File No. C-1

PROSPECT CARDS

Code No.

Property Name Queen of Bronze Mine

Property Owner

Submitted by

Location: State Oregon

County Josephine

Mining D. Near Takilma

T____ R____ Sec.____

Followup Recom. some potential

Later Review Recom.

Examined by

Company

Date

Where filed

Metals		Production Metal
	Cu	X
	Mo	
	Pb	
	Zn	
	Ag	X
	Au	X
	Fe	
	Mn	
	Cr	
	Ni	
	W	
	U	
	Re	
	P_2O_5	
	K_2O	
	Sn	
	Be	
	Coal	
	Hg	
Other		

AMS Quad

Other Quad

Production

Tons

NONE	10^2	10^3	10^4	10^5	5×10^5	10^6

Geology

Host Rock slates, limes, qtzites and basic igneous intrusions

Mineralization

Type

Trend

Ore chalcopyrite, Pyrr, Py

Gangue

Alteration

Type

Extent

Bibliography

USGS

USBM

Other

Remarks: Too small (or reserves calculations too difficult to figure)

Field Time

None

1 Day

1 Week

1 Mo

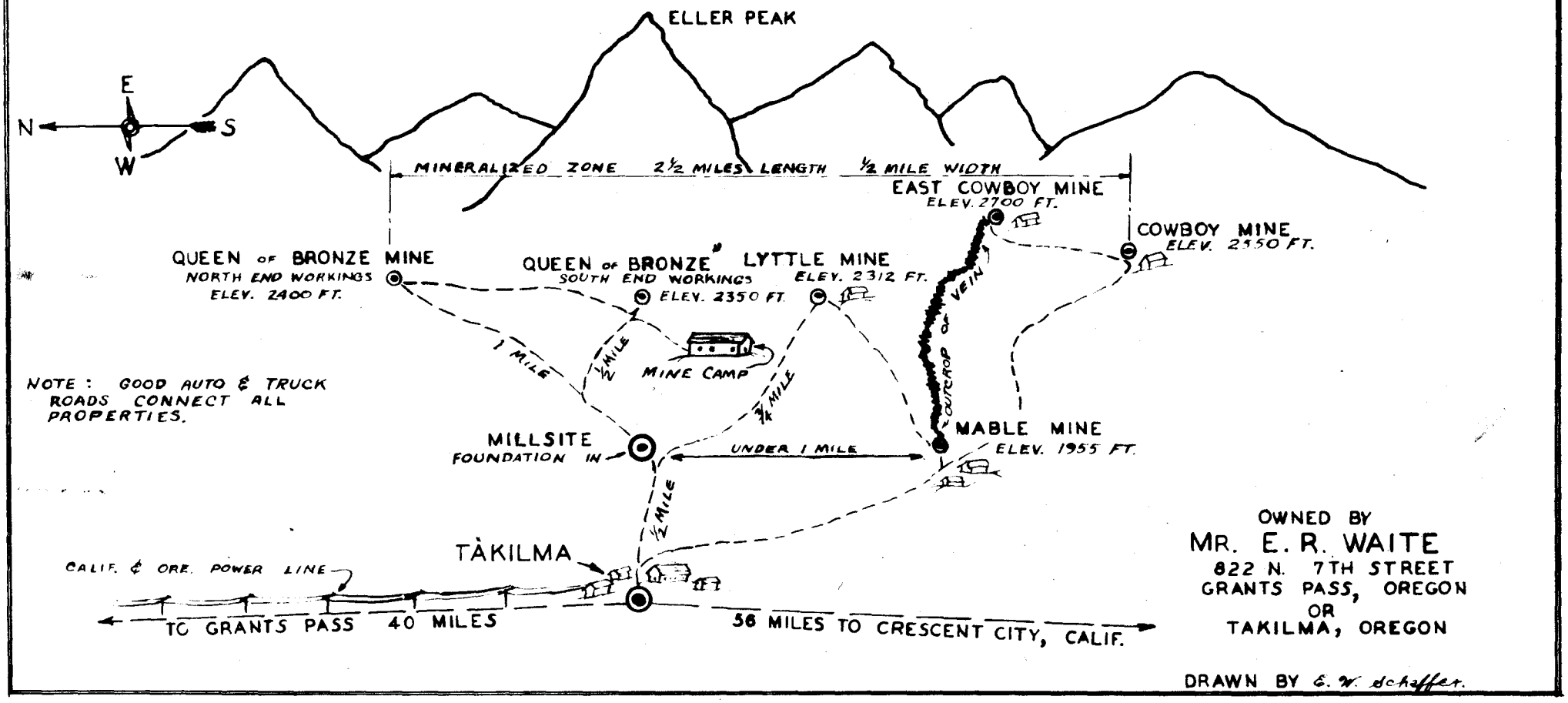
+1 Mo

Follow-up Recom.

BC 434

QUEEN OF BRONZE GROUP OF MINES — TAKILMA, OREGON

14 000 FEET OF UNDERGROUND DEVELOPMENT, PRODUCTION OF SHIPPING ORES. TOTAL IN EXCESS $1,700,000.



OWNED BY
MR. E. R. WAITE
822 N. 7TH STREET
GRANTS PASS, OREGON
OR
TAKILMA, OREGON

DRAWN BY E. W. Schaffer.

AMERICAN CYANAMID COMPANY
Ore Dressing Laboratory

COPY

T. I. 242
Problem 463
December 1, 1947

Heavy-Media Separation Tests on Gold-Copper Ore
From Queen of Bronze Mine Submitted by Mr. E. R. Waite

This report is based on metallurgical results obtained in the Ore Dressing Laboratory of American Cyanamid Company on a sample of material submitted by the Subject Company, and all recommendations and opinions expressed herein apply only to the treatment of material conforming to the sample submitted.

Introduction:

Mr. E. R. Waite of Grants Pass, Oregon shipped a 500 pound sample of dump ore from the Queen of Bronze Mine, Takilma, Oregon to our Mineral Dressing Laboratory for Heavy-Media Separation tests. Mr. Waite advised that a substantial tonnage was available but that it was too low grade to ship to a smelter. It was his thought that concentration by Heavy-Media would produce a shipping grade concentrate.

Object of Investigation:

We were asked to determine the amenability of the sample of dump ore to concentration by Heavy-Media Separation.

Description of Shipment:

Four kegs having a gross weight of 500 pounds were received at the Mineral Dressing Laboratory in good condition on October 7, 1947. The Sample consisted of pieces ranging in size from 6" to fines.

Preparation of Sample:

The contents of all four kegs were screened in 1". The screen oversize was crushed to pass 1" and the entire sample was then screened on 10 mesh. The minus 1" plus 10 mesh ore was mixed and representative 50 pound charges were cut out for testing. The minus 10 mesh fraction was weighed and samples for assay. The above treatment resulted in the following weight distribution.

COPY

AMERICAN CYANAMID COMPANY
Mineral Dressing Laboratory

TABLE I

Weight Distribution

Prepared Ore

Product	% Weight
Original	100.00
-1" + 10 mesh	89.95
-10 mesh	10.05

242 - 463

Heavy-Media Test Procedure:

Each 50 pound charge of -1" + 10-mesh ore was wet screened on 10 mesh to remove adhering slime. The wet minus 10 mesh fraction was dried, weighed and assayed. The 1-1 + 10-mesh ore was then fed continuously at the rate of 50 pounds per hour to a 20" diameter Heavy-Media Separatory cone. Preliminary examination of the ore had indicated the optimum specific gravity of separation to be slightly below 3.00. The Heavy-Media Separatory cone was therefore charged with a medium consisting of a mixture of finely ground ferrosilicon and magnetite suspended in water. The specific gravity of the medium was adjusted to 3.00 and the feed was started.

The ore particles having a specific gravity lower than 3.00 floated on the surface of the medium and were discharged from the cone over a weir. The heavier fraction of the ore sank and was discharged from the ore continuously by means of an air lift. Both products were drained free of medium on a vibrating screen. The drained medium was returned to the cone. The separated ore products were washed free of medium on the lower part of the screen. Each product was dried, weighed and assayed.

Separating mediums having specific gravities of 2.93 and 2.88 were employed in subsequent tests.

The metallurgical results of these tests are presented in Tables II, III and IV.

AMERICAN CYANAMID COMPANY
Ore Dressing Laboratory

CONF

Tech. Invest. No. 242-463

Table No. II - Test No. 1.

Sample: -1"

Product	% Weight In Size	% Weight In Total	Assays oz/ton Au	Assays % Cu
Ore		100.00	0.044	2.41
Total - 10-mesh		10.25	0.050	1.98
Cone Feed -1" / 10-mesh	100.00	89.75	0.043	2.46
Sink in 3.00	32.58	29.24	0.090	5.26
Float in 3.00	67.42	60.51	0.020	1.11

Remarks:

Product:	% Distribution In Size Au	In Size Cu	In Total Au	In Total Cu
Ore				
Ore			100.00	100.00
Total -10-mesh			11.76	8.41
Cone Feed -1" / 10-mesh	100.00	100.00	88.22	91.59
Sink in 3.00	69.51	69.50	60.44	63.75
Float in 3.00	31.49	30.40	27.78	27.84

AMERICAN CYANAMID COMPANY
Ore Dressing Laboratory

COPY

Table No. III - Test No. 2

Tech. Invest. No. 242-463

Sample: -1"

Product:	% Weight		Assays	
	In Size	In Total	oz./ton Au	% Cu
Ore		100.00	0.050	2.54
Total -10-mesh		10.25	0.050	1.98
Cone Feed -1" / 10-mesh	100.00	89.75	0.050	2.61
Sink in 2.93	53.41	47.94	0.076	4.11
Float on 2.93	46.59	41.81	0.020	0.88

Product:	% Distribution			
	In Size		In Total	
	Au	Cu	Au	Cu
Ore			100.00	100.00
Total -10-mesh			10.28	7.99
Cone Feed -1" / 10-mesh	100.00	100.00	89.72	92.01
Sink in 2.93	81.34	84.27	72.98	77.53
Float in 2.93	18.66	15.73	16.74	14.48

AMERICAN CYANAMID COMPANY
Ore Dressing Laboratory

COPY

Tech. Invest. No. 242-463

Sample: -1"

Table No. IV. - Test No. 3

Product:	% Weight In Size	% Weight In Total	Assays Oz/ton Au	Assays % Cu
Ore		100.00	0.045	2.30
Total -10-mesh		10.25	0.050	1.98
Cone Feed -1" / 10-mesh	100.00	89.75	0.044	2.33
Sink in 2.83	65.23	58.54	0.060	3.18
Float on 2.83	34.77	31.21	0.015	0.74

Product:	% Distribution In Size Au	% Distribution In Size Cu	% Distribution In Total Au	% Distribution In Total Cu
Ore			100.00	100.00
Total - 10-mesh			11.42	8.84
Cone Feed -1" / 10-mesh	100.00	100.00	88.58	91.16
Sink in 2.83	88.24	88.96	78.17	81.09
Float on 2.83	11.76	11.04	10.41	10.07

Discussion of Results:

The metallurgical results given in Tables II, III and IV Show that as the specific gravity of separation was decreased the ratio of concentration decreased and metal recoveries increased.

The highest grade concentrate was made in Test 1 in which a medium having a specific gravity of 3.00 was employed. The sink product amounted to 29.24% weight of the original ore and assayed 0.09 oz. Au per ton and 5.26 % Cu. Recoveries of 60.44 % of the gold and 63.75 % of the copper contained in original ore were made.

Conclusion:

Heavy-Media Separation tests conducted on a sample of dump ore from the Queen of Bronze Mine have shown the amenability of this material to concentration. The optimum specific gravity of separation will depend largely on the cost of marketing the concentrate. Heavy-Media Separation at 3.00 specific gravity produced the highest grade concentrate. If shipping costs are moderate the use of a lower specific gravity of seperation might be indicated so as to obtain higher metal recoveries even though at the expense of concentrate grade.

AMERICAN CYANAMID COMPANY

/s/ R. A. Pickens
R. A. Pickens
Mineral Dressing Laboratory

#1 about 15 ft down shaft
Green Copper glance ore
about 1 ft wide. East Cowboy

#2 Red oxide ore
East Cowboy

#3. Black Iron ~~[illegible]~~
about 1 ft wide
East Cowboy

GOLD ONLY, $1.00; COPPER, $1.00; LEAD, 50c
GOLD AND SILVER, $1.00; SILVER ONLY, 50c
SILVER BOUGHT. SPECIMENS BOUGHT, SOLD or EXCHANGED

Assay Office of Wm. H. Stowell &

421 SPRAGUE AVENUE
TELEPHONE MAIN 5924

Memorandum of Assay of Rock made for Pohlman Inv Co

Spokane, Wash., January 16th,

SAMPLE MARK	ASSAY VALUE PER TON OF 2,000 POUNDS AVOIRDUPOIS														
	per lb.			Copper 8¢ per lb.			Silver at 45 cents per ounce				Gold at $20.67 per ounce				
	Per Cent	DOLLARS	Cents	Per Cent	DOLLARS	Cents	OUNCES	10ths	DOLLARS	Cents	OUNCES	100ths	DOLLARS	Cents	DOLL
				18 1/10	28	96	1	2		54		2		41	2
Sulphur	38 1/10														
Iron	39 6/10														
Insoluble	3 5/10														

4,266

Duplicate

Wm H. Stowell & Co.
M. W. S. i

THIS AGREEMENT OF OPTION made this 28th day of December 1915 by and between SPENCER PENROSE, CHARLES L. TUTT and WILLIAM THAYER TUTT, as Trustees of the last will and testament of Josephine Thayer Tutt, deceased, late of El Paso County, Colorado, parties of the first part (hereinafter termed the Trustees), and JOHN HAMPSHIRE of the City of Portland, State of Oregon, party of the second part, WITNESSETH:

1. That for and in consideration of Ten Dollars ($10.00) to said Trustees in hand paid by said Hampshire, the receipt whereof is hereby acknowledged, and on further consi eration of the preliminary examination work to be done and performed by said Hampshire upon the property hereinafter described, and in further consideration of the covenants and agreements hereinafter contained to be by said Hampshire duly kept and performed, said Trustees have granted and by these presents do grant to said Hampshire the sole and exclusive option to purchase at any time up to and including the 31st day of March, 1916, the property and premises hereinafter described for the agreed price of One Hundred Forty Four Thousand ($144,000.00) Dollars in lawful money of the United States net to the Trustees, payable in three installments in time and manner following, that is to say:-

$25,000.00 on or before expiration of said option period:
$60,000.00 within twelve months after the date of said first payment:
$59,000.00 within six months after the date of said second payment.

On failure to duly pay any of said installments in time and manner herein provided, all payments theretofore made hereunder shall thereupon be forfeited to said Trustees as liquidated damages, and this agreement shall forthwith cease and determine, and all and several the parties hereto be released from all further obligation or liability hereunder.

All payments are to be made by deposit in the First National Bank of Colorado Springs, Colorado, for credit and advice and to the joint account of said

Trustees.

Upon payment of the first installment of the purchase price herein provided the certificates representing the Tutt Estate holdings of shares of stock affected by this option (i.e. 13,917 shares Waldo Smelting & Mining Company) and a deed or deeds of conveyance of the other property effected hereby shall be deposited in escrow with said Bank for delivery to said Hampshire upon due completion of all payments to be made by him hereunder as herein provided, or failing such payments or any one of them as so provided to be returned to said Trustees.

Said deed or deeds shall effectually convey good and marketable title to the premises herein-after described and all thereof, free and clear of encumbrances, to said Hampshire or to such other grantee or grantees as said Hampshire shall have designated by notice in writing to said Trustees prior to the execution and deposit in escrow of said deed or deeds.

2. The property and premises affected hereby are:-

14,867 shares (par value $100.00 each) of The Waldo Smelting & Mining Company, a corporation of Colorado; it being expressly understood that 950 of said shares are not now held by said Trustees and that deduction from the last installment of purchase price aforesaid shall be made at the rate of ten dollars per share for each of said shares which shall not be deposited by said Trustees in said Bank in escrow hereunder within said option period, unless subsequently tendered to and accepted by said Hampshire.

It is understood that property now owned by the said The Waldo Smelting & Mining Company consists of the following mining claims and ranch lands, to-wit:

(Mining Claims)

California,	Oregon,
Copper Queen,	Kelly,
Copper King,	Mabel,
Cowboy,	Morgan,
Ben Bull (or Brandt)	Dorothy,

containing about 222.168 acres:

(Ranch Lands)
Streng Ranch)
Jamison ") all in Section3, Twp. 41 S. Range 8 W.
The Lodge)

containing about 320 acres.

It is further understood and agreed that upon said Hampshire exercising his option to purchase and completing payments herein provided, the Directors of The Waldo Smelting & Mining Company representing the interests of said Trustees shall resign as such Directors and names as may be designated by said Hampshire elected in their stead;

All right, title and interest of The Takilma Smelting Company, a corporation of Colorado, in and to the following described premises situate and being in the County of Josephine, and State of Oregon, that is to say:

(a) The fractional acreage, being 20 acres more or less, in the southwest corner of the southwest fractional quarter of northwest quarter of Section 25, Twp. 40 South, Range 8 West of the Willamette Meridian;

(b) The northwest quarter of Section 36 of said Twp. 40, containing 160 acres, more or less, and known as the Queen of Bronze mine;

(c) The west half of the northeast quarter and the east half of the northwest quarter of Section 35 of said Twp. 40, containing 160 acres, more or less; together with the smelter, plant and other buildings and improvements thereon, including all machinery, tools, appliances, equipment, ores and dumps in or about said premises.

It is further agreed and understood that upon said Hampshire exercising his option to purchase and completing payments herein provided, the properties hereby affected shall be transferred and turned over free and clear of all incumbrances, except such indebtedness as may be now due said Trustees, as evidenced by promissory notes, said notes (less any payments made thereon by the application of any royalties received to a reduction thereof) to be delivered with the property hereby covered.

3. From and after the execution hereof and throughout said option

period the said Hampshire shall have the right to enter upon the premises of said The Waldo Smelting & Mining Company and also the premises above described of The Takilma Smelting Company, for the sole purpose of prospecting, exploring and examining the same, which said operations shall in no way interfere with operations of lessees now in charge and operating the Queen of Bronze mine of The Takilma Smelting Company and the Cowboy mine of The Waldo Smelting & Mining Company, and shall have the right to use the assay office and its appliances now upon said premises. It is understood and agreed that no hand tools, appliances, equipment or supplies now upon said premises shall be used by said Hampshire except upon payment therefor to said Trustees at a price to be agreed upon by said Hampshire and said Trustees, or their representatives; payments so made, as above provided for tools, appliances, equipment and supplies to be refunded to said Hampshire upon the payment of the final installment of purchase price herein provided, otherwise to remain the property of said Trustees as payment for material sold. No Tools, appliances, equipment or supplies of any character whatever shall be removed from any one of said properties for use on another property whether covered by this agreement or not without specific permission so to do of said Trustees or their representatives. No ore found or extracted during said option period shall be shipped or sold from said premises except by the express written permission of said Trustees or of said Companies, respectively.

4. As a further consideration for this option and irrespective of its exercise, said Hampshire is to furnish to said Trustees correct copies of all plats, engineer's reports and other results of examination or inspection of the properties of said Companies, including the premises hereinbefore agreed to be conveyed, as and when made by said Hampshire or to or for said Hampshire or his assigns by any surveyors, engineers or experts employed for that purpose.

In case said Hampshire shall exercise said option to purchase, and shall

make payment of the said Twenty Five Thousand ($25,000.00) Dollars on or before the first day of April, 1916, and the said Lessee now operating the Queen of Bronze, and Cowboy mines, shall have expended an amount up to, but not exceeding $3,000.00 in development work on said properties, then the said Hampshire agrees to repay to said Lessee such amount so expended not to exceed the sum of $3,000.00.

5. Within thirty days from the date hereof said Trustees will execute or cause to be executed and deposit in escrow in said Bank, for delivery to said Hampshire upon exercise of said option by payment of said initial installment of ($25,000.00) Twenty Five Thousand Dollars, but failing such payment, to be returned to said Trustees, good and sufficient leases of the smelter and all other premises and property above described of said The Takilma Smelting Company, and of all premises and mining property of said The Waldo Smelting & Mining Company. Said Leases shall run to said Hampshire or to such other Lessee or Lessees as said Hampshire shall have designated by notice in writing to said Trustees prior to the execution and deposit in escrow of said lease or leases, and shall grant the right to use, operate and mine the demised premises until completion of said agreed purchase price in time and manner above provided, and shall contain such usual clauses and provisions as said Trustees or the Lessors therein named may reasonably require. The rental under said leases shall be a royalty on all ores extracted, shipped or sold from the premises thereby demised, whether as crude ores or in matte, 33-1/3 per cent of the net smelter returns on shipments from the Queen of Bronze mine, of The Takilma Smelting Company, and 25 per cent of the net smelter returns on shipments from the Cowboy mine, of The Waldo Smelting & Mining Company, from the Tacoma plant of the Tacoma Smelting Company, or such other smelter as may be from time to time designated in advance by notice in writing from said Hampshire to said Trustees or one of them, after deduction by it of treatment and railroad transportation charges only, all payments of royalty or rent thereunder to apply

upon the purchase price aforesaid; Provided Always, that upon failure to pay any installment of said agreed purchase price as and when due hereunder, all of said leases shall forthwith cease and determine and the premises affected thereby be surrendered forthwith to the Lessors, free and clear of all liens and encumbrances.

6. Said Trustees will cause abstracts of title to the real property of said Companies, respectively, to be prepared or extended to the date hereof with all convenient speed, and will thereupon furnish thesame to said Hampshire or his attorneys for examination. Said abstract of title shall remain the property of said Trustees unless and until said option be exercised and the purchase price be paid in full in time and manner above provided.

7. The option to purchase hereby granted is an entity, and cannot be exercised as to part of the premises and property affected hereby but only as to the whole thereof.

8. No assignment by said Hampshire of his rights hereunder shall be effectual unless and until notified by telegram or letter to said Trustees or one of them.

9. This agreement is to be deemed a Colorado contract and shall be binding upon and enforcible by the respective successors, heirs, executors, administrators and assigns of the parties hereto, time being of its essence in all particulars.

IN WITNESS WHEREOF the parties hereto have hereunto set their hands and seals the day and year first above written.

