
	Junior Engineer	1
	Surveyors	2
	Timekeeper	1
	Clerk	1
	Janitor	2
	Sub total	13
II.	**Pit Operating Crew**	
	Loader Operators	4
	Drill Operator	2
	Drill Helper	2
	Blasting Crew	3
	Truck Drivers	15
	Equipment Operators	8
	Sub total	34
III.	**Pit Maintenance Crew**	
	Mechanics, Welders, Helpers	15
	Total	62

Because of the ore and rock complexity a resident geologist might well be worth considering