Executed in duplicate originals.

Signed, sealed and delivered in the presence of:

J A Hull } as to Trustees

James E. Wilson } as to John Hampshire

SPENCER PENROSE, CHARLES L. TUTT and WM. THAYER TUTT, Trustees of the last will and testament of Josephine Thayer Tutt, deceased.

By Charles L. Tutt (SEAL)

By Wm Thayer Tutt (SEAL)

John Hampshire (SEAL)

THIS AGREEMENT OF LEASE, made and entered into this 1st day of July, A.D. 1919 by and between John Hampshire, Trustee, Lessor, and C. E. Tucker, Lessee, WITNESSETH:

That the Lessor for and in consideration of Twenty per cent (20%) royalty on ores from the "Queen of Bronze" mine and Twenty seven per cent (27%) royalty on ores from the "COW BOY" mine of the NET SMELTER RETURNS of all ores shipped by the Lessee, does hereby let and lease the above described premises to said Lessee, beginning on this date and to expire on January 1st.,1921.

In consideration of this lease the Lessee agrees with the Lessor as follows:

1.

To enter upon said premises, and to commence work thereon within ten days from the date of this lease, and to work the same mine-fashion in manner necessary and appropriate to good and economical mining, so as to take out the greatest amount of ore possible, with due regard to safety, development and preservation of the same as a workable mine.

2.

To work and mine the said premises steadily and continuously during the term of this lease, with not less than 100 shifts, during each and every calender month of the term of this lease, and failure to work said premises at all for ten consecutive days may be considered by the Lessor a violation of this lease and shall, at the option of the Lessor, and without demand or notice, work a forfeiture of this lease.

3.

To well and sufficiently timber with strong, well fitted and durable timbers, all the workings on the premises at all points where proper, in accordance with good mining, and to promptly re-

by any cause whatsoever; to keep the timbering of said workings at all times in good, safe and serviceable condition.

4.

To keep at all times all drifts, tunnels, stopes and other workings on said premises thoroughly drained and clear of loose rock and rubbish, and in absolutely safe and securecondition, and to maintain at all times and leave the floors of all tunnels, drifts, cross-cuts and other workings in good, even, unobstructed condition.

5.

To do no underhand stoping, whether for the purpose of prospecting or taking out ore, or for any purpose whatsoever.

6.

To make and keep all tunnels, drifts, cross-cuts, raises and winzes not less than three and one-half feet by seven feet in the clear of all timbers.

7.

To keep said premises and every part thereof at all times free and clear of all liens, and to settle all bills that may be incurred during the time of this lease, before final settlement shall be given for any ore shipped.

8.

To allow no person not in privity with the parties hereto to take and hold possession of said premises or any part thereof under any claim or pretense whatsoever.

9.

To assume all responsibility for, and save the Lessor harmless from any and all accidents to the Lessee or any of his employees agents, associates or visitors, upon the property of the Lessor.

other person, on account of any such accident.

10.

To sort all ores to as high a grade of copper content as possible. To keep in repair necessary bin or bins at the Takilma Smelter.

11.

The Lessor is hereby authorized to ship such ore as may be mined by the Lessee, whenever a carload is available.

12.

The Lessee further agrees, that in case the Lessor sells the aforesaid properties during the term of this lease that this lease may be cancelled upon such sale being made, and the Lessor agrees that should such sale be made the Lessee will be granted 30 days time to extract and ship any ore that he may have discovered and opened up on the premises, before the cancellation of the lease.

13.

The words " Net Smelter Returns" are taken to mean the net cash returns from the smelter, in arriving at which only the actual cost of railroad freight and charges for treatment are deducted from the gross value of the ore. It being agreed and understood that all charges for mining, hauling, assaying, umpire etc. are to be borne entirely by the Lessee.

14.

Time is the essence of the agreement.

15.

All of the operations of the Lessee under this lease shall be conducted as to fully comply in every respect with the laws so of the State of Oregon.

IN WITNESS WHEREOF the lessor and lessee have hereunto

set their hands and seals the day and year first above written.

Executed in duplicate originals.

Witnesses.

[illegible] [illegible] Seal

Lessor.

[illegible] Harris [illegible] Seal

Lessee.

There are 15,000 tons of sulphide ores carrying sufficient copper and gold for profit. A careful sampling of the surface of the dump shows the average to be .035 oz. gold, .09 oz. silver and 2.57% copper per ton of sulphide dump ore. The sulphide ores are carried mostly in a dump apart from other dump material and are not oxodized except for the very fine portions. The dump was originally piled on a steep slope but later broke thru the center and spread over an area twice that of the original base. This permits ov a better sampling job than is usual. While no holes were dug down to bottom deep trenching of the surface and breaking down of coarse cumps resulted in a sample to twenty tons finally cut down to one ton and shipped to Tacoma, for sampling. At least two other sets of sampling give a fair check.

The sulphide dump resulted from the mining and sorting of 10,000 tons of copper-gold shipping ore the average content of which was roughly .15 gold and 10% copper per ton, Tacoma Smelter returns, settlement sheets. From such information as there is available it seems that about 10,000 tons of the sulphide dumps or that portion represented in the most southerly ore dump accumulated as a result of sorting out the same tonnage of shipping ore. From this it is fair to assume that the shipments represent one half and the sulphide dump one half of the ore extracted from what is known as the McAuley and East ore shoot. Assuming this to be correct the average copper-gold content of the ore in place was roughly 6.3% copper, and .09 oz. au. Sampling results and remarks in the 1919 report of G. E. Stowell varify the conclusion to a reasonable extent. The mine was open and in the hands of leasers at the time of the Stowell examination. Nothing of more recent date is available.

A preliminary flotation test made at the Wallace plant of the A. S. & R. on the sample lot of dump ore combined with a sample of mine ore of similar character shows that a recovery of 85% of the gold and 95% or the copper can be made in the form of an 8½% copper and .12 oz. gold concentrate from ore containing originally .05 au., .14 ag. 3.19% copper.

Flotation tests were made by Southwestern Engineering Co., M. S., and the Bureau of Mines. From letters and records in Mr. Hampshire's office it seems that none of the tests were carried out to a final conclusion. The testing was done with the idea of producing a high grade concentrate and the results established were not favorable as practically all of the gold was lost and from ten to twenty percent of the copper. The ore is a heavy chalcopyrite-pyrrhotite-pyrite combination and rarely exceeds 13% copper except at carefully selected points. It is fairly evident that a low grade concentrate offers the best chance for profitable recovery.

Rough hand sorting of the ore, mined and shipped, has established a reliable marker. It is not difficult to keep the grade up to 10% copper when the better areas are being mined. No doubt a well established sorting plant would recover a higher proportion of the crude shipping ore and eliminate more of the waste serpentine resulting in a better material for a milling plant. As it seems that a low grade concentrate will result if concentration is undertaken and a good recovery made, there is not much reason for disturbing the usual practice of shipping crude ore except to recover the gold and copper lost in ore of less than profitable shipping grade. Further testing is necessary in order to determine just where the economical or most profitable shipping grade ends. The grade of crude ore and the rejected material available for concentration are both of profitable copper-gold content, at the present copper price, if a sufficient tonnage can be established.

Mine Ore-Block A.

Block A contains a probably 10,000 tons of irony ore carrying .05 and 2.60% copper. The openings thru the block are mostly caved and the ore exposures limited. It is not possible to make an assay chart of the ore but a limited amount of work in opening caves in C Level and a few short

and drifts would make sampling and charting simple. The openings were driven in order to reach the better grade ore to the east and following the ore thru the block evidently was not given much consideration as the ore is mostly in the hanging wall of the openings. The ore exposures usually show better than five feet across the dip the dip or strike are difficult to define in any case.

Some shipments of crude ore were made from the irony block A ore during 1916 and 1918 and more recently as September and October 1928. An occasional profitable car was recovered by leasers but the usual result was near .05 gold and 6% copper per ton, or below profitable shipping grade.

Besides the sampling and testing done during September and October 1928, there is some evidence of record which is, I believe, fair evidence in establishing an ore grade in so far as block A is concerned. Earlier crude shipments check with the later ones. Roughly 6% copper resulted when crude shipments were sent out from the area. During 1917 a jig plant was planned and later built and placed in operation. A test lot of 7000 pounds of the iron ore from the Messinger stope is recorded in C. E. Stowell's report of June 1917. The assays are given as 3.4% copper, gold not recorded. The jig concentrates are given as containing copper from 4½% to 5½% and in one instance slightly over 6%. This is about what would be expected after sorting the waste serpentine from the mine run 3% copper ore.

A ton sample of the solid sulphide ore, in and around the Messinger stope, as now showing in short exposures, assayed, .065 au., .13 ag., 5.05% cu. The ton sample cut down to a hundred pounds and shipped to Tacoma for assay. This sample was forwarded to the A. S. & R. Wallace plant for testing. The ore widths as exposed are 5 feet. The margins of the old stope show that the stope widths are in all cases more than 5 feet.

Six samples were cut, at points indicated on the accompanying map, from short solid sulphide, exposures. The composite result shows width 5 ft., .09 au., tr. ag., 5.6% cu. These samples were taken to check the crude shipment grades and not to indicate average ore grades. The workings are almost inaccessible and the ore exposures not sufficient for reliable sampling except at the points indicated. A limited amount of work in opening the caves on 9 level would clear up the question of continuity of ore and make sampling possible.

A 500 ton lot of ore mined from the Twohy stope, September and October 1928, was hand sorted and the rejects sampled and assayed. The reject sample and tonnage from mine car lots. Weights estimated, the results as below:

Lot 2034	50 tons	.06 au.	.22 ag.	6.4% cu.
2207	" "	.05 "	.24 "	5.35 "
Rejects	400 "	.05 "	.20 "	1.78 "
Av.	500 "	.05 "	.20 "	2.6 "

The Twohy stope ore is well opened and it is probably below the average of the Messinger ore at the top of block A. I believe that it is safe to estimate the block A ore as at least an average of the Twohy ore.

The cost of producing a ton of crude shipping ore, labor and supplies, as established during September and October of this year, is almost exactly 10 dollars per ton. The mining was carried on in a small tho economical manner. The equipment used, a two drill gasoline driven compressor, and the usual machine drills and accessories. The ore was hand thru an inadequate plant. The sorting cost could be lowered and the ore grades improved. As further mining should be carried on in the same manner the cost established is retained for estimation purposes.

Mine Ore-Block A.

Five tons of rock, ore and waste, were mined in order to recover a ton of sorted shipping ore. The rejected material from the sorting plant contained roughly one fourth barren serpentine. The need for a properly planned sorting plant is evident whether any future mining is based entirely on crude shipments or crude and concentrate shipments combined. If the total cost is distrfbuted the cost per ton mined stand at two dollars. The record is made on a thousand tons mined during September and October. The crude shipments were below profitable, however, the cost per ton is retained. It is not probable that block A will produce profitable grade crude ore, except in limited tonnage. There is a natural segregation of values in the ore in place and that contained in the block is of lower grade ore. The earlier crude shipments originated in the more easterly sections of the mine and that region probably remains the more promising, so far as can now be determined.

Mine Ore-Block B.

The only ore exposure accessible in block B is that in the east face of the Leach drift. The Leach ore was opened in October, this year, and about ten tons sorted and shipped with smelter lot 2785. It is almost sure that the higher gold shown is due to the ore from this point. A mine run sample, assayed by Gerry Johnson, Grants Pass, shows a .80 gold, 3% copper. The Leach drift was not continued. The ore shows as an isolated bunch. A continuation of the Leach, to the east, and a raise connection from C. level is the first and most important development work confronting the [illegible] promising as the larger and most profitable ore body mined, or partially mined, came from immediately above. While C level, North Cross-cut, was driven under the block B ore it is fairly certain that the dip carries the ore over and to the east. A raise driven from C for a connection with a proposed easterly extension of the Leach drift offers the least expensive and quickest method of proving whether the East and McAuley ore carries downward. The cost would be 10,000 dollars and if ore is found it could be opened down to C level for 25,000 dollars and made available.

The mine was closed during 1919, C. E. Stowell, Oregon Bureau of Mines, examined and reported ore exposures at the time. Stowell sampled and recorded four sample cuts and remarked that the ore showed in the east end and bottom of the East stope. Cameron and Messinger, leasers, mined and later shipped the last car of ore from the area. The record is here given:

Stowell	Location	Width	au. ozs.	ag. ozs.	Cu. %
	East Stope	5 ft.	.04	tr.	4.8
"	" "	8 "	1.00	"	4.0
"	" "	6 "	.08	"	4.7
"	" "	7½ "	.06	"	7.0
Tacoma Smelter		61 tons			
C. & M.	" "	car lot	.09	.39	10.13

The McAuley and East Stope crude shipments, smelter records, show 10,000 tons of .15 oz. gold and 10½% cu. per ton and the Stowell record and Messinger shipment of about the same grade ore exposure in the Leach drift below are sufficient reason for reopening the area. If the ore continues down to C level the result should about duplicate the 1916 and 1919 production of 10,000 tons crude shipping ore and an equal tonnage of lower grade ore as represented by the sulphide dump ore, or 2.6% copper and .05 gold per ton.

Hampshire Stope-Block C.

C level, south drift, was originally driven into ore where indicated. The work was extended by Clarke 1916-1917, but the ground permitted to cave. During 1928 the ore was followed north and the Hampshire stope opened. A car lot, 50 tons, were shipped to Tacoma, last October, and a net check of 1190.45 dollars returned after freight and treatment, or 23.60 per ton. The 3.60 dollars is represented by gold content and is not to be credited to Hampshire ore as the gold evidently came from the Leach ore introduced in the shipment. The car lot shipment contained copper, silver, gold, as below:

Place	Smelter	au. ozs.	ag. ozs.	Cu. %
Hampshire	Tacoma	.19	.64	11.42

There is nothing on the surface to indicate the copper ore showing in the Hampshire stope and there was nothing underground till driven into. The South mine lies a thousand feet directly south of the Hampshire and it is reasonable to suppose that any work driven south from the Hampshire area and toward the South mine presents the same chance for ore discovery as the Hampshire did before it was prospected.

Cow Boy Mines.

The Cow Boy mines present somewhat a different problem than that at the Queen of Bronze proper. The ore is a pyrrhotite-chalcopyrite carrying uniformly from ten to twelve percent copper, no gold or silver of consequence. The ore shoots are short poorly defined spots in a broad intensely serpentinized flat lying structure. The country rocks seem to have consisted of flow rocks, probably andesite shot thru with porphyretic rocks, along the ore zones. The original structure is entirely lost or nearly so. A marker for following ore structure may be found by assuming that the Cow Boy ore shoots are following along a bed of formation formerly more muddy or more easily altered than the immediately underlying or overlying beds. The Cow Boy ore shoots are found along a bed much more highly serpentinized than those outside the ore structure and so long as the condition remains there is an equal chance for finding ore. There is nothing to indicate a termination of the ore shoots either laterally or horisontally and so long as the work is carried in the same formation and ore of profitable grade found there is no reason for stopping. There is no marker pointing to the isolated ore shoots and no way of determining the extent other than to mine them out.

Two car lot shipments were made to Tacoma, during October 1928. The shipments consisted of fifty tons of 13% copper ore and fifty tons of 11%. The two lots returned the owners 2,000 dollars or 20 dollars per ton after freight and treatment deductions, the settlements made at 15¢ copper. The labor and supplies cost were ten dollars per ton leaving a net profit of ten dollars per ton. The figures are approximate but close to the actual results. There is positively no way of estimating tonnage at the Cow Boy mines and it is doubtful whether any recovery can be made other than by hand sorting crude ore. Nothing has ever been done to open the ground ahead of immediate requirements. It is probable that a reasonable development program would result in a profit to the miner, at least, it always has with copper at 15¢ or better.

OTHER ORE SHOWINGS.

The Mable and Lyttle mines lying between the Queen and Cow Boy have some productions records but nothing is of record and of a reliable character

Other Ore Showings.

Queen of Bronze group. The production similar to the Queen and about of the same grade and character of ore. Since the Waldo is closely adjoining it should be taken into the consideration especially if an attempt is made to reopen and work the Queen properties. It is not probable that any one of the five mines here mentioned would justify the expenditure required for development and plant establishment but a combination of the group, including the Waldo are worthy of consideration.

Estimates.

Before and ahead of any other consideration or expenditure the development of the downward extension of the East ore shoot is necessary. This job involves opening C level for ventilation as well as opening block A for sampling. The cost should not exceed 25,000 dollars and if the East ore shoot is proven and of approximately of the same grade and dimensions as that mined, the Queen will then present the following tonnages and grades:

East Ore Shoot	tons	crude	au. ozs.	cu. %
Block B "	10,000	"	.15	10.00
" " "	10,000	mill	.05	2.60
Block A	10,000	"	.05	2.60
Sulfide Dumps	15,000	"	.05	2.60
	35,000	"	.05	2.60

The sulphide dump ore is positive and the grade is fairly well established. A fifty ton car shipment is advised to establish grade definitely. The loss on shipping fifty tons crude to the smelter would be about 300 dollars, or about the truck and freight cost and less than the sampling cost.

The mill grade ore estimated in block A is fairly well established. Opening the cores in C tunnel and mining and shipping, crude, a fifty ton lot would establish the grade and tonnage sufficiently for estimate. The loss would be the same as for the dump ore, 300 dollars, but the result would be conclusive and less than that of sampling and charting by the usual methods the cost.

The estimate for block B ore is entirely probable and the grade and volume based on former production records, and the assumption that the East ore shoots are not bottomed and that they will extend down to C level. The development program outlined is justified and is necessary in order to place sufficient tonnage in sight for the basis of any expenditure. The estimate of 10,000 tons of 2.6% copper ore in block B is made on the production record of the top of the same ore shoot and represented by the ore dump accumulated from that source.

Costs are based on those actually experienced during September and October, this year, in so far as mining costs are concerned. Development costs are covered in the lump sum of 25,000 dollars requires for opening the two blocks, A and B. It should be noted that the more expensive development costs are already taken care of in so far as the estimates here made are concerned. The main openings are now provided for in works already driven and open.

Estimates.

For the purpose of this estimate the figures used are those established by actual record of the past seasons operation: Mining cost, including sorting per ton of crude ore produced, including labor and supplys, only. Truck haul cost per ton actual contract cost. All mining and sorting cost charged to crude production per ton.

Smelter deductions based on those as shown for smelter lot 2785 dated at Tacoma, October 29, 1928. Date of receipt October 20. The quoted price of copper 15.250¢. For this estimate the flat price of 1500¢ is used. The grades are taken from the last car shipped from the East stope by Cameron and Messinger, dropping the fractional figures, using instead of 10.13% copper the flat grade of 10.00%. The shipments from the productive area were uniformly above the grade hereused, especially the gold content which always stood at about .15 ozs. per ton instead of.10 oz.

Crude ore, mining and sorting,	Per ton	10.00
Truck haul, to railroad	" "	2.75
Development, estimated	" "	3.00
Total		15.75
Copper @.15¢ less .025¢ or .125¢		
" 200 lbs less 15 lbs deducted @ .125¢ per ton		23.12
Gold, placed at $ flat		2.00
Pay for copper and gold, (silver below pay.)		25.12
Less freight and treatment charges,		7.25
Crude Ore. Net smelter, per ton.		17.87
Development, mining, sorting, trucking		15.75
Indicated profit per ton of crude ore at .15¢ copper		$2.12

It is evident that there would be no profit with copper at 14¢ but, there would be a profit of five dollars per ton of crude ore at a 16¢ copper market. The crude shipments from the Queen mine have always averaged around .15 ozs. gold per ton. Shipments from the lower grade irony material show uniformly about .15 oz. gold. It is probable that actual results would return an additional dollar per ton from gold and that the gold content of the ore actually would represent the profit so far as crude shipments are concerned, when all mining costs arecharged to crude shipments. The better chance for a profit seems to be tied to a combination of crude and concentrate shipments especially as the crude shipment method has resulted in the accumulation of an equal tonnage of ore beneath shipping grade.

The grade of so called milling ore is fairly well established from sampling of the sulphide dumps, exposures under ground, and actually mining and sampling 500 tons of mine run ore from the Twohy stope. The estimate of 10,000 tons 2.6% ore in block B is based on the idea that the East and McAuley ore shoots will open down to D level. This estimate does not take account of whether there may be ore below D level or at any other point besides the areas indicated. If ore is opened by the development indicated, and of the grade and tonnage estimated the profit probably will supply the development fund required for further exploration. Profit here is given only as that figure above the actual cost of mining, sorting, milling, trucking, transportation and treatment. Purchase of property, financing, overhead, etc., are not considered.

Testing for flotation concentration is not conclusive tho of sufficient reliability to point to the conclusion that a concentrate of 10% copper and .15 oz gold per ton, can be accomplished without excessive loss. The concentrate would, therefore carry almost the same copper-gold content as the crude ore. It may be that a better grade concentrate could be made tho it is doubtful. Mining costs and development are already charged against crude ore, therefore, so far as the dumps and underground ore, under consideration, are concerned milling costs are the only additional charge. The same settlement figures are used, except, that a 26 lb. copper deduction is made, per ton, instead of the 15 pound deduction for crude ore.:

Milling grade ore under estimate	35,000 tons
Grade of ore, 2.6% copper, .05 oz. gold	
Ratio of concentration, 4 to 1	
Recovery, Copper 95% plus, gold 80%	
Milling and sorting cost, 4 tons @ 1.50, conc. cost.	6.00
Trucking from mill to railhead, estimated,	3.00
Cost per ton of concentrate, at railhead,	9.00
Net smelter return, after freight and treat, 15¢ cu.	17.50
Profit per ton concentrates	8.50

Indicated profit:

8750 tons	concentrates	@ 8.50	74,375.00
10000 "	crude ore	@ 2.12	21,200.00

Indicated profit, 15¢ copper, $ 95,575.00

Conclusions:

The indicated profit here given is based on the idea that an expenditure of twenty-five thousand dollars will open ore beneath the formerly productive ground. The expenditure is justified by the evidence available. A lesser tonnage, or that assured tonnage in the sulphide dumps and in block A, is not sufficient for profit. It is assumed that further exploration is justified after the preliminary project is proved. The milling cost charged against the dump ore is the same as for the underground ore. As a matter of fact the cost would not exceed fifty or seventy-five cents per ton, however, there does not seem to be a very good chance for finding crude ore within block A, and as the success of the property depends largely upon whether or not crude shipments can be maintained it is advisable to retain the higher cost figure.

The cost of a small prospecting plant, such as the one now on the ground and available, would be about 10,000 dollars. Before any headway can be made with the ore after mining, it will be necessary to provide a sorting plant of modern design. Such a plant can be provided for another 10,000 dollars. The cost of a modern flotation plant, of fifty tons capacity, would not exceed 50,000 dollars including power plant. The items here given aggregate 70,000 dollars and it would not be required that any expenditure be made for housing men or other camp construction as the little community of Takilma, provided the men for the operation of the property and the housing and boarding of such men. The majority of them live in Takilma and when given the opportunity they are glad to report for work.

The figures given indicate that a profit or 25,000 dollars can be

opment work is required. If the job is undertaken and proves successful the operation could be extended, first to the adjacent areas, and later on to the South mine, by extending the work on south and from the Hampshire ore area, and finally on to the Waldo. The outlook would not be good when copper drops below 14¢, if it does, but the prospect for spreading the operation beyond that of a small prospect when copper is at the present high level is good.

Considerable has been written as a result of geological examinations of the Queen of Bronze properties and other southwestern Oregon ore deposits of like or similar character. The majority opinionk amongst geologists seem to be that these deposits do not continue to any considerable depth. So far as small crude shippers are concerned the deposits do not have to go to any great depth in order to furnish a consistent crude out-put and keep it up indefinitely. It is likely that a small tonnage can be maintained from a depth of 300 feet and above.

Byron Wilson

mailed Dec. 3rd
new 1" attached

OF CALIFORNIA, N. A.
SEATTLE, WASHINGTON #95615
TRANSIT DEPARTMENT

L. A. Levensaler
512, Hoge Bldg, City

GENTLEMEN:

We are without returns on the following items, placed to your credit on date indicated below and subject to final payment:

CREDITED	PAYER	AMOUNT
Dec 1	Queen of Bronze Mine, Waldo Smelting & Mining Co Grants Pass, Ore. (Nov 28th 1923)	27560

This notice does not necessarily imply non-payment, but rather lack of advice from our correspondent—an advice which is now due. We communicate this information to you for such action as may seem best to you under the circumstances. We have traced same and will use every effort to secure returns as soon as possible.

Respectfully,

GEO. T. S. WHITE, *Manager.*

Seattle, Wash., December 19 1923

ants Pass, Oregon.
December 21, 1923.

tter from Mr. Hare E.
o, again requesting
u will you please
n any information which
tates in his letter
r to make further

e and a happy and

STEE, TRUSTEE.

Conner,
Secretary.

R E C A P I T U L A T I O N

(To simplify the work so that Mr. Tucker can check himself, all courses given here are Magnetic bearing)

COW BOY: Minimum 60' new work @ $10.00 per foot $ 600.00

Continue Tucker Intermediate Drift S. 56° E. for 25 feet (following the ore if it continues).

From end of advanced drift cross-cut N. 34° E. for 35 feet (if ore does not continue).

60' 250 3100

QUEEN OF BRC[illegible]S: Minimum 250' @ $10.00 [illegible] foot $2500.00

3110.00

Continue raise from South cross-cut D Level. (This raise is about 42 feet south of Raise #2)

New Raise, approximately 30 feet, offsetting ½ foot North for each foot of Raise

Main East drift continue East on Magnetic course N. 69° 12' East for 50 feet, then cross-cut at right angles to the north on Magnetic Course N. 20° 48' West for 100 feet; then vertically 70 feet

If no ore is found at 70-foot Level incline slightly to the North and continue Raise to 100 foot mark

CONDITIONS.

A careful examination of all accessible underground workings of the mine concludes that there is no ore of any importance remaining above "C-Level". The country within this area has been so thoroughly prospected that the chances of having missed any ore is extremely remote. I therefore recommend a complete abandonment of this section of the mine, as per my letter of the seventh of September.

Now, before planning any new development let us first determine what conditions must exist to profitably operate the mine. The only index as to values in the undeveloped areas is an average of what has been shipped from the mined areas. It is to be emphasized that this is not an absolutely reliable forecast but must be assumed for the purpose of calculating an outcome to compare with costs.

Also the outlook for the future price of copper must be considered. The present price, which is a fair pre-war average, must either increase or prices for material and labor must decrease. The present conditions cannot much longer exist. Granting a slight readjustment in both cases I have assumed 15 cents per pound a fair future price. With an expected production of 10 percent copper and $4.50 gold content, we have the following:

Copper per ton= 10%= 300# @ 15¢		$30.00
Less 26# @ 15¢ =	$3.90	
(300-26) = 174# @ 3¢	5.22	9.13
Pay for copper		$20.88 @ 17¢= 24.26
Gold per ton	$4.50	
Less 5% =	.22	4.28
Pay for copper & gold		$25.16
Less Deductions:		
Smelter treatment	$3.50 (new rate 3.50)	
R.R. Freight haul	5.00	
Truck haul	6.50	
Development & mining	11.00	35.00
Profit =		$ 0.16

Too close a margin to be considered safe. These figures very closely approximate costs as of 1916 and are of sufficient reliability to prove that the mine cannot be operated at a profit on normal market conditions.

I do not therefore recommend any attempt at mining at this time.

The history of the copper market shows that prices for the metal often rise much out of proportion to cost of production. So the only hope for the property is

to be prepared to take advantage of these peak prices when they offer.

The nature of the deposit and the distribution of mineral values precludes the possibilities of developing enough tonnage to justify considering it as a milling proposition.

Its destiny is a producer of selected or sorted ore of a medium grade that can ship a limited tonnage when the copper market will permit a profit. The history of the property justifies the belief that this profit might, at times, be considerable.

The risk of development is therefore warranted.

The mine at present has a number of ore showings but of insufficient importance to attract a prospective investor. If the property is to be operated or sold some assets further than possibilities must be developed.

RECOMMENDATIONS:

A drift run due east (magnetic N. 69°12'E) from the end of the Hall cross-cut on the "70 foot level" a distance of 120' to 125' should encounter ore. It may be necessary to do some cross-cutting both ways from the end of this proposed drift. This work is of first importance.

Also extend the "East-Drift" on the "D-Level" due east (Magnetic N 69°12'E) 180' to 200' and cross-cut first north and if no ore is found cross-cut south. This work with favorable results will improve the outlook of the entire mine.

In the "South Workings" the lower tunnel should be extended east with cross-cuts south at a point just east of the shaft and at 150' east of the shaft.

The trend of the ore in both the north and south workings is east and the dip is south.

COW BOY

With reference to the Cow Boy I recommend the following development:

Extend the Tucker Intermediate Drift due east (Magnetic N.69°12'E) 40' to 50'. Having proven the ore on this level run a drift S.50' E. (Magnetic S.70°48'E) from Station A (see accompanying sketch) lower tunnel 100' to 120' and cross-cut first north and then south.

The present showing at the Cow Boy is especially attractive and I believe the prosecution of the above work as well as following up the present ore exposures will prove the existence of some important ore bodies.

Due to the location of the Cow Boy I am not in favor of operating it on Company account but think it will be more advantageous to lease on a royalty basis not to exceed 10 15% of the net smelter returns, i.e., less R. R. freight and smelter treatment and deductions, if the lessee assumes the development or 20 25% royalty if you do the development.

RECAPITULATION

QUEEN OF BRONZE — NORTH WORKINGS	
70 foot level—150' @ 8.00 =	$1200.00
East Drift, D Level 250' @ $8.00 =	2000.00
	$3200.00
QUEEN OF BRONZE — SOUTH WORKINGS	
East drift lower tunnel 200' @ $8.00	$1600.00
COW BOY:	
Tucker Intermediate Drift 50' @ $10.00	500.00
Lower Tunnel S.E. Drift 150' @ $10.00	1500.00
	$2000.00
TOTAL	$6800.00

CONCLUSIONS:

First: Complete preparatory work detailed in letter of September seventh, 1923.

Second: While all the work is warranted it is not necessary now to do more than develop the ore in the north workings of the Queen. Until this is done I do not advise any further examination or showing the mine to prospective investors.

All development should be done by contract.

At the Cow Boy I believe Mr. Tucker might be persuaded to do the work as lessee.

After the above preliminary work has been completed I advise a final examination anda re-organization of the Company for financing and extensive development.

Respectfully,

L. Ackermann

After these rails have been assembled, they and the mine cars should be taken to D-Level and stored near the portal of that tunnel, as all work in the future will be from that level.

SECOND: The "Z" type Fairbanks-Morse gas engine and other equipment for framing timbers now at "C" Tunnel should also be removed to "D" Tunnel, as there will be no further use for this equipment at its present location.

The shed under which the engine is placed will probably fall down this winter. The posts of this shed will make good mine timbers and should be saved. I do not advise putting in additional supports or making any attempt to hold up this structure after the material it is sheltering is removed.

THIRD: The blacksmith shop, equipment, tools and drill steel at "A" Tunnel should be removed, and the shop rebuilt at "D" Tunnel; also the dry or change room and storehouse. It is not necessary that the dry or storehouse be rebuilt at "D" Level just now, but the blacksmith shop and its equipment is of immediate importance. However, all the material can be more cheaply removed to "D" Tunnel now than at any future date, for after this winter's rains the roads leading to "C" Tunnel will probably be in pretty bad shape.

FOURTH: Thorely clean out and drain "D"Tunnel. Elevate the track to grade where necessary, and generally put the tunnel in

good working condition. It is not necessary to clean out the entire level--only the main drift from the portal to No. 1 South-Cross-Cut; No. 1 South-Cross-Cut as far south as the East Drift; and all of the East Drift.

It is going to be necessary to do some timbering in No. 1 South-Cross-Cut. Whenever it is necessary to retimber the main level workings, the best timber procurable and properly prepared should be used. I have noticed that in the past some timbers with the bark unremoved have been placed in the mine. This is a very dangerous practice.

FIFTH: No. 1 Raise from "D" Level South-Cross-Cut to "A" Level should be drawn clean and kept open, as it must be used for an air raise to ventilate all of the workings south and east from this point. A heavy type door on strong hinges must be put in just north of No. 1 Raise in No. 1 South-Cross-Cut "D" Level to be used to control the air for the south and east workings on this level.

Also "A" Level from the head of No. 1 Raise to its portal must be kept open. At present it is in good condition.

SIXTH: No. 2 Raise from "D" Level South-Cross-Cut must also be drawn clean. This Raise at the elevation of the back of the 70-foot level must be protected with a heavy bulkhead to make it safe for a manway and protect it from any material falling in from above. If No. 2 Raise is not large enough for a manway and

chute, it must be enlarged for these accommodations. The manway in cross-section will be as long as the width of the raise, and its other dimension should be not less than 2-1/2 feet, preferably 3 feet, and the chute, say, not less than 4' x 6' or close to these dimensions. A new chute must be constructed, and preferably 3" planking used for the chute and partition between the manway and chute. Both the Tuckers or Messinger are familar with this class of work, and it is probably only necessary to tell them that you want a manway and chute in Raise No. 2.

SEVENTH: The 70-foot level must also be cleaned out and prepared for working.

The principal new work at the "Queen of Bronze" to be outlined in my report will consist of extending east and cross-cutting the 70-foot Level and East Drift from No. 1 South-Cross-Cut. In fact, all the exploration work in the future will be fr[om] the "D" Level.

Re: COW BOY MINE.

In my report I will also outline some development work to be done at the Cow Boy. Present indications are quite encour[ag]ing, but before any work can be done here it will be necessary to retimber a portion of the lower tunnel. Mr. Ed. Tucker and I discussed this situation thoroly, and he knows the exact sections that need immediate attention.

It is imperative that this timbering be done before [the]

rainy season sets in, even tho you decide not to do the development work, or you will soon lose a portion of this tunnel.

Of course, if you decide not to carry out the development work we discussed Tuesday, all that is necessary to be done now is to take the rails out of the workings above the "C" Tunnel at the "Queen," and retimber a portion of the "Cow Boy" lower tunnel where it is taking the most weight.

But if you have plans to do anything with the mine in the near future, it must be put in shape. At present it looks like a "last year's bird nest."

Very truly yours,

LALtF

At a meeting held January 8, 1929, of the owners of the QUEEN OF BRONZE mining properties, the following owners, representing such percentage of ownership as set after their names, attended:

The Columbia Company - - - - - - - -	60%
John Hampshire- - - - - - - - - - -	19%
M. S. Boss - - - - - - - - - - - - -	6%
R. B. Miller - - - - - - - - - - - -	6%

T. F. Ryan, six (6%) percent, and Roy Clark, three (3%) percent, were not present.

The following resolutions were adopted and are herewith subscribed to:

1. The trusteeship under which these properties have been operated is herewith dissolved.

2. Whereas, the interests of the present owners have heretofore been undivided partnership interests, they will be converted into corporate share-holding interests.

The properties shall be incorporated under the name of the QUEEN OF BRONZE MINING COMPANY, and shares of stock amounting to fifty percent of the total number of shares authorized shall be issued to the present owners, in the ratio of their percentage of interest.

The remaining fifty percent of the authorized stock shall be Treasury Stock and sold, to the end that funds so derived shall be used for the development of the prop[redacted]

3. Whereas, John Hampshire has acted as Trustee for the past seven years without salary, and further that he has, in the year 1928, spent in the development of the properties the sum of $6,000.00 (more or less) it is further resolved that said John Hampshire be allowed in stock, at the par value thereof, the following sums:

1. To advances in development work during 1928 - - - - - - - - - - -	$6,000.00
2. To allowance for salary and service from 1922 to 1928 inclusive - - - - - - - - - - - -	4,000.00
Being a total allowance in stock at par value of	$10,000.00

4. Inasmuch as the properties are now indebted in the following amounts:

Taxes - - - - - - - - - - - - - - - - -	$ 737.00
Note at First National Bank - - -	1,300.00
Attorneys fees to Norton & Allen -	500.00
Total indebtedness of	$2,537.00

there is herewith levied an assessment of $2,600.00 in order that the properties may be clear of obligation when effort is made to sell Treasury Stock.

5. Inasmuch as Mr. Miller and Mr. Boss do not wish to pay their proportion of this assessment, it is resolved further that the remaining owners may pay their pro rata of such assessment, and shall be issued stock at par value for same. Any obligations left unpaid shall be paid from proceeds of sale of Treasury Stock.

6. The value of the property is herewith set up in the sum of $250,000, and it shall be divided into shares in total number of one million, at a par value of twenty-five (25) cents.

On the above mentioned basis and for the purpose of determining the proportion of interest, there is herewith appended a schedule of disposition of one million shares, predicated on the above mentioned basis, and further on the assumption that Mr. T. F. Ryan will pay his 6% of the $2,600 assessment.

In addition to the normal division of fifty percent of the stock, as set forth in Paragraph 1 of this set of resolutions, there will be further distributed from the Treasury Stock a total of 49,15[illegible] shares as follows:

To John Hampshire:

A. 40,000 shares, as set forth in Paragraph 3 of these resolutions.

B. 1,976 shares, representing an advance of $494.00, being 19% of $2,600 as assessed.

To the Columbia Company:

6,240 shares, representing $1,580, being 60% of $2,600, as assessed.

To T. F. Ryan:

624 shares, representing $156.00 being 6% of $2,600, as assessed.

To Roy Clark:

312 shares, representing $78, being 3% of $2,600, as assessed. (It is understood that Mr. Clark has already paid in $66 to apply on his

It is furthermore stated that shares alloted to the various owners in these resolutions will be issued to them only upon payment of their full percentage of the $2,600 assessment.

For the prupose of putting into effect all of the above resolutions, Philip Twohy is herewith delegated to take steps toward the organization of such a corporation and for such articles, draft subject to the approval of the interested parties.

We, the undersigned owners of the QUEEN OF BRONZE properties, subscribe to these resolutions.

_____

_____

_____

_____

_____

_____

	Owners Interest	Advances	Assessment	Total
The Columbia Company	300,000	---	6,240	306,240
John Hampshire - - -	95,000	40,000	1,976	136,976
R. B. Miller - - - -	30,000	---	---	30,000
M. S. Boss - - - - -	30,000	---	---	30,000
J. F. Ryan - - - - -	30,000	---	624	30,624
Roy Clark - - - - - --	15,000	---	312	15,312
Treasury Stock				450,848
	500,000	40,000	9,152	1,000,000
	40,000			
	9,152			
Issued ----------	549.152			
Treasury Sotck	450,848			
	1,000,000			

FINANCIAL STATEMENT

"QUEEN OF BRONZE" AND "COWBOY" MINES

NOVEMBER 30, 1923

	Dr.	Cr.
Cash in Bank		$1,445.95
Balance due C. E. Tucker	$280.00	
Estimated 1923 Taxes	700.00	
Bill of L. A. Levensaler	275.60	
Expense Acct. John Hampshire	88.00	1,340.60
BALANCE		$ 105.35

TUNNEL CONTRACT

This contract, made and entered into this 1st day of May, A.D. 1923, by and between JOHN HAMPSHIRE, Trustee, as Owner, and C. E. TUCKER, Contractor,

WITNESSETH: That the owner, in consideration of the sum of One Dollar ($1.00)to him in hand paid, the receipt whereof is hereby acknowledged, and of the covenants to be performed by the contractor, grants to the contractor a contract for performing two hundred and fifty feet (250') of underground development work on the Queen of Bronze Mine, and sixty feet (60') of underground development work on the Cow Boy Mine, Josephine County, Oregon.

The owner agrees to pay for all completed work as measured at the end of each month at the rate of Ten Dollars ($10.00) per lineal foot of drift, cross-cut or raise driven, less 15%, which amount will be deducted and held until the contract is completed. The owner further agrees to build a suitable cabin to accommodate four (4) men and a small blacksmith shop near the portal of the Lower Cow Boy Tunnel.

In consideration of this contract the contractor agrees with the owner, as follows:

III.

To work steadily and continuously during the term of this contract, with not less than ninty (90) shifts during each and every calander month of the term of this contract, and to complete this contract within one hundred and twenty (120) days from date of this agreement, unless delayed by causes for which the contractor is not responsible.

IV.

To sufficiently timber with strong, well fitted and durable timbers, all new workings that require timbering, made under this contract, in accordance with good mining, at the contractor's expense; and to promptly repair and replace all timbering which may be rendered insufficient by any cause whatsoever, in the old workings used as a right-of-way by the contractor, at the expense of the owner; except that for which the contractor is responsible, in which event the contractor agrees to replace the timbers at his own expense.

V.

To keep at all times all drifts, cross-cuts, raises and tunnels used by the contractor, thoroughly drained and clear of loose rock and rubbish, and in absolutely safe and secure condition.

clear space of at least five (5) feet of width; to maintain a grade not to exceed six inches (6") per hundred feet.

VII.

To keep said premises and every part thereof at all times free and clear of all liens, and to settle all bills that may be incurred during the term of this contract before final settlement shall be made.

VIII.

To assume all responsibility for and save the owner harmless from any and all accidents to the contractor or any of his employees.

IX.

To save all ore encountered and dump same separate from waste.

X.

Time is the essence of this contract agreement.

XI.

All of the operations of the contractor under this contract shall be so conducted as to fully comply in every respect with the laws of the State of Oregon.

: LEAD, 50c
)NLY, 50c
:, SOLD or EXCHANGED

Assay Office of Wm. H. Stowell & Co.

421 SPRAGUE AVENUE
TELEPHONE MAIN 308

Rock made for

man Investment & Spokane, Wash., Sept 25th 1929

ASSAY VALUE PER TON OF 2,000 POUNDS AVOIRDUPOIS

Copper 18c per lb.			——— per lb.			Silver at 50 cents per ounce				Gold at $20.67 per ounce				Total	
Per Cent	DOLLARS	Cents	Per Cent	DOLLARS	Cents	OUNCES	10ths	DOLLARS	Cents	OUNCES	100ths	DOLLARS	Cents	DOLLARS	Cents
65 9/10	237	20									Trace			237	20
15 7/10	56	52									2		41	56	92
3/10	1	08									2		41	1	49

Wm. H. Stowell & Co. Assayers
Stowell

TACOMA SMELTER #51 Queen

AMERICAN SMELTING & REFINING COMP.

TACOMA, WASHINGTON November 7th 19 29

BOUGHT OF Queen of Bronze Mining Co.,

Waters Creek, Oregon

MATERIAL Ore

SMELTER LOT 3462 MINE LOT DATE RECEIVED 10-29

CAR OR VESSEL 53931

QUOTATION: SILVER COPPER 17.775 LESS 2.50 CENTS 15.275 CENTS

		ASSAY	VALUE PER TON		
GROSS WEIGHT					
LESS SACKS		08	GOLD 95% AT 20.00	1 52	
NET WET WEIGHT	108 400	22	SILVER AT		
LESS 0.40 % MOISTURE		6 36	WET COPPER 127.2 LESS 15 LBS.		
NET DRY WEIGHT	107 966		112.2 LBS. AT 15.275	17 14	
			GROSS VALUE	18 66	
			TREATMENT	2 50	
NET TONS	53 983	AT	NET VALUE	16 16	872 37

ANALYSIS		TREATMENT		DEDUCTIONS		
		BASE		LESS FREIGHT 4.41	239 02	
LEAD				" Demurrage	6 00	
ZINC	.50			"		
ARSENIC				"		
ANTIMONY				"		
NICKEL				"		
				"		
				TOTAL DEDUCTIONS		245 02
				NET PROCEEDS		627 35

CAR NUMBER	WEIGHT	

NOTICE: All prices on Ore, not under contract for a specified time, are subject to change without notice.

An additional charge of $10.00 per lot, for sampling and assaying, will be made on all lots containing less than five tons.

A reasonable size sample of above lot will be held for a period of 30 days.

FORM 6-A

TACOMA SMELTER #52 Queen

AMERICAN SMELTING & REFINING COMPANY

TACOMA, WASHINGTON November 20th 1928

BOUGHT OF Queen of Bronze Mining Co.,

Waters Creek, Oregon MATERIAL Ore

SMELTER LOT 3655 MINE LOT DATE RECEIVED Nov.11,1928

CAR OR VESSEL 10896

QUOTATION: SILVER COPPER 17.775 LESS 2.5 GRADE 15.275 CENTS

			ASSAY	VALUE PER TON			
GROSS WEIGHT							
LESS TARE			16	GOLD 95% AT 20.00	1	90	
NET WET WEIGHT	108 540		25	SILVER 1 oz 1/2 "			
LESS 0.70 % MOISTURE			6 45	WET COPPER 1220 LESS 15 LBS.			
NET DRY WEIGHT	107 582			1140 LBS. AT 15.275	17	41	
				GROSS VALUE	19	31	
				TREATMENT	2	50	
NET TONS	53.791		AT	NET VALUE	16	81	904 23

ANALYSIS		TREATMENT		DEDUCTIONS			
		BASE	2 50	LESS FREIGHT at 4.41			
LEAD				" 108540 at 3.05	330	85	
ZINC	0.5			" 108540 at 56	30	31	
ARSENIC				"			
ANTIMONY				"			
NICKEL				"			
				"			
				TOTAL DEDUCTIONS			361 16
				NET PROCEEDS			543 07

NUMBER CAR	WEIGHT	

NOTICE: All prices on Ore, not under contract for a specified time, are subject to change without notice.

An additional charge of $10.00 per lot, for sampling and sacking, will be made on all lots containing less than five tons.

A reasonable size sample of above lot will be held for a period of 30 days.

FORM 9-A

TACOMA SMELTER #54 Queen

AMERICAN SMELTING & REFINING COMPA..

TACOMA, WASHINGTON December 12th 19 29

BOUGHT OF Queen of Bronze Mining Co.,

Waters Creek, Oregon MATERIAL Ore

SMELTER LOT 3952 MINE LOT DATE RECEIVED 11-30

CAR OR VESSEL 45778

QUOTATION: SILVER 49 1/2 COPPER 17.775 LESS 2.5 CENTS 15.275 CENTS

			ASSAY		VALUE PER TON					
GROSS WEIGHT										
LESS SACKS				12	GOLD 95% AT 20.00	2	28			
NET WET WEIGHT	109	060		29	SILVER AT					
LESS 0.85 % MOISTURE			9	19	WET COPPER 183.8 LESS 15 LBS.					
NET DRY WEIGHT	108	153			168.8 LBS. AT 15.275	25	78			
					GROSS VALUE	28	06			
					TREATMENT	2	50			
NET TONS	54	066	AT		NET VALUE	25	56	1	381	93

ANALYSIS		TREATMENT			DEDUCTIONS					
		BASE	2	50	LESS FREIGHT 4.75				258	86
LEAD					"					
ZINC	0.80				"					
ARSENIC					"					
ANTIMONY					"					
NICKEL					"					
					"					
					TOTAL DEDUCTIONS					
					NET PROCEEDS 908 Old Nat. Bank Bldg., Spokane, Wash.			1	123	07

NUMBER CAR	WEIGHT	

NOTICE: All prices on Ore, not under contract for a specified time, are subject to change without notice.

An additional charge of $10.00 per lot, for sampling and assaying, will be made on all lots containing less than five tons.

A reasonable size sample of above lot will be held for a period of 30 days.

FORM 8 SM 12-28

TACOMA SMELTER

Queen # 49 Lot 49

AMERICAN SMELTING & REFINING COMPANY

TACOMA, WASHINGTON, October 30th 1929

BOUGHT OF Queen of Bronze Mining Co.,

Waters Creek, Oregon MATERIAL Ore

SMELTER LOT 3354 MINE LOT DATE RECEIVED Oct. 21, 1929

CAR OR VESSEL 53366

QUOTATION: SILVER COPPER 17.775 LESS 2.50 CENTS 15.275 CENTS

			ASSAY		VALUE PER TON				
GROSS WEIGHT									
LESS SACKS				08	GOLD 95% AT 20.08	1	52		
NET WET WEIGHT	104	580		24	SILVER AT				
LESS 0.25 % MOISTURE			6	45	WET COPPER 129.0 LESS 15 LBS.				
NET DRY WEIGHT	104	319			114.0 LBS. AT 15.275	17	41		
					GROSS VALUE	18	93		
					TREATMENT	2	50		
NET TONS	52	159	AT		NET VALUE	16	43	856	97

ANALYSIS		TREATMENT			DEDUCTIONS				
		BASE	2	50	LESS FREIGHT 4.41	230	60		
IRON					"				
INSOL.					"				
LEAD					"				
ZINC	1.4				"				
NICKEL					"				
					"				
					TOTAL DEDUCTIONS			230	60
					NET PROCEEDS			626	37

CAR NUMBER	WEIGHT	

Emt

NOTICE: All prices on Ore, not under contract for a specified time, are subject to change without notice.

An additional charge of $10.00 per lot, for sampling and assaying, will be made on all lots containing less than five tons.

A reasonable size sample of above lot will be held for a period of 30 days.

FORM 8 5M 12-28

TACOMA SMELTER

Queen # 45

AMERICAN SMELTING & REFINING COMPANY

TACOMA, WASHINGTON, September 14th 19 29

BOUGHT OF Queen of Bronze Mining Co.,

Waters Creek, Oregon MATERIAL Ore

SMELTER LOT 2718 MINE LOT DATE RECEIVED 9-4

CAR OR VESSEL 45283

QUOTATION: SILVER 52 1/4 COPPER 17.775 LESS 2.50 CENTS 15.275 CENTS

			ASSAY		VALUE PER TON					
GROSS WEIGHT										
LESS SACKS				05	GOLD 95% AT 20.00		95			
NET WET WEIGHT	107	020		22	SILVER AT					
LESS 0.30 % MOISTURE			7	36	WET COPPER 147.2 LESS 15 LBS.					
NET DRY WEIGHT	106	699			132.2 LBS. AT 15.275	20	19			
					GROSS VALUE	21	14			
					TREATMENT	2	50			
NET TONS	53	349	AT		NET VALUE	18	64		994	45

ANALYSIS		TREATMENT			DEDUCTIONS					
		BASE	2	50	LESS FREIGHT 4.41	235	98			
IRON					"					
INSOL.					"					
LEAD					"					
ZINC	0.6				"					
NICKEL					"					
					"					
					TOTAL DEDUCTIONS				235	98
					NET PROCEEDS				758	45

CAR NUMBER	WEIGHT	

NOTICE: All prices on Ore, not under contract for a specified time, are subject to change without notice.

An additional charge of $10.00 per lot, for sampling and assaying, will be made on all lots containing less than five tons.

A reasonable size sample of above lot will be held for a period of 30 days.

FORM 8 SM 12-28

TACOMA SMELTER Queen 42

AMERICAN SMELTING & REFINING COMPANY

TACOMA, WASHINGTON, August 22nd 1929

BOUGHT OF Queen of Bronze Mining Co.,

Waters Creek, Oregon

MATERIAL Ore

SMELTER LOT 2441 MINE LOT DATE RECEIVED 8-13

CAR OR VESSEL 46229

QUOTATION: SILVER 52 1/2 COPPER 17.775 LESS 2.50 CENTS 15.275 CENTS

			ASSAY		VALUE PER TON					
GROSS WEIGHT										
LESS SACKS				04	GOLD 95% AT 20.00		76			
NET WET WEIGHT	104	860		32	SILVER AT					
LESS .30 % MOISTURE			8	04	WET COPPER 160.8 LESS 15 LBS.					
NET DRY WEIGHT	104	545			145.8 LBS. AT 15.275	22	27			
					GROSS VALUE	23	03			
					TREATMENT	2	50			
NET TONS	52	272	AT		NET VALUE	20	53	1	073	14

ANALYSIS		TREATMENT			DEDUCTIONS					
		BASE			LESS FREIGHT 4.75				249	04
IRON					"					
INSOL.					"					
LEAD					"					
ZINC	.60				"					
NICKEL					"					
					"					
					TOTAL DEDUCTIONS					
					NET PROCEEDS				824	10

CAR NUMBER	WEIGHT	

NOTICE: All prices on Ore, not under contract for a specified time, are subject to change without notice.

An additional charge of $10.00 per lot, for sampling and assaying, will be made on all lots containing less than five tons.

A reasonable size sample of above lot will be held for a period of 30 days.

Form 6-A

TACOMA SMELTER

#69 Queen

AMERICAN SMELTING & REFINING COMPANY

TACOMA, WASHINGTON May 10th 1930

BOUGHT OF Queen of Bronze Mining Co.,

Waters Creek, Oregon MATERIAL Ore

SMELTER LOT 1567 MINE LOT DATE RECEIVED 4-28

CAR OR VESSEL 45127

QUOTATION: SILVER COPPER 13.775 LESS 2.50 CENTS 11.275 CENTS

			ASSAY		VALUE PER TON				
GROSS WEIGHT									
LESS SACKS				17	GOLD 95% AT 20.00	5	23		
NET WET WEIGHT	94	140			SILVER AT				
LESS 1.05 % MOISTURE			9	92	WET COPPER 198.4 LESS 15 LBS.				
NET DRY WEIGHT	93	152			183.4 LBS. AT 11.275	20	68		
					GROSS VALUE	23	91		
					TREATMENT	2	50		
NET TONS	46	576	AT		NET VALUE	21	41	997	19

ANALYSIS		TREATMENT			DEDUCTIONS				
		BASE			LESS FREIGHT				
LEAD					" 100,000 4.10	205	00		
ZINC					" 100,000 .65	32	50		
ARSENIC					" Demurrage	2	00		
ANTIMONY					"				
NICKEL					"				
					"				
					TOTAL DEDUCTIONS			239	50
					NET PROCEEDS			757	69

NUMBER OF CAR	WEIGHT	

NOTICE; All prices on Ore, not under contract for a specified time, are subject to change without notice.

An additional charge of $10.00 per lot, for sampling and assaying, will be made on all lots containing less than five tons.

A reasonable size sample of above lot will be held for a period of 30 days.

FORM B-A

TACOMA SMELTER

Queen # 66

AMERICAN SMELTING & REFINING COMPANY

TACOMA, WASHINGTON April 19th 19 30

BOUGHT OF Queen of Bronze Mining Co.,

Waters Creek, Oregon

MATERIAL Ore

SMELTER LOT 1358 MINE LOT DATE RECEIVED Apr.14

CAR OR VESSEL 92129

QUOTATION: SILVER 42 5/8 COPPER 17.775 LESS 2.5 CENTS 15.275 CENTS

			ASSAY		VALUE PER TON				
GROSS WEIGHT									
LESS SACKS				16	GOLD 95% AT 20.00	3	04		
NET WET WEIGHT	108	500		16	SILVER AT				
LESS .75 % MOISTURE			8	44	WET COPPER 168.8 LESS 15 LBS.				
NET DRY WEIGHT	107	488			153.8 LBS. AT 15.275	23	49		
					GROSS VALUE	26	53		
					TREATMENT	2	50		
NET TONS	53	744	AT		NET VALUE	24	03	1 291	47

ANALYSIS		TREATMENT			DEDUCTIONS				
		BASE			LESS FREIGHT 4.75			257	12
LEAD					"				
ZINC					"				
ARSENIC					"				
ANTIMONY					"				
NICKEL					"				
					"				
					TOTAL DEDUCTIONS				
					NET PROCEEDS			1 034	35

CAR NUMBER	WEIGHT	

NOTICE: All prices on Ore, not under contract for a specified time, are subject to change without notice.

An additional charge of $10.00 per lot, for sampling and assaying, will be made on all lots containing less than five tons.

A reasonable size sample of above lot will be held for a period of 30 days.

TACOMA SMELTER

Queen # 65

AMERICAN SMELTING & REFINING COMPANY

TACOMA, WASHINGTON April 12th 19 30

BOUGHT OF Queen of Bronze Mining Co.,

Waters Creek, Oregon MATERIAL Ore

SMELTER LOT 1201 MINE LOT DATE RECEIVED Apr.1

CAR OR VESSEL 44542

QUOTATION: SILVER COPPER 17.775 LESS 2 1/2 CENTS 15.275 CENTS

GROSS WEIGHT			ASSAY	VALUE PER TON			
LESS SACKS			14	GOLD 95% AT 20.00	2 66		
NET WET WEIGHT	108 440		15	SILVER AT			
LESS 1.30 % MOISTURE			8 05	WET COPPER 161 LESS 15 LBS.			
NET DRY WEIGHT	107 030			146 LBS. AT 15.275	22 30		
				GROSS VALUE	24 96		
				TREATMENT	2 50		
NET TONS	53 515		AT	NET VALUE	22 46	1 201	95

ANALYSIS		TREATMENT		DEDUCTIONS			
		BASE	2 50	LESS FREIGHT 4.75		257	66
LEAD				"			
ZINC				"			
ARSENIC				"			
ANTIMONY				"			
NICKEL				"			
				"			
				TOTAL DEDUCTIONS			
				NET PROCEEDS		944	29

CAR NUMBER	WEIGHT	

NOTICE: All prices on Ore, not under contract for a specified time, are subject to change without notice.

An additional charge of $10.00 per lot, for sampling and assaying, will be made on all lots containing less than five tons.

A reasonable size sample of above lot will be held for a period of 30 days.

FORM 8-A

TACOMA SMELTER

#63 Queen

AMERICAN SMELTING & REFINING COMPANY

TACOMA, WASHINGTON March 29th 1930

BOUGHT OF Queen of Bronze Mining Co.,

Waters Creek, Oregon MATERIAL Ore

SMELTER LOT 1038 MINE LOT DATE RECEIVED Mar. 21

CAR OR VESSEL 93336

QUOTATION: SILVER COPPER 17.775 LESS 2.5 CENTS 15.275 CENTS

		ASSAY	VALUE PER TON		
GROSS WEIGHT					
LESS SACKS		19	GOLD 95% AT 20.00	5 61	
NET WET WEIGHT	107 280	17	SILVER AT		
LESS 1.15 % MOISTURE		9 65	WET COPPER 192.6 LESS 15 LBS.		
NET DRY WEIGHT	106 046		177.6 LBS. AT 15.275	27 13	
			GROSS VALUE	32 74	
			TREATMENT	2 50	
NET TONS	53 023	AT	NET VALUE	28 24	1 497 37

ANALYSIS		TREATMENT			DEDUCTIONS			
		BASE			LESS FREIGHT 4.75			254 85
LEAD					"			
ZINC					"			
ARSENIC					"			
ANTIMONY					"			
NICKEL					"			
					"			
					TOTAL DEDUCTIONS			
					NET PROCEEDS		1 242	52

CAR NUMBER	WEIGHT

NOTICE: All prices on Ore, not under contract for a specified time, are subject to change without notice.

An additional charge of $10.00 per lot, for sampling and assaying, will be made on all lots containing less than five tons.

A reasonable size sample of above lot will be held for a period of 30 days.

FORM 6 SM 12-28

Quan #32

TACOMA SMELTER

AMERICAN SMELTING & REFINING COMPANY

TACOMA, WASHINGTON, June 3rd 1929

BOUGHT OF John Hampshire

Waters Creek, Oregon

MATERIAL Ore

SMELTER LOT 1464 MINE LOT DATE RECEIVED 5/25/29

CAR OR VESSEL 91915

QUOTATION: SILVER 53 3/4 COPPER 17.775 LESS 2.50 CENTS 15.275 CENTS

GROSS WEIGHT			ASSAY		VALUE PER TON				
LESS SACKS				23	GOLD 95% AT 20.00	4	37		
NET WET WEIGHT	103	940		52	SILVER less 1/4 53 3/4		01		
LESS 0.55 % MOISTURE			10	89	WET COPPER 217.8 LESS 15 LBS.				
NET DRY WEIGHT	103	269			202.8 LBS. AT 15.275	30	98		
					GROSS VALUE	35	36		
					TREATMENT	2	50		
NET TONS	51	634	AT		NET VALUE	32	86	1	696 69

ANALYSIS		TREATMENT			DEDUCTIONS				
		BASE	2	50	LESS FREIGHT 6.58				531 25
IRON					"				
SILICA					"				
LEAD					"				
ZINC					"				
NICKEL					"				
					"				
					TOTAL DEDUCTIONS				
					NET PROCEEDS			1	365 44

CAR NUMBER	WEIGHT	

NOTICE: All prices on Ore, not under contract for a specified time, are subject to change without notice.

An additional charge of $10.00 per lot, for sampling and assaying, will be made on all lots containing less than five tons.

A reasonable size sample of above lot will be held for a period of 30 days.

FORM 3-6

TACOMA SMELTER #60 Queen

AMERICAN SMELTING & REFINING COMPANY

TACOMA, WASHINGTON March 11, 1930 19

BOUGHT OF Queen of Bronze Mining Co.,

Water Creek, Ore MATERIAL Ore

SMELTER LOT 686 MINE LOT DATE RECEIVED Feb. 24, 1930

CAR OR VESSEL 46977

QUOTATION: SILVER COPPER 17.775 LESS 2.5 CENTS 15.275 CENTS

GROSS WEIGHT			ASSAY		VALUE PER TON			
LESS SACKS				18	GOLD 95% AT 20.00	5	42	
NET WET WEIGHT	104	200		25	SILVER AT			
LESS 1.05 % MOISTURE			9	80	WET COPPER 196.0 LESS 15 LBS.			
NET DRY WEIGHT	103	106			181.0 LBS. AT 15.275	27	65	
						31	07	
					GROSS VALUE			
					TREATMENT	2	50	
NET TONS	51	553	AT		NET VALUE	28	57	1 472 87

ANALYSIS		TREATMENT			DEDUCTIONS			
		BASE	2	50	LESS FREIGHT @ 4.75	246	92	
LEAD					"			
ZINC					"			
ARSENIC					"			
ANTIMONY					"			
NICKEL					"			
					"			
					TOTAL DEDUCTIONS			246 92
					NET PROCEEDS			1 225 95

CAR NUMBER	WEIGHT	

NOTICE: All prices on Ore, not under contract for a specified time, are subject to change without notice.

An additional charge of $10.00 per lot, for sampling and assaying, will be made on all lots containing less than five tons.

A reasonable size sample of above lot will be held for a period of 30 days.

FORM 6-R

TACOMA SMELTER

AMERICAN SMELTING & REFINING COMPANY

TACOMA, WASHINGTON

Queen #101

March 22nd 1930

BOUGHT OF Queen of Bronze Mining Co.

Waters Creek, Oregon

MATERIAL Ore

SMELTER LOT 854 MINE LOT DATE RECEIVED Mar. 10

CAR OR VESSEL 12670

QUOTATION: SILVER 41¾0 COPPER 17.775 LESS 2.5 CENTS 15.275 CENTS

GROSS WEIGHT			ASSAY		VALUE PER TON		
LESS SACKS				19	GOLD 95% AT 20.00	5 61	
NET WET WEIGHT	108 040			19	SILVER AT		
LESS 0.50 % MOISTURE			9 88		WET COPPER 197.6 LESS 15 LBS.		
NET DRY WEIGHT	107 500				182.6 LBS. AT 15.275	27 89	
					GROSS VALUE	31 50	
					TREATMENT	2 50	
NET TONS	53 75		AT		NET VALUE	29 00	1 558 75

ANALYSIS		TREATMENT		DEDUCTIONS		
		BASE	2 50	LESS FREIGHT 4.75	256 58	
LEAD				"		
ZINC				"		
ARSENIC				"		
ANTIMONY				"		
NICKEL				"		
				"		
				TOTAL DEDUCTIONS		256 58
				NET PROCEEDS		1 302 17

CAR NUMBER	WEIGHT	

NOTICE: All prices on Ore, not under contract for a specified time, are subject to change without notice.

An additional charge of $10.00 per lot, for sampling and assaying, will be made on all lots containing less than five tons.

A reasonable size sample of above lot will be held for a period of 30 days.

FORM 8 SM 11-28

TACOMA SMELTER

AMERICAN SMELTING & REFINING COMPANY

Queen #29

TACOMA, WASHINGTON. May 13 1929

BOUGHT OF John Hampshire

Waters Creek, Oregon

MATERIAL Ore C/10

SMELTER LOT 1149 MINE LOT DATE RECEIVED May 2,1929

CAR OR VESSEL 92917

QUOTATION: SILVER 54 3/4 COPPER 17.775 LESS 2.50 CENTS 15.275 CENTS

			ASSAY		VALUE PER TON			
GROSS WEIGHT								
LESS SACKS				18	GOLD 95% AT 20.00	3	42	
NET WET WEIGHT	108	480		35	SILVER AT			
LESS 0.30 % MOISTURE			10	45	WET COPPER 209.0 LESS 15 LBS.			
NET DRY WEIGHT	108	155			194.0 LBS. AT 15.275	29	63	
					GROSS VALUE	33	05	
					TREATMENT	2	50	
NET TONS	54	077	AT		NET VALUE	30	55	1 652 95

ANALYSIS		TREATMENT			DEDUCTIONS			
		BASE	2	50	LESS FREIGHT 6.38	346	05	
IRON					"			
INSOL.					"			
LEAD					"			
ZINC	0.40				"			
SILICA					"			
					"			
					TOTAL DEDUCTIONS			346 05
					NET PROCEEDS			1306 90

ORE TREATMENT	WEIGHT	

NOTICE: All prices on Ore, not under contract for a specified time, are subject to change without notice.

An additional charge of $10.00 per lot, for sampling and assaying, will be made on all lots containing less than five tons.

A reasonable size sample of above lot will be held for a period of 30 days.

QUEEN OF BRONZE JOSEPHINE

QUEEN OF BRONZE MINE

Takilma, Oregon

Our file of data on this copper property includes a series of letters and reports covering aspects of the operation prior to 1929 and is fairly complete in the picture presented of the earlier productive era in its history.

Reports by P. Cushing Moore in 1929 conclude that there are 6% copper ores amounting to 26,000 tons positive, 48,900 tons probable and 72,000 tons possible in the various mines and 40,000 tons of 3% ore on the dumps.

Byron Wilson at the same time assigned substantially smaller ore reserves and ore grades to the property, but considered it to have possibilities as a small tonnage producer and worthy of an exploration program.

The record is then blank until the Richelson letter of May, 1949. Ore grade and tonnage in this letter are not consistent with earlier reports. E. R. Waite's summary of February 13, 1953 seems to fit the picture closer and may be the result of more detailed knowledge of the workings since Waite has owned and tried to operate the property for a number of years.

On the basis of our record of this property, there is little cause for thinking that the property has other than speculative value under present conditions.

There is one point that may be worth following up for further information on this property.

Major Waite points out that U.S. Bureau of Mines conducted some field studies of the area in 1950 under the direction of Robert Hundhausen. If Hundhausen's report is available, it might possibly throw additional light on the possibilities.

E. E. Eddy

EEE:DO

March 16, 1955

QUEEN OF BRONZE

The matter of exploratory work by the U.S. Bureau of Mines was discussed with Earl Volin, former chief in the region, early in 1954.

He reported that Robert Hundhausen was engineer in charge of the project and that he had been conducting some soil tests covering quite an area. Hundhausen was hopeful, but Volin was not at all optimistic. He thought the ore bodies, if found, would be too low grade.

A conversation with Hundhausen earlier this month brought out the information that his report on this project has not been completed. He doubts that the Bureau will undertake any further testing, certainly not right away. He remarked that the work done, i.e. the diamond drilling, was poorly directed and the results very inconclusive.

My impressions were that Hundhausen was being very careful not to express a definite opinion, but I think he is not much interested in going further with exploration.

RK:DO

Rowland King

March 22, 1955

Form 1 50M 12-21 McCann

ALASKA STEAMSHIP COMPANY

192

Cow Bay Shipments

	Tons	Au	Cu	Prod Au	Prod Cu
1916	38.212	.11	6.12%		
"	31.170	.04	12.67%		
1918	842.162	.04	13%	[illegible]	[illegible]
1919	51.953	.04	12.87%	2.07812	668.63511
"	52.151	.04	13.36%	2.08604	696.73736
"	53.539	.04	12.98	2.14116	694.80642
	157.633			6.30532	2060.17889
	873.332			.04	13.06% Cu

1031 ~~965~~ tons = .04 oz Au & 13% Cu

38.212 tons Dump .11 oz Au & 6.12% Cu.

Prior to

Queen of Bronze (Tutt)

1909	5,000 tons	.22 oz Au	9.40%	
1915	1542 "	.26 oz Au	10.98% Cu	
1918	9992	.24 oz Au	10.12	Gross Val = [illegible]
1919	56 "	.08 oz Au	10.14% Cu	
	16,590	.22 oz Au	10%	
		20	20	
		$4.50	200	
			16	
			3200	
			$36.50	

10
6.50
5.00
21.50

PORTLAND
501 WILCOX BLDG.

TACOMA
202 TACOMA BLDG.

KOBE
122 HIGASHI-MACHI

I. F. LAUCKS, INC.

ANALYTICAL AND CONSULTING

CHEMISTS - ASSAYERS - METALLURGISTS

SAMPLERS INSPECTORS

MAIN OFFICE
99 MARION STREET
SEATTLE

CERTIFICATE NO **19596**

SEATTLE, **NOVEMBER 28--1922**

WE HEREBY CERTIFY THAT WE HAVE ASSAYED THE SAMPLES SUBMITTED TO US BY

MR. L. A. LEVENSALER
HOGE BLDG.
SEATTLE, WN.

AND THAT THEY CONTAINED THE FOLLOWING PER TON OF 2,000 POUNDS:

SAMPLE MARKED	GOLD		SILVER		COPPER				TOTAL VALUE PER TON
	OZ. TROY PER TON	VALUE PER TON	OZ. TROY PER TON	VALUE PER TON	PER CENT	VALUE PER TON	PER CENT	VALUE PER TON	
		$		$		$		$	$
No. 1	0.015	0.30	NONE		10.65	29.82			30.12
No. 2	0.01	0.20	NONE		12.50	35.00			35.20
No. 3	0.01	0.20	NONE		13.55	37.94			38.14
No. 4	0.01	0.20	NONE		17.70	49.56			49.76
No. 5	0.01	0.20	NONE		15.10	42.28			42.48
No. 6	0.01	0.20	NONE		2.70	7.56			7.76

VALUATION

GOLD $20.00 PER OZ. LEAD CTS. PER LB.
SILVER CTS. PER OZ. COPPER 14 CTS. PER LB.
FOR FURTHER INFORMATION REGARDING THESE SAMPLES REFER TO SAME BY CERTIFICATE NUMBER

RESPECTFULLY SUBMITTED,

I. F. LAUCKS, INC.
BY H. W. Banks

202 TACOMA BLDG.

KOBE
122 HIGASHI-MACHI

I. F. LAUCKS, INC.

ANALYTICAL AND CONSULTING

CHEMISTS - ASSAYERS - METALLURGISTS

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		$		$		$		$	$
No. 1	0.015	0.30	NONE		10.65	29.82			30.12
No. 2	0.01	0.20	NONE		12.50	35.00			35.20
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SILVER CTS. PER OZ. COPPER 14 CTS. PER LB.
FOR FURTHER INFORMATION REGARDING THESE SAMPLES REFER TO SAME BY CERTIFICATE NUMBER

RESPECTFULLY SUBMITTED,

I. F. LAUCKS, INC.
BY I. F. Laucks

OREGON BUREAU OF MINES AND GEOLOGY
417 Oregon Building,
Portland, Oregon

Mr. Erick
Kindly return to L. A. Levensaler
512 Hoge Bldg - Seattle, Wash
soon as you are through with this. L.A.L.

REPORT

on

THE QUEEN OF BRONZE

and

ASSOCIATED MINING PROPERTIES

Waldo District, Josephine County, Oregon.

November 29, 1919.

Medford, Oregon,
November 29, 1919.

Mr. Henry M. Parks, Director,
Oregon Bureau of Mines and Geology,
417 Oregon Building,
Portland, Oregon.

Dear Sir:

Pursuant to your request I have examined the Queen of Bronze Mine and related groups and, herewith, submit my report thereon.

The time consumed in the examination was fifteen days from September 3 to September 18, 1919. My associates in the work were W. H. Whittier, assayer-surveyer, A. W. Parks and C. M. Swartley, assistants, all employees of the Oregon Bureau of Mines field party.

The nature of this report is in a sense preliminary. The magnitude of the property and the various factors involved are such that a more detailed presentation would require much more time than was available. I believe, however, that the following pages contain the essential facts as to present condition and future prospects and limitations of the property.

Respectfully,

(Signed) G. E. Stewell, Assistant
Mining Engineer.

SUMMARY AND CONCLUSIONS

The following points summarize the salient features regarding the copper property controlled by the John Hampshire Company:

1. The copper is in the form of Chalcopyrite associated with pyrrhotite, pyrite and small amounts of quarts and calcite.

2. The ore occurs in irregular, disconnected bodies in serpentine and related rocks.

3. The ores were originally deposited at considerable depth by hot mineral bearing waters arising from igneous rocks through shear zones or planes of weakness in the country rocks. Eresion has subsequently exposed these deposits at the surface.

4. A few hundred tons of ore of shipping grade are in sight. The underground workings and ore dumps show several thousand tons of low grade ore, but the tonnage is not sufficient and present transportation difficulties and high costs do not warrant the construction of a plant to concentrate this material.

5. The total gross production of the property is approximately a million and a half dollars.

During three years of operation from 1916 to 1919, the present Company's gross value production was over half a million dollars and the property was paid for out of proceeds from the mines.

6. In its present condition the property is classed as a prospect of merit but not a mine from which immediate production can be assured. The expenditure of a considerable sum of money is fully warranted to establish the occurrence and extent of possible ore bodies.

Location, Ownership and General Features

The property, including the Takilma Smelting Co. and Waldo Smelting and Mining Co. groups, is located in the Waldo district, southern Josephine County, Oregon, on the east branch of Illinois River from two to three miles north of the Oregon-California line.

The nearest town of importance is Grants Pass situated about 40 miles to the northeast on the main line of the Southern Pacific railroad. The nearest railway point is Waters Creek station on the California-Oregon Coast Railroad, 27 miles from the property by good auto road. Nearby smaller towns are Takilma about 1 mile from the Queen of Bronze Mine and Waldo about 4 miles by road to the west on the Grants Pass-Crescent City highway.

The descriptions of the various blocks included in the property are as follows: (See accompaning property map)

The N.W.¼ of Sec. 36, Twp. 40 S., R 8 W. (Willamette Meridian) 160 acres known as the Queen of Bronze; also the W.½ of N.E.¼ and E.½ of N.W.¼ of Sec. 35, Twp. 40 S., R 8 W. 160 acres smelter site known as the Takilma Smelting Co.; also the S.W. corner of S.W. fractional ¼ of N.W.¼ of Sec. 25, Twp. 40 S., R 8 W. 20 acres quarry location; also the group of 12 claims located in sections 1, 2, 11 and 12, Twp. 41 S., R 8 W. within the Siskiyou National Forest on Page creek.

This latter block totaling 222.18 acres is known as the Waldo Smelting and Mining Co. group and contains the following claims: California, Copper Queen, Copper King, Mabel, Dorothy, Marion, Lyttle, Ben Ball, Morgan, Kelly, Oregon and Cowboy. In addition there is included the Strong and Jamison ranches totaling 320 acres located on the flats in the vicinity of where Page creek joins the East branch of Illinois river.

The total acreage therefore, involved is 882.18 all of which is patented and held by purchase by the John Hampshire Co. Trustee for the owners.

Those holding an interest in the property with their proportionate shares are as follows:

John Twohy - - - - - -	60/100
T. F. Ryan - - - - - -	6/100
R. B. Miller - - - - -	6/100
M. S. Boss - - - - - -	6/100
R. H. Clark- - - - - -	3/100
John Hampshire - - - -	19/100

TOPOGRAPHY

The property is located on the west slope of a prominent ridge between Althouse creek and East branch of Illinois river. The Takilma area is made up of rather flat river and creek bottoms which rise abruptly to high ridges. The flats are valuable for agriculture and placer mining, while the open forested bushy ridges are covered with good timber and contains mineral deposits of value.

Some idea of the topography may be gained from the accompaning topographic and property map.

The elevation of Takilma is about 1600 feet, the Queen of Bronze Mine on the ridge to the east has an elevation of about 2300 feet, while the heights of the claims on Page creek vary from 2200 to 3200 feet.

The average yearly precipitation according to the U. S. Weather Bureau is about 60 inches and comes mostly during the fall and winter months. Snowfall is generally light at

This condition is being improved upon, however, by road work now in progress in which certain sections are being macadamized and grades reduced. Timber is plentiful on the property for future mining or construction needs and water may be had by diverting the flow of the several local streams.

HISTORY

The facts regarding the discovery, original locators and the early development of the property are not known to the writer. It is reported that practically no development was done previous to 1903 in which year Mr. Tutt and associates of Colorado Springs, Colorado, purchased the property. The first production of consequence was made in 1906 by the Takilma Smelting Co., a colorado Corporation, all of the stock of which was held by the Tutt estate. This company owned the Queen of Bronze Mine, the smelter and smelter site and the 20 acre claim with the limestone quarry. The group of claims on Page creek and the Strong and Jamison ranches were owned by the Waldo Smelting and Mining Co., also a Colorado corporation, all of which stock was held by Mr. Tutt excepting 950 shares owned by Mr. Hull and Mr. McNeil.

The Takilma Smelting Co. built the smelter which had a daily capacity of 100 tons and was of the semi-pyritic matte

aging about 8% copper was smelted from the Queen of Bronze Mine besides a considerable tonnage from the Cowboy and Lyttle properties.

Coke was hauled in from Grants Pass and Crescent City and the matte was hauled out and shipped to Tacoma Smelter for refining. High cost of transportation and the difficulty and expense of getting good coke caused operations to be discontinued finally.

From 1910 to 1916 the properties were worked by leasers who managed with small overhead expense to make mining profitable and shipped a few thousand tons to Tacoma by way of Grants Pass.

In April, 1916 the entire property was sold to the John Hampshire Company and operated continuously until the close of 1918 when the slump in the copper market discouraged further operation. At the time the John Hampshire Company took over the property it was estimated by Mr. Roy Clarke, managing engineer, that 300 tons of shipping ore was in sight. Shortly afterwards in running the Mc Cauley crosscut on the E level, the Mc Cauley ore body was opened up which is claimed to have produced enough ore to pay for the property.

The books in the Company's office show that during the three years' operation there was produced from the Queen

value of $75,562.91. The total net smelter returns deducting freight and smelter charges was approximately $250,000. The number of cars shipped was 183, the grade of ore varying from 5.16% to 16.33% copper and .04 oz. to .44 oz. gold. During the past year the property has been leased to Mr. Tucker, a local rancher and miner, and 2 to 3 cars per month are being shipped.

GEOLOGY - ORE DEPOSITS

Space does not permit going into details of the general geological features of the area and for this information the reader is referred to the following publications:

U. S. Geological Survey Bulletin No. 380 - Kay
U. S. " " " " 546 - Diller
The Mineral Resources of Oregon, Vol. 1. No. 5, by Winchell.

Briefly the rocks of the district include both sedimentary and igneous types. The sediments are represented by older slaty rocks, quartzites and limestones, and more recent gravels and sands. Associated with these are the igneous intrusives and possibly some volcanic flows; practically all being classified as andesite, periodotite, serpentine, auganite, pryoxenite, augitite, gabbro, diorite, greenstone, etc.

Doubtless the alteration of such rocks as peridotite, pyroxenite and gabbro have produced the serpentines so common

ted with one or more of the above named igneous rocks and so far as is known have no direct relation to the sedementaries other than that the ore deposits are found in a belt of country extending east of north in direction which donforms roughly with the strike of the sedimentary rocks in the vicinity. No sedimentary rocks were recognized in the mine workings or in close proximity to the ore deposits. The copper deposits examined show, with a few minor exceptions, a striking similarity in manner of occurrence and in rock and mineral associates.

The most common rock associateis serpentine although the ores are not confined to this rock alone but are found in adjacent rocks less altered and less basic in character.

Since the Queen of Bronze Mine has the most extensive development, a description of ore occurrence in that mine will best serve to acquaint the reader with general conditions.

The ore bodies mined out in the Queen of Bronze have been found in an ill-defined mineralized zone that is supposed to be a contact between serpentine and a less basic rock which has been classified as "auganite", but which for practical purposes may be considered as related to diorite or andesite.

The above relations are dubiously recognized underground. In places the contact between serpentine and intermediate less altered rock is sharp and again the change is gradational; suggesting that the serpentine may represent a basin border zone phase of a less basic rock mass and also that the serpentine in the district may be the result of alteration of a wider range of rock types than is generally supposed.

The ore occurs in irregular bodies and fragments varying in size from several thousand tons to a few pounds, showing broken angular shapes and slickensided surfaces, crushed into the softer inclosing rock with apparently no order or connection--good evidence that post mineral movement occurred probably at the time of alteration of the rocks to serpentine. The crushing, dragging effect, on the ore is plainly less marked in the more unaltered rocks.

The copper in the unoxidized ores is in the form of chalcopyrite ($CuFeS_2$) and the mineral associates in order of importance are pyrrhotite (Fe_nS_{n+1}), pyrite (FeS_2), quartz and calcite. The quartz and calcite are in minor amounts, but the pyrrhotite and pyrite, especially the former mineral, are found at times in such quantity and so intimately intergrown with the chalcopyrite as to require rejection or close sorting to make

a shipping product.

The surface showings on all the properties consist invariably of iron-stained gossan with copper carbonates and oxides appearing within a short distance of the surface.

Minerals found within the oxidized zone are malachite, azurite, tenorite, cuprite, chrysocolla, native copper, and iron oxides. No chalcocite was definitely determined but black sooty copper oxide was found to contain sulphur so it may be mixed with some sulphide.

The depth of the oxidized zones varies greatly. It is reported that some 10,000 tons of sulphide ore were mined in the south workings of the Queen of Bronze within 30 feet of the surface. Remnants of this ore are still to be seen in the open cut, while in the north workings the oxide zone extends in places to a depth of over 100 feet and has furnished a large tonnage of high grade oxide ore.

Origin of Ore

Several theories have been advanced as to the origin of these copper deposits. That this is an important consideration cannot be overlooked, for once the habit of these ore bodies is satisfactorily established, the basis of a practical development program is available that can be applied to many deposits of similar character in southwestern Oregon.

The more important theories are as follows:

1. Magmatic segregation, that is, the separation and solidifying of the ores within the original molten rock mass.

2. The dissolving by ground waters of copper widely dispersed in the basic country rocks, and redeposition from solution of the copper as sulfide to form high grade ore bodies.

3. Deposition by mineralized waters directly related to the intrusion of igneous rocks within the area.

4. Secondary enrichment by oxidation, leaching and precipitation from a comparatively lean ore body, possibly formed by one of the above named causes.

Discussion of Theories

The presence of primary silica intergrown with sulphides cannot be associated with segregation within a basic magma, and again, the fact that the ores are found in different rock types would seem to argue against the segregation theory.

No evidence can be found to suggest or sustain theory No. 2, that of solution of copper from the country rocks followed by its redeposition.

While it is known that basic rocks may contain copper as an original constituent, such an occurrence has not been proven in the district, and for lack of evidence such a theory must be considered pure speculation.

The third theory, that of mineralized waters arising from deepseated rock intrusions, conforms more closely to present observed conditions. The evidence of this is in the mineral associates and character of ore deposits. Pyrrhotite, especially, is a mineral found in deposits formed

at considerable depths where comparatively high temperature and pressure is the rule.

It is found also in magmatic segregations and contact metamorphic deposits, but rarely if ever in solution deposits at or near the surface.

As has been stated, the original form of the ore bodies is poorly preserved. Only the larger massive ore bodies have withstood the punishment of rock movement. In places less disturbed blocks of the enclosing rock have protected to some extent the inclosed sulphides so that now one may observe stringers and veins of various size running through the country rocks. These veins bear resemblance more to restricted channels than to open fissure structure to be expected in near surface fractional zones.

Again the larger masses of sulphides, with their irregular outlines, suggest replacement of the country rock within the zone, and in this respect conform to deep deposits rather than to near surface deposits. It is thought by some that surface oxidation and leaching with reprecipitation at greater depth has accounted for the larger, richer bodies of chalcopyrite that have furnished the bulk of ore mined.

Chalcopyrite is known to occur as a secondary mineral, but the reactions governing such an occurrence are rather obscure. Granting that such may be the case, we must concede that the pyrrhotite, because of its close relationship, is likewise secondary, which makes the reasoning difficult.

Pyrrhotite is easily attacked by dilute sulphuric acid, which would be formed from oxidation of copper and iron sulphides, yielding abundant hydrogen and sulphide gas, which latter in the presence of copper solutions would precipitate copper sulphide at once. The result would be a shallow superficial zone of enrichment, but would not in this case account for the deeper chalcopyrite-pyrrhotite ores.

CONCLUSIONS ON ORE DEPOSITION

I would describe the succession of events affecting the deposition of the ore as follows:

Processes of mountain building and intrusion of various rocks caused a system of shear zones to be developed across the belt of country now occupied by the Queen of Bronze and neighboring properties.

These shear zones offered a means of circulation and escape for deep-seated mineralized waters coming from heated igneous rock masses. Where the solutions reached points where conditions of temperature, pressure and chemical compositions were favorable, copper and iron sulphides, with some gold and silver, together with silica and lime, were precipitated from solution and deposited in more or less regular form as replacements of the country rock within the zone. Subsequently, perhaps, with the approach of the erosion surface, the rocks containing the ore deposits were altered to serpentine.

The formation of serpentine is essentially a hydration process causing an increase of from 30% to 50% in volume of the rocks involved. This increase in volume caused a general readjustment in position of rock-masses and the ore deposits were broken, dragged and crushed into the softer altered rocks until any original regularity of form was destroyed.

Erosion progressed until finally some of the ore bodies were exposed to surface oxidation and leaching and gossan was formed, which led the prospector to search for ore.

Surface oxidation and leaching progressed somewhat faster than erosion, resulting generally in an enrichment within the oxidized zone, but not affecting the sulphide ores at deeper levels. This is saying that the present erosion surface has no relation to existing sulphide ore bodies and such is my belief. If this theory is correct, we may expect future development within the mineralized belt to disclose ore bodies whose presence is not indicated by surface outcrops.

EXAMINATION OF THE PROPERTY

The individual properties having underground workings and included in the examination are the Queen of Bronze, Cowboy, Lyttle and Mabel. The examination consisted of a geologic study of surface and underground workings, and the sampling and assay of ore in sight. The working maps at the mine office were found sufficiently up-to-date to serve the purpose of the work, so were copied and are presented, herewith as being essentially correct. The sampling and assaying were done entirely by members of the examining party, the assaying being conducted by standard methods in the mine assay office, making use in part of the apparatus there available.

Since it was obviously impossible to measure up and estimate tonnages of positive or probable ore, the sampling was done merely for the purpose of aiding in differentiating between ore of shipping grade and lower grade material. For this reason no assay maps are shown. The added information would in no way compensate for the expense of preparing such a map.

A description of present conditions and mine development of the different properties is as follows:

Queen of Bronze

(See plan maps Queen of Bronze Mine). The north end workings of the Queen of Bronze include over 7,000 feet

of crosscuts and drifts besides several hundred feet of raises, winzes and stopes.

The mined area within these workings is roughly circular in plan, about 450 feet east and west by approximately the same distance north and south. The mine has entry on four principal levels, entering the west slope of the ridge and known as the A, B, C, and D, levels in order of depth with a "blind" intermediate level known as E level, located between A & C levels.

The main raises are No. 1 extending from D to A levels, a vertical distance of 170 feet, and No. 2, extending from D to C levels, a distance of 110 feet.

The stopes representing larger bodies of ore mined out are known as the Johnson, McCauley and East stopes. Minor stopes are the Stevens, Messinger, Stiasy, Ermin and Campbell, besides various smaller unnamed stopes.

Due to caving and system of filling it was usually impossible to determine the outline or extent of the larger stoped areas. They appear, however, to have their greatest dimension on the dip to the south southeast, and to have irregular boundaries. The McCauley stope, which is the largest, extends from C level to above the A level, a vertical distance of about 70 feet, and measures approximately 40 X 50 feet at its widest point.

the bulk of ore thus far mined, but a considerable amount has come from levels between D & C, and good showings are to be seen on the 50 and 70 levels where little development has been done.

The ores on and above A level have been mixed oxides and sulphides, oxidation extending even below this level where porous condition of the rocks has allowed free circulation of air and ground water.

The open cut at a point 90 feet above A level has a surface outline roughly measuring 80 X 120 feet and the oxide ores were mined through a glory hole and chutes from A level.

Ore in sight

First hand inspection is the only satisfactory way of getting an idea of the true conditions of ore in sight in the Queen of Bronze. The mine has been practically stripped of all developed ore of minable grade or quantity. No part of the workings is entirely lacking in copper showings but they consist for the most part of high grade ore remnants about the peripheries of stopes, and of isolated bunches and low grade masses so intermixed with iron sulphides as to preclude sorting to shipping grade.

The best showing of ore to be seen at the time of the examination was in the East stope between C & B levels. Leasers were working at this time on the 4th floor set above the C level. A sample taken across 5 feet of the east face

The B & C levels have limited in vertical extent the bulk of ore thus far mined, but a considerable amount has come from levels between D & C, and good showings are to be seen on the 50 and 70 levels where little development has been done.

The ores on and above A level have been mixed oxides and sulphides, oxidation extending even below this level where porous condition of the rocks has allowed free circulation of air and ground water.

The open cut at a point 90 feet above A level has a surface outline roughly measuring 80 X 120 feet and the oxide ores were mined through a glory hole and chutes from A level.

Ore in sight

First hand inspection is the only satisfactory way of getting an idea of the true conditions of ore in sight in the Queen of Bronze. The mine has been practically stripped of all developed ore of minable grade or quantity. No part of the workings is entirely lacking in copper showings but they consist for the most part of high grade ore remnants about the peripheries of stopes, and of isolated bunches and low grade masses so intermixed with iron sulphides as to preclude sorting to shipping grade.

The best showing of ore to be seen at the time of the examination was in the East stope between C & B levels. Leasers were working at this time on the 4th floor set above the C level. A sample taken across 5 feet of the east face

of the stope assayed 4.8% copper and .04 ounces of gold. Portions of the face were quite siliceous and for a width of two sets the appearance of the ore was that of high grade streaks of chalcopyrite interspersed with bunches of barren serpentine and masses of low grade intermixed chalcopyrite and pyrrhotite.

The material as mined would probably sort about one-half, making a shipping product that would assay between 8% and 9% copper. A sample taken in the west end of the stope on the same floor across a width of 7.8 feet, assayed 4.0% copper and .1 ounce gold.

In the top of the east end of the stope a 6 foot sample gave 4.7% copper and .08 ounces gold, while a 7.5 foot sample directly below on the sill floor assayed 7.0% copper and .06 oz. gold.

The ore is said to continue below the C level and the leasers plan to drive a drift from the Hall crosscut on the 70 level to tap this ore.

The country East of the east stope is wholly unexplored and would seem to offer good possibilities for development in that direction.

No. 2. Raise passes through the 50 and 70 levels and stoping has been done between those levels and above the 70 level. A notable showing of copper is in the south drift on the 50 level where a sample twelve feet in width assayed 5.4% copper and .06 ounces of gold. A "bunch" of ore three feet wide in the top of the stope above 70 level gave 6.4% copper.

Development on the D level is comparatively small and work is necessary to establish the possibilities of this section of the mine. It was the experience of the operators that the ore at this level was less in amount and associated with greater quantities of iron sulphides than on the upper levels, and this belief apparently discouraged any extensive exploration work on the lower level. Large masses of pyrrhotite barren of copper, and low grade pyrrhotite-chalcopyrite material are to be seen about the margins of the stopes, but this condition may be observed on the uppermost levels also.

It may be that D level limits in vertical direction ore deposition within the mineralized zone, but again, the apparent impoverishment may be due to meager development.

It is my judgment that a diamond drill could be used to good advantage on the lower levels in prospecting both in lateral and vertical directions.

The Rynski Raise is driven on an incline from D level to C level. This Raise is in less altered sub-basic rock and excepting a narrow stringer carrying chalcopyrite, showed no ore of consequence. A good example of low grade pyrrhotite chalcopyrite material is to be seen in the 25 foot raise located 35 feet south of No. 2 Raise on D level in the harder footwall rock (auganite). A five foot sample taken in the Raise assayed 1.3% copper.

South Workings

The south workings of the Queen of Bronze are located approximately 1100 feet south of the north workings and were the first developments on the property. The workings consist of (See blue print) about 800 feet of crosscuts on two levels together with an open cut, old stopes and a shaft 109 feet deep. Parts of the workings are inaccessible. No ore of consequence is in sight but showings of copper sulphide oxides, carbonates and sulphate are common in the open cut and about the old stopes. From the open cut some 10,000 tons of sulphide ore of good grade is reported to have been mined close to the surface.

The country rocks are of basic type (peridotite or gabbro) more or less altered to serpentine. The evidence of relationship of ore occurrence to a contact is not plain.

The Cow Boy Mine

The Cow Boy Mine is located about 2 miles south of the Queen of Bronze on Page creek on a west spur of the main ridge.

No maps of this mine were available. The workings consist of two adits which, with drifts, aggregate about 400 feet. The lower tunnel is about 80 feet below the upper one and is approximately 300 feet long. An incline raise from the lower tunnel to the surface is about 120 feet. An open cut is located above the upper tunnel from which oxidized ores have been mined.

The ore occurs in serpentine, and it has been the experience that the ore is generally of higher grade than that found on the other properties.

Ore in sight at the time of the examination consisted of a few tons of good grade chalcopyrite. In the Rose Stope a sample taken across a lens measuring 7 feet by 10 feet gave 15.1% copper, .08 oz. gold, .8 oz. silver.

From 1916 to 1919 this property produced 842 tons of shipping ore. No authentic information was obtained as to the production previous to 1916, but it is said to have been a steady producer when the smelter was in operation.

Mr. Tucker, the present leaser, has two men working in the mine and some ore is being taken out for shipment to Tacoma smelter.

Lyttle and Mabel properties

The workings of the Mabel Mine are located directly on Page creek, while the Lyttle is at a higher elevation on the south slope of the divide between Waldo and Page creeks (See map). The examination of these properties was unsatisfactory, due to caving of the old workings.

The only accessible workings in the Mabel is a crosscut about 80 feet long connecting with a raise about 30 feet high. Copper leaching was observed at the bottom of the raise, but no ore was in evidence. The main workings, said to consist of a shaft 75 feet deep and several hundred feet of crosscuts and drifts, were inaccessible. The workings of the Lyttle consist of a main level, little of which was open for inspection.

Raises connect with a glory hole which is about 100 feet by 40 feet, and 20 feet deep. Approximately 100 feet from the portal of the main tunnel an incline was sunk, said to be 60 feet deep which showed ore. This winze is caved 20 feet below the collar. Copper oxides and carbonates were observed in the open cut and in the crosscut and winze, but no amount of sulphide ore was to be found.

Some distance below these workings a 700 foot crosscut was driven in serpentine and greenstone, but no copper found.

It is claimed that about 1500 tons of ore was produced from the Lyttle Mine under the Tutt management.

PROSPECTS AND LIMITATIONS OF THE PROPERTY

The property as a whole in its present condition can be classed as a promising prospect only, with some attractive features to encourage further exploration.

An estimate of ore in sight of shipping grade must be limited to a few hundred tons, and while several thousand tons of low grade material is in sight the amount is not sufficient to support a concentrating plant of capacity to make the operation profitable. Only large scale operation with a railroad into the district would make treatment of the low grade ore possible. Under present transportation facilities and scale of mine development, the district must remain a producer of sorted shipping ore.

From a geological standpoint I see no reason why the property should not duplicate past production, but an intensive campaign of development is the only means of proving such a future for the mines.

Mining Methods

During the earlier operations of the present company the ore was stoped using stull timbering, but with the opening up of larger ore bodies the soft wet serpentine showed a tendency to swell and cave, so it became necessary to modify the system of mining. Square set timbering with partial waste filling was adopted with success.

Ore is rarely mined to make sorting unnecessary. The practice at the Queen of Bronze has been to rough sort underground after which the material was trammed to outside bins to be drawn off to sorting tables where the low grade was discarded through chutes to the dump and ore of shipping grade dropped to bins discharging into wagons.

The mine waste has been kept separate from the low grade ore dump, and this latter dump has accumulated several thousands tons. The uneven surface of the ground and slumping during wet weather make a close estimate of the dump difficult, but there is perhaps between 5,000 and 6,000 tons that could be sorted out to make a mill feed of about 4% grade. A sample taken over the entire surface of the largest dump at the north workings gave an assay of 2.42% copper and .1 oz. gold. In taking the sample all material that was obviously waste was excluded while material showing sulphides of any nature was included.

Due to the steepness of the road to the mine it is necessary to haul the ore by wagon from the mine bins to the foot of the hill at a cost of 75 cents per ton. It is then reloaded and hauled by wagon and auto trucks to Waters Creek station at a cost of about $6.00 per ton.

COSTS

The following figures were furnished by Mr. Hampshire as showing the cost distribution per ton of ore shipped during the first six months of operation from April to October, 1916.

Wages, supplies and transportation costs have generally increased since that time, so that present costs would be somewhat greater.

Mining		Cost per ton
Supplies and Labor - Queen of Bronze	$8,453.55	
" " " - Cowboy	273.12	
1756.72 tons mine weight cost	$8,726.67	$4.97

Teaming, Auto Truck and Railroad Freight		
Haulage and Freight - Queen of Bronze	$18,808.69	
" " " - Cowboy	832.93	
1756.72 tons mine weight cost	$19,641.62	$11.18

Exploration and Development		
Exploration - Queen of Bronze	$2,382.60	
Development - " " "	5,870.49	
Exploration - Cowboy	267.09	
Development - "	287.32	
1756.72 tons mine weight cost	$8,757.50	$4.98

Improvements, Repairs, General Expense, Salaries, etc.		
Queen of Bronze- - - - - - - - - - - - - -	$6,390.75	
Cowboy- - - - - - - - - - - - - - - - - - -	156.88	
1756.72 tons mine weight cost	$6,547.63	$3.73
	Total cost per ton	$24.86

Total tonnage shipped - - - - -	1756.72
Average price of copper- - - - -	26.46 cents
" copper content - - - - -	9.54%
" gold " per ton - - -	$3.66
Total net return per ton- - - -	$59.84

- - - - - - - - - - - - - - - -

Smelter charges and deductions as follows:

Treatment - - - - - - $2.50 per ton

Less 26 lbs. copper " " (slag loss)

" 3¢ per lb. from market quotation for selling, commission, shipment east, etc.

Equipment, Buildings, etc.

The only buildings of consequence are located at the Queen of Bronze.

Superintendent's house, cook-house, and mine office are located near the south workings, and a blacksmith shop at the north workings.

A well-equipped assay office is located at the foot of the hill near the old smelter.

The buildings were constructed some years ago, but are in good state of repair.

The old smelter has been dismantled except a few pieces that have a certain salvage value.

A small steam engine with compressor and air receiver is located in a room adjoining the mine office. This equipment appears to be in good repair and has been recently used in driving some of the longer raises in the mine. The usual layout of mine cars and tools is at hand.

figure commensurate with unknown factors and risk involved, for with no ore tonnage assured to form a basis for calculation, the value of the property is entirely speculative and hence a purchase price must be left to the judgment and arbitration of seller and buyer.

(Signed) G. E. Stowell, Assistant Mining Engineer,
Oregon Bureau of Mines and Geology.

Queen of Bronze

PRELIMINARY REPORT

on the

Property of

~~THE WALDO SMELTING & MINING CO.~~

~~and~~

TAKILMA SMELTING CO.

JOSEPHINE COUNTY, OREGON

Made for

John Hamshire, et al,
Portland, Oregon

By

Roy H. Clarke,
Peyton Building,
Spokane, Washington.

February 7th, 1916

CHAS. L. TUTT,
PRESIDENT.

E. W. WALTER,
MANAGER.

TAKILMA SMELTING COMPANY

March 6th 1905 Takilma, Oregon, 190

Spillsbury Elevations: Barometric.

Lodge	1700
Smelter Office	1700
Cabo upper tunnel	2800
Little mine	2580
Page creek, surface of water at Smelter Bridge	2000
Queen of Bronze, collar of shaft	2290
Murphy house	2310
Waldo mine, upper dump	2325
Waldo mine, [illegible]	2290

The Waldo Smelting & Mining Company was organized December 3rd, 1901, under the laws of Colorado. Capitalization, $3,000,000.00 in one hundred dollar shares par value, with Spencer Penrose, President, Charles L. Tutt, Vice-President and Secretary, and J. A. Hull, Treasurer.

The Takilma Smelting Company was organized later under the laws of Colorado, with capitalization of $250,000 in ten dollar shares par value, and its officers are: Charles L. Tutt, President; William T. Tutt, Vice-President, and J. A. Hull, Secretary and Treasurer. This company now also controls the Waldo Smelting & Mining Company.

The office of the above companies is Colorado Springs, Colorado.

LOCATION:

The property is located 42 miles Southwest of Grants Pass, in Josephine County, Oregon. At present, Waters Creek, a station on the California & Oregon Railroad, distant 27 miles, is the nearest railroad point.

CLAIMS AND AREA:

The property under option embraces the following, owned by the Waldo Smelting & Mining Company:

Patented mining claims

California, Copper King, Copper Queen, Cowboy, Ben Ball, Lyttle, Oregon, Kelly, Mabel, Morgan, Dorothy and Marion. Containing about 222.168 acres. The above claims are located in the Siskiyou National Forest Reserve.

Also the Strong, Jamison ranches and The Lodge, all in Section 3, Township 41, S. R. 8 W. of the Willamette Merid-

~~ian, containing about 320 acres.~~

Also the following ~~property~~ owned by the Takilma Smelting Company:

(a) The Southwest quarter of the Northwest quarter of Section 25, Township 40, S. R. 8 W., containing about 20 acres.

(b) The Northwest quarter of Section 36, Township 40, S. R. 8 W., known as Queen of Bronze Mine, containing about 160 acres.

(c) The West half of the Northeast quarter and the East half of the Northwest quarter, Township 40, South Range 8 W., together with smelter plant and other buildings and improvements, containing about 160 acres.

The total area is about [illegible] acres.

The relative position of the above properties is shown in red on Map No. 1, herewith. The situation of the district, with reference to Grants Pass and the Coast, is shown on map of the Siskiyou National Forest herewith. —

HISTORY:

The first discoveries of copper ore near Waldo were made in 1860, but no production was made from the district until 1902. The present smelter was blown in, in 1904, and was operated until some time in 1909. The properties have remained idle until last year, 1915, when C. E. Tucker, former superintendent of the Takilma Smelting Company, operated the Queen of Bronze mine under lease.

ADJOINING AND NEARBY PROPERTIES:

There are several properties in the vicinity, including the Waldo Mine, once included in this property, which have made small shipments. Large tonnage has so far not been developed in any properties in the immediate vicinity. At

Happy Camp, on the Klamath River in California, the Gray Eagle property, located about 20 miles southeastward has copper ore reserves, approximating 500,000 tons already developed. The Preston Peak properties located about 16 miles by trail, southward in Siskiyou County, California, are said to have large bodies of copper developed. The outlook for a considerable tonnage of copper ore from this and smaller properties in the surrounding country appears favorable.

FACILITIES:

(a) Transportation

The distance by wagon road to Grants Pass is 42 miles. The grades for the most part are light and permit of hauling heavy loads during summer months. The long haul, however, has heretofore retarded shipments and development of mining properties as well as farm lands. At present, the California & Oregon Coast Railway is building from Grants Pass to Crescent City, by way of Waldo, and is completed fifteen miles to a point called Waters Creek, shortening the haul from the properties to about 27 miles. During the present season, the ore is being hauled from the mine to Grants Pass by auto truck and also by team with profit at $5.50 per ton.

(b) Power

There are no developed water powers in the vicinity, although at some future time hydro electric power on a large scale may be developed on the Klamath river. For present needs power must be developed by steam or gas plants, using wood or distillate for fuel.

(c) Water

Water sufficient for domestic and steam purposes is

available on the property. Water for irrigation, unless already filed on can be obtained from Page Creek, or the East Fork of the Illinois River.

(d) Timber

Timber for present mining and lumber purposes is fairly abundant on the property.

TOPOGRAPHY:

The mining property occupies the Northwest slope of the Klamath mountains, while the ranch lands lie in the flat valley of the East Fork of the Illinois River. The elevation ranges from about 1700 feet above sea level in the valley to elevations of 4000 to 5000 feet at the summits of the mountains to the east. None of the workings exceeds 3000 feet elevation. The East Fork of the Illinois river, drains in a northerly direction and flows into the Rogue river. Depth sufficient to develop the ore deposits can generally be gained by driving adit tunnels.

GEOLOGY:

(a) General

The geological conditions of the Waldo Mining District are complex and similar to those observed in Northern California, in the Klamath river drainage below Happy Camp. The rocks exposed consist of gabbros, greenstones, wide belts of serpentine and other altered basic igneous rocks containing in places remnants of quartzites and other sedimentary rocks. Apparently, this section of country has been the center of adjustment for dynamic stresses exerted at intervals for long periods of time.

Veins and dikes and other structural features formed

in a remote past, as well as the country rock, have alike been subjected to deformation and metamorphism. On this account ore deposits other than placers, of Northern California and Southwestern Oregon, although numerous, are generally shallow, of small tonnage and of irregular occurrence.

(b) Local

Due to the fact that at the time of my examination, the ground was covered with several feet of snow, I was unable to see much of the surface. This condition together with the fact that the workings were caved or full of water at most of the other properties, limited my examination chiefly to the Queen of Bronze Mine.

The formation, however, seems to be similar in the main to all the properties under consideration, being largely gabbro, greenstone and serpentine. The main lines of fissuring extend northward (eastward) and southward (westward) with dip to the ~~east~~ South.

(c) Ore Occurrence

The ore bodies occur in flat zones with longest axis extending northward (eastward) and southward (westward), conforming often in strike and dip to the surface, and following a contact of serpentine with greenstone or gabbro. The ore appears to replace greenstone or other erruptive rock in a shattered or breccia zone. Some of the replacements have made fair-sized stopes of clean sulphide ore, but generally the replacements are of small size necessitating mining considerable waste. This apparent brecciated condition is typical of the unoxidized, deepest portions as well as of the upper portions of the ore deposits.

In my opinion, the deposits represent a normal replacement vein subjected to great dynamic stresses, resulting in

tilting, splitting and breaking up the original deposit into segments and fragments lying unconformable to present main lines of fissuring.

(d) Nature of the Ore

The chief unoxidized ore consists of chalcopyrite with small amounts of pyrite, pyrrhotite, quartz and calcite. It occurs in segments or lenses surrounded generally by serpentine or greenstone, or by heavy iron sulphides containing small copper contents on one side. The heavy sulphides mentioned probably represent the core or lower part of an original replacement vein.

A large part of the ore nearer the surface consists of tenorite, a black oxide of copper, although a considerable possible tonnage of low grade material, containing azurite and malachite is available above and below some of the sulphide lenses. Above the oxidized ore bodies at the surface from ten to forty feet in depth, an iron gossan consisting of porous quartz heavily stained with iron oxide, generally occurs, and its reddish brown and yellow color is a great aid to prospecting.

EQUIPMENT:

Mine

Most of the properties have accomodations in the way of bunk houses and boarding houses, sufficient to accomodate as many men as may be required. At the Queen of Bronze mine, there are two blacksmith shops equipped for sharpening hand steel. At the south workings of the Queen of Bronze, a small building contains a 40 h.p. boiler, receiver and small compressor large enough to run two air drills. The north workings are equipped with blacksmith shop, track and cars, ~~but the tools~~ and thaland tools for mining on hand are

largely worn out.

SMELTER:

On the site for smelter there is a boarding and bunk house, large barn, saw mill,(in bad repair) assay office, ore bins and smelter building. The smelter building contains a matting furnace of 125 tons per day capacity.

1 - 80 h.p. boiler
3 - Engines
1 - Dynamo for electric lights
1 - Blower
1 - Elevator
2 - Pumps
1 - 10" Blake Crusher
1 - Sampler
3 - pr. Large scales
1 - Drill press and many other machine shop tools

The assay office is fully equipped.

The sawmill contains:

1 - 80 h.p. boiler
1 - 40 h. p. engine
1 - Carriage and tracks

DEVELOPMENT:

Queen of Bronze Mine

North Workings

About ~~3500~~ 4500 feet of tunneling has been done at the North Workings as shown on Map No. 2 herewith. Most of the ore has been taken in the past from above the ~~165~~ A foot tunnel (shown in ~~blue~~ brown on map) from a depth less than 90 feet below the surface. A small tonnage came from between the ~~165 foot~~ A tunnel and the ~~150 foot~~ E tunnel. Considerable ore of low grade and a ~~small possible~~ large possible tonnage of shipping grade, is available above and below the ~~150 foot~~ E tunnel. The lower tunnel exposes a small streak of shipping ore in south drift No. 1. This ore may be a part of the same

ore zone B. now being mined in the ~~165 foot~~ A tunnel, whose hanging wall is indicated on Map No. 2 by dotted line F-G. Map No. 3 herewith, is a vertical section showing the possible extension of this ore on lower tunnel, to ore zone ~~X~~ on the ~~165 foot~~ A level. It ~~appears to~~ lies above the ~~130~~ E foot tunnel. It has also been encountered in Riase No. 2, from lower tunnel. The condition of the timbering in this raise made it impossible to see how wide the ore was at the point of intersection, although a rough sample of ore from shoot indicates a fair grade of ore.

The South drift No. 2, Lower Level, shows fragments and lenses of ore some 18 inches wide scattered through a width of 10 or 12 feet for a distance of 90 feet and limited on the west by a fault C-E, extending northward and southward with dip east. This ore may be separate ore body from that found above in the Glory Hole, or it may extend up at flat angle, limited in its depth, passing upward by fissure marked C-D, as a foot wall, since no ore has been found in other parts of the Lower Tunnel, below Glory Hole. The ore here is very much scattered, like plums in a pudding, so that it is impossible to make estimates of tonnage. In my opinion, shipping ore can be profitably mined and sorted from this drift. Some ore has already been taken from drift and raise No. 4, extending 25 feet above drift. A sample of the clean chalcopyrite from the above raise gave a return of 16.5% copper. The iron sulphides here contain about 2.0% copper. It is possible that development may prove that this ore zone extends up towards the surface along line E-C projected at flat angle.

The position of ore now remaining in the North work-

ings is indicated by red lines on map No. 2. Most of the ore shipped during the past year came from Ore Zone B, above the ~~165~~ 165 foot tunnel, and marked "Stopes" on Maps Nos. 2 and 3. Some ore came from Glory Hole at points marked "I" and "J". In addition to ore in Zone B, on ~~165~~ 165 foot Tunnel, shipping ore now remains at point "H", Map No. 2, in Ore Zone A, as well as at several other points in this tunnel. Also in face of Cut Tunnel, see Map No. 2.

Queen of Bronze

South Workings.

About 600 feet of work consisting of tunnels and shafts has been done at a point about 1200 feet south of the North workings, and about 10,000 tons of sulphide ore mined from a cut extending about 30 feet below the surface. A tunnel 50 feet below bottom of cut, cuts a vein 28 inches wide near its face, which assays 7.7% copper and $2.00 gold. The ore, excepting for perhaps [redacted] is exhausted in these workings. ~~There is a possibility of developing more ore by sinking on the vein in face of lower tunnel, although the expense of handling water may make it unprofitable.~~ On the hill to the east, above the south workings, about 400 feet distant, occurs a gossan outcrop, work upon which may develop ore as the indications are favorable.

METHODS AND COSTS:

The ore is mined by hand and costs per ton for the ore shipped last year, amounted to from $4.00 to $5.00 for mining.

The heavy sulphide ore cannot be reduced except by smelting. When sufficient oxide and carbonate ore is developed, the copper may be extracted by leaching. Considerable copper

Insert A

in solution is lost continually from the Queen of Bronze workings in the drainage channels. This could be saved in tanks using iron as a precipitant.

Transportion from the mine to Grants Pass costs ~~$2.00~~ 5.50 per ton. Freight from Grants Pass to Kennett, California, is $2.80 per ton. The charge for smelting at Mammoth Copper mining Company's Smelter is ~~$3.00~~ per ton. The total expense for mining, freight and treatment is $17.50 per ton under present conditions.

PRODUCTION:

The Queen of Bronze mine during the years 1904 to 1909, is credited with a production of 15,000 tons, averaging 8% copper and $3.00 gold, giving a net smelter return of $310,000.00.

During 1915, 1,542 tons of ore was shipped from the above property while under lease to C. E. Tucker which gave a net smelter return of about $55,000.00. 10980 # 5.38

omit

~~The Cowboy, Lyttle and Mabel Groups are reported to have produced 4,000~~ 1500 tons of ore of about the same grade as that of ~~the Queen of Bronze mine.~~

ORE RESERVES:

(a) Tonnage

There are no positive ore reserves to be given for any of the properties seen in this examination.

In the Queen of Bronze property, the probable ore reserves are small and confined largely to remnants left in such shape as to necessitate high costs for extraction. I estimate that there is a probable 300 tons of ore available above the 165 Foot Level in the North workings of which 180

tons are in Ore Zone B, and 120 tons in Ore Zone A (chiefly at point marked H, on map No. 2 herewith).

As to possible tonnage, I believe that there may be a considerable tonnage developed both above the 165 foot Level South of Ore Zone B, and Ore Zone A, which may extend southward to connect with and continue below the ore now observed in the south drifts No. 1 and No. 2, in Lower Level. Exploration and development will have to be done before any estimates can be made of possible tonnage.

(b) Values

The probable ore sampled in Ore Zone B, gave an average return of 10.5% copper and $4.60 in gold. The probable ore sampled in Zone A, gave an average return of 12.2% copper and $5.00 gold, or an average of 11.18% copper and $4.76 gold.

PRICE AND TERMS:

A copy of the agreement of option for purchase of the property designated under caption "Claims and Area" herewith, is attached. The price is $144,000.00 for practically the entire property described, and in fact, all of the property that could be made a basis for a report under existing conditions, being limited to the Queen of Bronze Mine. ~~The price is $144,000.00~~. The terms are $25,000.00 paid on or before March 31st, 1916; $60,000.00 paid twelve months after date of first payment,; and $59,000.00 paid six months after date of second payment.

A royalty of 33-1/3% of the net smelter returns on shipments from the Queen of Bronze mine and 25% of the net smelter returns on shipments from the Cowboy mine to be applied

on the purchase price is exacted by the above agreement under a lease to be granted during the time before final payment is made.

COST OF EQUIPMENT AND DEVELOPMENT:

In case the property can be secured for development on a reasonable basis, it will not be necessary to expend a great deal for equipment other than for new tools, cars and rails, and possibly the installing of the compressor now on the property, or a new one of small size at the Queen of Bronze north workings.

Most of the development and mining can most advantageously be done by hand. I believe a churn drill of small size could be used for surface exploration to good advantage. I believe $25,000.00 should be set aside for development purposes.

VALUE OF THE PROPERTY:

The property based on an examination of the Queen of Bronze mine has value only as a prospect worthy of development. There are no positive ore reserves and only a small probable tonnage in the form of remnants in an area largely stoped, necessitating high mining costs. The situation at the North workings of the Queen of Bronze indicates that the possibility of finding more similar lenses of ore near the surface are not entirely exhausted.

The gossan showing to the east of the south workings indicates some further possibilities of finding ore at that point. While I have not seen the other properties embraced

in this option, from descriptions of others I am satisfied that the Cowboy property justifies the expense of further exploitation. The other mining claims also may have possibilities in that they have produced small quantities of ore in the past and lie within the copper bearing zone.

The ranch lands embrace the level river valley, can be irrigated, are free of brush and timber, but on account of scanty and rather unproductive soil, do not add very materially to the value of the property.

CONCLUSION:

I do not recommend the purchase of the property on the terms of payment and royalty of the option contained herewith.

Although the price asked is large compared to the present showing, and the possible ore to be developed, if the property could be obtained under terms allowing six months' time for exploration before making cash payment, with the lease modified so that ore could be shipped either paying 10% royalty on the net smelter returns on same, or possibly paying a royalty graduated according to the market value of copper. I would recommend the purchase of the property.

Yours respectfully,

LIST OF ASSAYS QUEEN OF BRONZE MINE

Sample Number	Width sampled Inches	Copper Percent	Gold Ozs.	Gold Value
1	4	6.9	0.06	$ 1.20
2	12	1.5	0.03	.60
3	12	8.9	0.30	6.00
4	12	14.0	0.34	7.02
5	in chute	5.9	0.10	2.00
6	18	16.5	0.08	1.60
7	48	3.6	0.16	3.20
8	30	23.9	0.24	4.80
9	24	6.0	0.10	2.00
10	48	5.9	0.11	2.20
11	60	7.7	0.18	3.60
12	60	14.2	0.34	6.80
13	72	13.7	0.31	6.20
14	84	1.5	no trace	
15	10	11.4	0.09	1.80
16	48	10.4	0.24	4.80
17	54	16.8	0.51	10.20
18	24	1.1	0.41	8.20
19	60	8.4	0.06	1.20

REPORT ON THE PROPERTY

of the

QUEEN OF BRONZE MINING CO.,

TAKILMA, JOSEPHINE COUNTY,

OREGON.

Made for
Pohlman Brothers,
Spokane, Wash.

By

F. Cushing Moore,
Spokane, Wash.

December, 1929.

F. Cushing Moore
Mining Engineer
Spokane, Washington

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F. Cushing Moore
Mining Engineer
Spokane, Washington

REPORT ON THE PROPERTY

of the
QUEEN OF BRONZE MINING CO.,

Waldo District, Josephine County, Oregon.

By F. Cushing Moore,
Spokane, Wash.
December 1929.

I. Location, Transportation, Topography, Climatic Conditions. See Map No. 2.

A. Location & Topography:
The property is located on the west slope of a prominent ridge between Althouse Creek and East Branch of the Illinois River, about one mile southerly from Takilma, 40 miles southwest of Grants Pass, Josephine County, Oregon, and 50 miles northeast of Crescent City on the California Coast. Elevation at Takilma is 1567 ft. above sea level,
" " Queen of Bronze 2200 to 2450 do.
" " Cowboy, 2600 to 2800 do.
" " Waldo, 2250 to 2400 do.

B. Climatic Conditions:
The climate is very mild; snow never remains for any great length of time, and the average rainfall is about 40 is about 40 inches.

C. Roads, Railroads, Transportation:
The Redwood Highway passes within about six miles of Takilma, and a good road is available from the mine to the Highway. At the Cowboy workings it is necessary to haul the ore on sleds to the lower tunnel, which is in process of construction. This entails a cost of 50¢ per ton, which will be eliminated when the lower tunnel is connected with the upper workings. Truck hauling from the mine to Water Creek railway station, 27 miles distant, is contracted for at $2.75 per ton.

II. Extent of Property, Area, Titles. See Map No. 1.

The total area of the patented ground owned by the Queen of Bronze MiningCo., is 882.2 acres, comprising an irregular-shaped block extending over three miles north and south, by two miles east and west. Within this block of ground is located the property of theWaldo Copper Company, comprising 480 acres of patented ground, which is held under lease and bond by the Queen of Bronze Mining Co., at a total purchase price of $30,000 with very favorable terms.

In addition to the patented ground, mining claims have been located to the extent of 220 acres, giving a total area held by the company of 1582.2 acres. The county records have recently been searched by Roy H. Clarke, and title is clear as above stated.

III. History:

The district was first discovered in 1860, but practically no development was undertaken until 1903. A 100-ton smelter was built on the premises under the Tutt ownership in 1904, and operated for five years, handling 20,000 tons of 8% Copper ore from the Queen of Bronze, besides considerable from the Cowboy, Lyttle, Mabel, and other properties. This ore was reduced to matte and hauled over 40 miles of rough road to Grants Pass. Coke costs $20,00 per ton delivered, and other supplies in proportion.

From 1910 to 1916, theproperty was handled by leasors who shipped a few thousand tons of ore to the Tacoma Smelter. In 1916 the property was taken over by the Twohy interests, under the management of Roy H. Clarke, who operated it until the drop in the price of Copper, at the close of 1918. During these three years, 9,992.3 tons were produced from the Queen of Bronze, having a new smeltery value of $432,440.88, and 1644.46 tons from the Cowboy, with a net smeltery value of $73,562.91.

During the following few years, leasers made intermittent shipments from the property.

In 1928 and the spring of 1929, John Hampshire of Grants Pass, Oregon, operated the property under lease, producing up to the time of the incorporation of the Queen of Bronze Mining Co., (May 31, 1929) about $39,994.20 net smeltery value. Shipments from June 1st to November 1, 1929, netted $23,589.30, giving a total for 1928 and 1929 of $63,994.20.

The gross value of all ore shipped to date from the Queen of Bronze and other properties in the Queen of Bronze ore zone is approximately $1,500,000 with net operating profit of about $500,000.

IV. Economic Value of Ore:

A. Valuable Metals in Ore:
The metals of commercial value occurring in the ore given in the order of their importance, are: - Copper, Gold and Silver.

B. Stability of Metal Prices:
Copper averaged about 15¢ per lb. for many years, and, during the War, it was much higher, but up to this spring has been lower. In April of this year, it advanced to 24¢ per lb., and now is quoted at 18¢. I feel that it is safe to assume that the price of Copper will stay around 18¢ per lb., and all estimates of value in this report are based on this price.

Gold is paid for by the smelters at $20,00 per ounce; Silver is worth about 50¢ per ounce at present, and is likely to remain about that.

C. Character of Ore; Amenability to Reduction; Values:

1. Ore Minerals:
The principal ore mineral is Chalcopyrite, and intimately associated with it is found Pyrrhotite and Pyrite, accompanied by some quartz and Calcite. In the upper areas, the Chalcopyrite has oxidized to Malachite, Azurite, Cuprite and Chalcocite. Native Copper is found occasionally.

2. Sampling:
I made no attempt to do any hand sampling, but from samples taken by Roy H. Clarke, G. E. Stowell and Byron Wilson, and from the mining andshipment records to date, it is safe to estimate that the ore, as sorted for shipment, will average about 10% Copper and .06 ounces in Gold, per ton. The Silver is below pay value. Recent shipments from the Cowboy have run as high as 14% Copper. The last car of 54 tons netted $1868.49.

3. Sorting:
As operations are being conducted at the present time, one ton of hand-sorted shipping product is the resultant of about 5 tons of ore removed from the mine, and some waste is sorted out under ground and put back as filling. Metallurgical tests made on the mine run ore would indicate that probably after the construction of the mill, sorting will be confined to the elimination of waste.

4. Milling:
Milling tests made by the Federal Mining & Smelting Co. on samples taken from various parts of the mine and dump indicated that a saving of 95% of the Copper and 85% of the Gold could be made when a concentrate of 8-1/2% Copper content was produced. This, of course, was a high extraction, but the grade of the concentrate was too low. Recently other tests made by the Bunker Hill & Sullivan M. & Co. Co., and the General Engineering Co., of Salt Lake, indicate that a much higher grade of concentrate is possible, but the extraction is slightly lower. The results of these tests are as follows:

Bunker Hill & Sullivan M. & C. Co.,- Heads, Copper 5.2%; Gold .0325 oz. Copper recovery 92%; Gold recovery 75%. Concentrate produced Copper 18% carrying .10 oz Gold per ton.

General Engineering Co., Salt Lake,- Heads, Copper 5.345%; Gold .025 oz. Copper recovery 92%; Gold recovery 75%. Concentrate produced, Copper 18% carrying .12 oz. Gold per ton.

The second test just being made by the Bunker Hill & Sullivan on a different sample, indicates that probably the most efficient grade of concentrate to be produced by the mill will contain at least 20% Copper.

5. Smelting:
The most favorable smelter schedules under which shipments can be made are those offered by the American Smelting & Refining Co. plant at Tacoma, which are as follows:

Crude Ore Schedule:
Pay for 100% of the Copper at the market price, less 2¢ per lb. where Copper is under 14-1/4¢ per lb., and 2-1/2¢ per lb. when Copper is selling for over 14-1/4¢ per lb. Deduct 10 lbs. Copper when Copper is under 14-1/4¢ per lbs., and 15 lbs. when over 14-1/4¢ per lb. Treatment $2.50 per ton.

Pay for 95% of Gold at $20.00 per oz.
Pay for 95% of Silver at New York market.

Flotation Concentrate Schedule:
Pay for 100% of Copper, less 2-3/4¢ per lb.
Deduct 20 lbs. Copper from assay.
Pay for 95% Gold at $20.00 per oz.
Pay for 95% Silver over half an oz.
Base charge for treatment, $2.75 per ton, for concentrates containing 12% Copper or less. Add to base charge 10¢ per unit over 12% up to a maximum additional charge of $2.25 a ton. Add to base charge 25¢ for each 1¢ increase in Copper quotation over 17¢.

Freight from Waters Creek to Tacoma-

Under $20.00	valuation	$4.41	per	ton
$20. to $30.	"	$4.75	"	"
$30. to $40.	"	$5.10	"	"
$40. to $50.	"	$5.45	"	"
Over $50.00	"	$5.80	"	" (Assumed)

D. Power:
No water power has been developed, although I am informed that power can be had on the Klamath River. For the present, gasoline engine driven compressors are used. Steam power is too expensive and therefore out of the question.

When the mill is constructed, either water power will have to be developed, or electric power brought in, or Diesel engine power used. The latter is probably the most feasible, as the initial cost will be very much less, and the fuel oil can be handled in drums by the trucks as a back-haul from Waters Creek Station. To minimize the expense of handling the oil, the power plant should be located conveniently to the loading bins at the mill, and a storage tank provided at Waters Creek Station.

The cost of power generated by Diesel engines compares vary favorably with electric power, where fuel oil can be delivered cheaply. Gasoline is being delivered in tank trucks by the oil companies at Takilma and Grants Pass from ship cargoes landed at Crescent City, California at a very reasonable figure, but I see no reason why fuel oil cannot be delivered via Crescent City for less money than via Waters Creek Station. In any event fuel oil can be delivered at Waters Creek Station at 5-1/2¢ per gallon and contracted for as a back-haul on the hauling of concentrate shipments for not to exceed 1¢ per gallon. This would furnish fuel oil at the plant for 6-1/2¢ a gallon.

This would mean a fuel cost of 7¢ per hp. hour including labor. This cost compares very favorably with that of electric power at the smaller properties in the Coeur d'Alene Mining District.

Estimate of cost of operation of a 275 H.P. Diesel-Electric power plant:

Fuel oil consumption @ .42 lbs. per h.p. hour - 15 gals, 15 gals. @ 6-1/2¢ - 97¢ per hour, or about 4/10¢ per h. p. hour.	
Labor, 3 men @ 5.00 per day,	$15.00
Lubricating oils, & incidentals	2.50
Fuel oil, 15 gals per hour for an average of 20 hours per day @ 6-1/2¢	19.50
Total cost per day for275 h.p.	$37.00
Total cost per h.p. hour - 7/10¢	
Cost per h.p. year - $48.50	
Cost per ton mined on 100-ton-per-day basis - 37¢	

E. Ore Extraction - Mining & Milling:

1. Mining:

Ample labor for mining purposes has been available at less wages than prevail in most of the larger mining camps.

Working conditions have been ideal for economic operation, as all the miners have lived in the vicinity of Takilma and go back and forth in privately owned autos. When a large crew is required it may possible be necessary to operate a company-owned boarding and bunk house adjacent to the mine workings.

Early mining was done by hand, but recently drilling underground has been accomplished by two portable gasoline-driven compressors, one each at the Queen of Bronze and the Cowboy workings.

Ample timber for mining purposes for many years to come is available on the premises. Water for milling and domestic purposes is also in abundance.

Supplies are procurable at Grants Pass and there is a small store at Takilma.

Practically all of the ore will be taken from tunnel workings. Most of the ore thus far mined has come from the Queen of Bronze workings, although the Cowboy is now a material producer.

Extraction cost for hand-sorted ore has been quite definitely established during the past year at $15.75 per ton, made up as follows:

Mining	$10.00 per ton
Development	3.00 " "
Hauling to Waters Creek	2.75 " "
Total extraction cost,	$15.75 per ton

This figure is based upon one ton of shipping product averaging 10% Copper and .06 oz. Gold per ton. It includes all underground costs, surface sorting, and transportation to the railroad. The cost for each ton as taken from the mine averages $2.60 per ton.

Under the present method of sorting, for every ton of shipping product, four tons of low-grade ore are put on the dump. When a mill is available, this sorting will be done on a belt, and the rejects go directly to the mill; thus materially reducing the cost per ton of the shipping product, as well as converting the rejects from the sorting into mill feed, with no additional mining expense.

As nearly as can be determined from samples taken and records of operations to date, the mine ore should average 6% Copper, and, after having one-fifth sorted out as high-grade (10% Copper), the remaining mill feed would average 5% Copper, i.e.-

5 tons Mine ore averaging 6% Copper equals (1 ton hand-sorted ore (10% Copper)
(4 tons mill feed (avg. 5% Copper = (1 ton conc. (18% Copper)
(3 tons tails to waste.)

2. Milling:

It will be advisable to centralize the equipment by locating the mill and power plant at some convenient point, and transport the ore to the mill by short tramways. The necessity for this is apparent, as ore will be mined from a number of points along the course of the ore zone, which extends over two miles. The topography of the country lends itself admirably to this arrangement. A mill site has been selected immediately west of the west end of the Annie Claim on patented ground. See Maps No. 1 and 14.

A surface tram, constructed from the "D" tunnel of the Queen of Bronze to the proposed mill site would be 1665 feet long. Map No. 17 shows a profile of the tram line.

Estimate on Operating Cost of Mill, 100 tons of Mine Ore Per Day, Straight Concentration - No Sorting.

	Per Day	Per Ton
2 men in mill per shift @ 5.00 each	$30.00	.30
1 man in sorting plant sorting waste	5.00	.05
Reagents per ton - 15¢		
Power - 40¢		
Supplies - 25¢		.80
Total cost per ton		$1.15

In all computations following, I have used the figure, $1.50 per ton, to be on the safe side, but in actual practice it will be probably be nearer the above estimate of $1.15 per ton, as the indicated flow sheet is extremely simple and the power cost low.

3. Calculations for Net Value of the Ore.
Calculations based on the present grade of shipments taken in combination with the concentrate to be produced by the mill and upon concentrates of various grades exclusive of any hand-sorted product are as follows:

(a) A combination of hand-sorted and milling ore where the hand-sorted product taken from the picking belt prior to milling will be of the average grade of shipments from the mine during the past year, and the concentrate produced from the rejects from this sorting will be equal to the results obtained from the recent metallurgical tests on such ore:

Mine run ore - Copper 6%.
Hand-sorted product (one-fifth of mine run Ore) Copper 10%; Gold, .06 oz.
Copper concentrate produced from rejects, Copper 18%; Gold $2.00, per ton.

Hand-sorted ore, Copper 10%; Gold .06 oz. per ton.
Copper @ 18¢ less 2-1/2¢ = 15-1/2¢ net per lb.
Copper 200 lbs. less 15 lbs. @ 15-1/2¢ = $28.67
Gold .06 oz. x 95% @ $20.00 per oz. = 1.14
Pay for copper and gold $29.81

Freight	4.75	
Treatment	2.50	
Net smelter return (hand-sorted ore)		7.25
		$22.56 per ton

Mining 5 tons -	10.00	
Development -	3.00	
Truck haul -	2.75	15.75

Net operating profit on each ton of hand-sorted ore . . . . . $ 6.81

As the milling ore is the reject from the sorting of the crude ore, all expenses have been deducted except milling expense and the transportation of the concentrates. Therefore we have four-fifths of the original mine run of ore as mill feed.

Assumed flotation concentrate, Copper 18%, Gold .11 oz. per ton.
Copper 18¢, less 2-3/4¢ = 15-1/4¢ per lb.
Copper 18%; 360 lbs. less 20 lbs. = 340 lbs.
@ 15-1/4¢ $51.85
Gold .11 oz. x 95% @ $20.00 ..... 2.00
Silver 1 oz. less 1/2 oz. = 1/2 oz. @ 50¢ . .25
Pay for Copper, Gold, Silver, per ton . . . $54.10

Hauling	- 2.75	
Freight to smelter	- 5.80	
Treatment Charge	- 3.60	
Milling 4 tons	- 6.00	18.15
Net operating profit on concentrate		35.95
Net operating profit on hand-sorted ore		6.81
Total operating profit on 2 tons ore		42.76 per ton

On one ton each of crude ore and concentrate produced from 5 tons of mine run.

Combined profit per ton of ore as mined . . . . $ 8.55

(b) Concentration without sorting of Crude Ore. Copper 18% concentrate carrying $2.00 in Gold. Mill feed, Copper 6% - Recovery 92%. Copper 18¢. Ratio of concentration - 3.26 to 1.

Copper 18% - 360 lbs. less 20 lbs. @ 15-1/4¢		$51.85
Gold		2.00
Silver		.25
		54.10
Hauling to RR	- 2.75	
Freight	- 5.80	
Treatment	- 3.60	
Milling @ 1.50	- 4.90	
Mining @ 2.50	- 8.15	25.20
Net value of 1 ton of concentrates		28.90
Net operating profit 1 ton mine ore		8.87

(c) Concentration without sorting - 6% Copper mill feed. Copper 20% concentrate, recovery 90%, Gold $2.25. Ratio of concentrate 3.7 to 1.

Copper 20% - 400 lbs. less 20 lbs. 380 lbs. @ 15-1/4¢		$57.95
Gold $2.50, Silver 30¢		2.55
		$60.50
Hauling	- 2.75	
Freight	- 5.80	
Treatment	- 3.80	
Mining @ 2.50	- 9.25	
Milling @ 1.50	- 5.50	27.10
		33.40
Net operating profit 1 ton mine ore		9.00

(d) Concentration without sorting - 6% Copper feed. Copper 20% concentrate, recovery 92%, Gold $2.25. Ratio of concentrate 3.62 to 1.

Copper 20% - 400 lbs. less 20 lbs. 380 lbs. @ 15-1/4¢		$57.95
Gold $2.50, Silver 30¢		2.55
		60.50
Hauling	- 2.75	
Freight	- 5.80	
Treatment	- 3.80	
Mining @ 2.50	- 9.05	
Milling @ 1.50	- 5.44	26.84
		33.66
Net operating profit 1 ton mine ore		9.30

(e) Tabulations of Net Values Per Ton of Mine Ore, Based upon a Concentrate Carrying 18% Copper and $2.00 in Gold Per Ton. Range of ore values 3% to 6% Copper. Range of Copper prices 0 to 25¢ per lb. Overhead expense 50¢ per ton.

Mine Run 6% Copper:

Price		Operating Profit		Net Profit
Copper price 22¢ per lb.	– $12.35	less 50¢ overhead		$11.85
" " 20¢ " "	– 10.70	" " "		10.20
" " 18¢ " "	– 8.87	" " "		8.37
" " 15¢ " "	– 5.43	" " "		4.93
" " 12¢ " "	– 2.61	" " "		2.11

Mine Run 5% Copper:

Copper price 22¢ per lb.	– $10.12	less 50¢ overhead		$ 9.62
" " 20¢ " "	– 8.30	" " "		7.80
" " 18¢ " "	– 6.67	" " "		6.17
" " 15¢ " "	– 4.09	" " "		3.59
" " 12¢ " "	– 1.77	" " "		1.52

Mine Run 4% Copper:

Copper price 22¢ per lb.	– $ 6.90	less 50¢ overhead		$ 6.40
" " 20¢ " "	– 5.82	" " "		5.32
" " 18¢ " "	– 4.53	" " "		4.03
" " 15¢ " "	– 2.29	" " "		1.79
" " 12¢ " "	– 0.41	" " "	Loss–	.09

Mine Run 3% Copper:

Copper price 22¢ per lb.	– $ 4.18	less 50¢ overhead		$ 3.68
" " 20¢ " "	– 3.36	" " "		2.86
" " 18¢ " "	– 2.40	" " "		1.90
" " 15¢ " "	– .72	" " "		.22
" " 12¢ " "	– Less–.54	" " "	Loss–	1.04

See Graph on next page.

(f) Record of Shipments of Queen of Bronze Mining Co. Between August, 1928, and November 1st, 1929

No. Cars	54
Net Tons Shipped	2798.86
Average Copper content	10.29%
Average Settlement price	14.42
Average Gross value Copper per ton	28.98
Average Gross value Gold per ton	1.18
Total net value shipments after all deductions, freight and smelter	63583.50
Average Net value per ton	23.72

F. Overhead Expense:

Up to the present time, the overhead expense on the property has consisted of a manager, drawing $300 a month, and an office girl, with intermittent engineering expense and traveling expense for officers of the Company. From now on, the overhead expense will be considerably greater, and will include the salaries of a manager, an assayer-engineer, and a bookkeeper at the property. This with traveling expense of officials, and other incidentals will probably amount to $1,000.00 a month. For the sake of conservatism, I am placing the overhead at $1,500.00 per month, or 50¢ a ton on a 100-tons-per-day basis.

V. Extent and Permanency of Ore Deposits:

A. Geology: Ore Genesis:

At the Cowboy, Lyttle and Mabel Mines, the ore bodies are found in ill-defined mineralized zones in or associated with the serpentine, closely related to the diorite or andesite. The ore occurs in irregular bodies and fragments varying in size from several thousand tons to a few pounds, showing broken angular shapes and slickensided surfaces, crushed into the softer enclosing rock with no apparent order or connection. But at the Queen of Bronze the ore has been less disturbed and less serpentine is in evidence.

Quoting from a report by G. E. Stowell, Mining Engineer with the Oregon Bureau of Mines & Geology, who investigated the property several years ago when many of the upper works of the mine were accessible, which have since been closed by caving: "Briefly the rocks of the district include both sedimentary and igneous types. The sedimentaries are represented by older slatey rocks, quartzites and limestones, and more recent gravels and sands. Associated with these are the igneous intrusives, practically all being classified as andesite, peridotite, augite, gabbro, diorite, greenstone and serpentine, etc.

"Doubtless the alteration of such rocks as peridotite, pyroxenite and gabbro have produced the serpentines so common in the region, while alteration of the diorite and andesite, etc., accounts for the greenstones. The Copper deposits are associated with one or more of the above named igneous rocks, and so far as is known, have no direct relation to the sedimentaries other than the ore deposits are found in a belt of country extending east of north in the direction which conforms roughly with the strike of the sedimentary rocks in the vicinity.

"The most common rock associated with the ore is serpentine, although the ore is not confined to this rock alone, but is found in adjacent rocks less altered and less basic in character."

In all geological publications, the theory of descending solution with resultant shallow deposits, has been advanced, but recent development has established the fact that the ores do extend to considerable depth. The theory that mineralized waters arising from deep-seated rock intrusions are responsible for ore deposits conforms more closely to present observed conditions. The evidence is in the mineral associations and character of the ore deposit. Pyrrhotite especially

is a mineral found in deposits formed at considerable depth where comparatively high temperature and pressure was the rule. It is found also in magmatic segregations and contact metamorphic deposits, but rarely, if ever, in solution deposits at or near the surface.

As stated, at the Cowboy Mine and in one of the stopes at the Queen of Bronze Mine, the original form of the ore bodies is poorly preserved, and only the larger masses have withstood the crushing of the rock movements. In places less disturbed blocks of ground have protected to some extent the enclosed sulfids, so that one may observe stringers and veins of various sizes running through the country rock.

Briefly stated, intrusions of various rocks caused a system of shear zones to be developed across the country. These shear zones offered a means of circulation for the deep-seated mineralized waters coming from heated igneous masses. When these solutions reached points where conditions of temperature, pressure and chemical composition were favorable, the sulfids were deposited along with the silica and lime. Subsequently some of the rocks were altered to serpentine. In the formation of serpentine, an increase of from 30% to 50% of volume of the rock is involved. This increase in volume was sufficient to account for the movement indicated in the vicinity of the ore bodies, and could easily have accounted for the crushed and dragged appearance of the ore.

Erosion has progressed to a point that some of the ore bodies have been exposed on the surface with their consequent development of gossany outcrop. As surface oxidation has progressed more rapidly than the erosion, we find the oxidized copper minerals in the upper horizon of the ore bodies.

All of the ore thus far mined has been taken from comparatively small areas, that at the Queen of Bronze being the most extensive. At this point, the ore bodies are contained within an area of approximately 400 feet in diameter, although other ore bodies have been mined in the same property at a little greater distance.

The Cowboy workings cover a very small area, and those in the Lyttle and Mabel even less.

Undoubtedly many other ore bodies will be found on theproperty, as the territory is not only geographically well located, but iron stained float and gossan material are exposed at various points along the line of the other ore deposits.

B. Ore Reserves:
Owing to the irregular character of the ore bodies and the deficiency of development work, it is impossible to technically calculate much ore, but from the past history of the property it is safe to assume considerable reserves, although not conclusively proven.

Cowboy: Map No. 13: The Cowboy has been a very good material producer during the past year. The last carload of ore shipped during the month of November ran over 14% Copper. The ore shoot has an approximate length of 150 feet on the Intermediate Tunnel level, and a width of about 10 feet. On the No. 2 Tunnel Level, the ore as developed is not quite so long, but possibly the northern extension of the ore has not yet been encountered.

The southern extension of the ore is in question, but as the drift encountered heavily serpentinized rock and the identity of the vein destroyed, possibly the ore has been displaced to the east. Recently a new showing of oxidized ore, carrying some high values in Chalcocite with native copper has been opened east of the old workings. This exposure is shown on Map No. 13. Tunnel No. 4 has undercut the old shaft at about 26 feet and some high grade ore is being sacked for shipment. This ore appears to follow the old fault, as some chunks of unaltered sulfid ore were found in the fissure. The development of this showing to the east may possibly lead to the location of the southern extension of the Cowboy ore body.

I have estimated roughly 8000 tons of positive ore; 8000 tons of probable ore, and 8000 tons of possible ore of average grade of 6% Copper in the Cowboy.

Lyttle: Map No. 12: The Lyttle Mine produced considerable high grade ore years ago but the workings have been caved until recently. A raise is being driven from No. 2 Tunnel to tap the bottom of the incline shaft from No. 1, where, according to best authority, considerable ore was left by former operators.

I have estimated 5000 tons of probable ore and 5000 tons of possible ore of 6% Copper grade for the Lyttle.

Queen of Bronze; Maps Nos. 3 to 11, inc.

Stevens Ore Body: The Stevens ore body was mined from the "E" level through to the surface. These workings are all closed, but from information obtained from former operators and from a study of the maps, it is probable that this ore body should be encountered on "C" Tunnel level a short distance west of the top of the Rynski raise. Again this ore body should be found soth and west of the southwest drift of the Hampshire workings on "D" Tunnel level.

I have estimated 1000 tons of probable ore and 6000 tons of possible ore of average grade of 6% Copper for the Stevens ore body.

Messenger-Twohy Ore Body: The Messenger ore body above "C" Tunnel level broke into the old glory hole workings and below that level is represented by the Twohy ore body partially stoped between the "D" and Leach levels. This ore body connects with the McCauley ore body above "C" level, so that more than likely it will connect with the Leach ore body, which is the downward continuation of the McCauley ore body on the Leach level.

Driving of the crosscut at transit point 96 on the Leach level should cut this ore.

On "D" level, the Twohy ore body was developed by drift east and west for a distance of about 50 feet, but the drift at the east end turned to the left away from the ore. This face should be advanced, as the Twohy-Messinger ore should be found to the south and east. The ore body should be approximately 100 feet long, ten feet wide, with 40 feet of backs between the Leach and "C" levels, giving 2200 tons of positive ore, 2200 tons of probable ore and between the "D" and Leach levels 7800 tons of possible ore.

McCauley-Leach Ore Body: The Leach stope on the Leach level has the downward continuation of the McCauley ore body, which was the greatest producer of any of the Queen of Bronze ore bodies. This ore body was followed through practically to the surface, is about 100 feet long, 10 feet wide, and has above the Leach level 2200 tons of positive ore, between the "D" and Leach levels 7800 tons of probable ore, and below the "D" Tunnel level 11,000 tons of possible ore.

Cameron Ore Body: Above the Cameron stope at the east end of "C" level is a block of virgin ground with the exception of the tunnel workings on the extreme east end of "A" level. This ore body is at least 50 feet long by 20 feet wide, and should extend to the surface.

I have estimated 5500 tons of positive ore, 5500 tons of probable ore, and between "D" and "C" levels 12,000 tons of possible ore.

Pohlman Ore Body: The Pohlman ore body is that ore which has recently been encountered in the top of No. 7 raise above the Pohlman crosscut on the Leach level. This ore is undeveloped, in virgin ground, and may either connect with the Cameron ore body or may be an entirely independent deposit. I have estimated 2800 tons of probable ore and 2800 tons of possible ore.

Hampshire Ore Body: The Hampshire ore body was discovered on the "D" level and has been encountered by no other workings. The ore body is 100 feet long by about 15 feet wide, has been mined at the west end about 40 feet high but practically no stoping has been done at the east end. This ore should extend through to the surface and to depth. I have estimated for this deposit 8300 tons of positive ore, 16,600 tons of probable ore and 20,000 tons of possible ore.

South Workings: Former operators removed considerable ore from the South Workings of the Queen of Bronze, and a large glory hole is in evidence, but as the workings beneath this glory hole are inaccessible, and as I have no authentic data on them, I am making no inclusion of reserves in my estimates.

Dumps: At the Queen of Bronze workings there are large dumps of ore which have accumulated from the shipping of high grade ore during the past years. From estimate of volume and trenching it has been determined that there are from 25,000 to 40,000 tons of ore in these dumps which will average approximately 3% Copper. The copper sulfid has oxidized to some extent in these dumps, but flotation tests have shown that very satisfactory extraction can be made. Possibly it might pay to leach them prior to flotation concentration.

RECAPITULATION OF ORE RESERVES.

	Tons of Ore Reserve		
	Positive	Probable	Possible
Cowboy Mine	8000	8000	8000
Lyttle Mine	-	5000	5000
Queen of Bronze:			
Stevens, above "D" level	-	1000	6000
Twohy-Messenger, above "D" level	2200	2200	7800
McCauley-Leach " " "	2200	7800	-
" " below "D" level	-	-	11000
Cameron, above Cameron Stope	5500	5500	-
" above "D" level	-	-	12000
Pohlman, above "D" level	-	2800	2800
Hampshire " "D" level	8300	16600	-
" below "D" level	-	-	20000
	26,200	48,900	72,600
Ore in dumps, 3% Copper	25,000	15,000	
	51,200	63,900	72,600
Positive ore	51,200 tons		
Positive & Probable ore		115,100 tons	
All estimated ore reserves			187,000 tons

Positive			
In mine	26,200 tons @ $8.00	$209,600.00	
Dumps	25,000 tons @ $4.00	100,000.00	
	Positive ore value		$309,600.00
Probable			
In Mine	48,900 tons @ $8.00	391,200.00	
Dumps	15,000 tons @ $4.00	60,000.00	
	Probable ore value		451,200.00
	Positive and Probable-		760,800.00
Possible			
In Mine	72,600 tons @ $8.00	580,800.00	580,800.00
	T o t a l		$1,341,600.00

This estimate includes only known ore bodies, and does not consider the prospective value of ore bodies partially indicated by surface manifestations.

In ore reserve calculations, a ton of ore is figured as nine cubic feet of ore in place.

VI. Summary, Conclusions, Deductions and Recommendations:

A. Ore Deposits: The rocks in the immediate vicinity of the Queen of Bronze ore bodies are igneous intrusions or flows, principally andesite, gabbro, diorite, etc., altered in places to serpentine and greenstone. The ore bodies were undoubtedly deposited by ascending mineralized solutions related to the intrusion of igneous rocks at great depth.

Subsequent to the ore deposition, erosion lowered the surface to the proximity of the ore deposits. Oxidization has been more rapid than erosion, causing oxidized zones above the sulfid ore bodies. Rock movement along shear zones has caused alteration of the basic rocks and alteration of the basic rocks to serpentine may have been responsible for the movement. In any event, some of the ore bodies are much disturbed and the ore in places occurs as boulders surrounded by a serpentine coating. The ore has the appearance of being squeezed rather than crushed or rolled by movement.

The ore is a chalcopyrite associated with pyrrhotite and pyrite. In the oxidized zone are found asurite, malachite, cuprite, chalcocite and native copper. Gold andSilver are associated with the Copper. Average milling ore will carry approximately 6% Copper and .06 oz. Gold, and .30 oz. Silver per ton. The Gold values are very erratic, but the average of all shipments made to date ran $3.07 per ton in a hand-sorted ore averaging 10.84% Copper,-exclusive of early shipments, of which we have no detailed record.

The ore bodies thus far developed are within areas of no great dimensions, but the mineralized belt extends through the property in a northerly and southerly direction for over two miles. Undoubtedly the Queen of Bronze ore zone is in the northern extension of the mineralized copper area of Northern California.

From the nature of the deposit and associated minerals, it is safe to assume that the ore bodies will extend to depth.

B. Ore Reserves: Owing to the complex character of the ore bodies and the lack of conclusive development, it is necessary, in estimating the ore reserves in some instances, to rely upon the history of past production rather than upon a strictly technical determination, but the estimates include only known ore bodies, and do not consider the prospective value of ore bodies partially indicated by surface or underground manifestations.

Ore Reserve Estimates.

Positive			
In mine	26,200 tons @ $8.00	$209,600.00	
Dumps	25,000 tons @ $4.00	100,000.00	
Positive ore value			$309,600.00
Probable			
In mine	48,900 tons @ $8.00	391,200.00	
Dumps	15,000 tons @ $4.00	60,000.00	
Probable ore value			451,200.00
Positive & Probable ore value			$760,800.00
Possible			
In Mine	72,600 tons @ $8.00	580,800.00	580,800.00
Total -	187,700 tons		$1,341,600.00

C. Development: The development work on the property consists of several thousand feet of tunnel and raises. A considerable portion of the surface workings are still closed by caving, but the principal tunnels have been opened. The Queen of Bronze mine is developed to a depth of about 250 feet, and the Cowboy to a depth of about 120 feet. A tunnel 200 feet deeper is being driven at the Cowboy, but deeper development at the Queen of Bronze can be accomplished by sinking to better advantage.

The large unexplored area within the property, likely to carry sulfid ore bodies lends itself admirable to geophysical prospecting, followed by drilling, sinking or tunneling.

D. Equipment: At the present time the equipment on the property consits of two small gasoline-driven compressors, one each at the Queen of Bronze and Cowboy workings. For extensive work, the compressors should be larger at each property.

A mill of 100 tons daily capacity should be constructed at a convenient central point, probably the one shown on Map No. 17, about 1700 feet westerly from the "D" tunnel dump, and a surface tramway constructed between these points. Other tramways can be brought to the mill from ore producing points as needed.

A 100-ton modern flotation mill can be built for approximately $100,000. including a power plant and necessary surface tramway. There is on hand at the property surface tramway equipment, which, with some repair, can be utilized for this purpose. Either a Diesel-electric central power plant will have to be constructed at the mill and electric energy transmitted to the various mine workings, or else separate Diesel-driven compressors will be necessary at the Queen of Bronze and Cowboy workings. The two portable gasoline-driven compressors now in use can be used to good advantage in prospect work at other points on the property.

E. Recommendations: Practically all the recommendations made in my former report have been complied with. They were in substance as follows: Drive a lower crosscut tunnel at the Cowboy Mine; survey the Queen of Bronze Mine for possible lower tunnel site; build a mill and make other correlated improvements; make a complete survey of the property; acquire the Waldo Group; drive a raise from "D" tunnel level to intersect the downward continuation of ore bodies mined in the upper workings, and extend the Leach level to an intersection with the raise.

The lower Cowboy tunnel is being driven. The survey of the Queen of Bronze surface indicated that the lower tunnel is not feasible, and that sinking can be done to better advantage. The construction of a mill has been decided upon. The property has been surveyed and maps prepared. The Waldo property is held under option. The raise was driven from "D" level, and the Leach level connected with it; in the process of this development, the Leach and Pohlman ore bodies have been opened.

I now recommend at the Queen of Bronze:

(1) That the crosscut on the Leach level, starting at transit point 96 be extended to the south to prospect the downward continuation of the Messenger ore body mined above "C" level.

(2) Drive the Twohy drift on "D" level easterly to develop the Messenger and Leach ore bodies on that level.

(3) Drive "C" level west from a point at the top of the Rynski raise to locate the downward continuation of the Stevens ore body.

(4) If the work outlined above under (3) proves successful, drive a crosscut south from the extreme southwest drift of the Hampshire workings to expose the Stevens ore body on "D" level.

(5) Explore the country from the top of No. 3 Raise, "D" level.

(6) Follow ore found in top of No. 6 Raise, "D" level.

(7) Crosscut south at the foot of No. 6 Raise on "D" level.

(8) Attempt to find downward continuation of Pohlman ore on Leach level.

(9) Explore ore exposed in back of Cameron stope and east end of "A" level.

(10) Continue lower tunnel (No. 3) of the Cowboy.

(11) Follow ore found in No. 4 Tunnel of the Cowboy.

(12) Continue raise to connect with incline shaft and explore ore at the Lyttle.

(13) Make geophysical survey of the surface of the attractive unexplored portions of the group, to be followed by drilling or underground work, if mineralized area is indicated.

(14) Build a modern flotation mill of 100-ton daily capacity following flow sheet indicated by metallurgical tests.

(15) Install Diesel electric power to operate mill and mines at the mill site, or install Diesel power for driving the mill, and separate diesel compressor plants at the Queen of Bronze and Cowboy Mines.

(16) Use the surface tram equipment now on the ground, or similar device for transporting the ore from the Queen of Bronze Mine to the mill.

F. Conclusions: Climatic conditions, accessibility to a good highway, the present low hauling cost to the railway - $2.75 per ton, the prospect of being placed on direct rail transportation to the coast in the future, the large available reserves of profitable ore, together with the large unexplored area included in its holdings, make the Queen of Bronze a very attractive mining venture.

The profit made on the ore contained in the dumps should pay for mill construction and mine equipment on 100-tons-per-day basis, and the profit from the mine reserves can be used for dividends and expansion.

The demand for copper is so great, and the price so well regulated, that there is reasonable assurance of the price remaining about where it is at the present time, 18¢. With 6% mill feed, which is also reasonable to expect from the mine, the price of copper will have to drop below 10¢ per pound before operations will become unprofitable, and at the present price of copper, the ore will return a net profit of over $8.00 per ton.

In the body of this report, I have made no attempt to estimate the probable ultimate daily capacity for production of the property, but feel it is safe to assume that the 100-ton per day mill now to be constructed will be materially enlarged as unexplored territory is developed. However, the present indicated ore reserves assure not only a large return on the money already expended, but an equally profitable return on the expense necessary to bring the property to the contemplated production basis.

Respectfully,

(Sgd.) F. Cushing Moore

Spokane, Washington,
December 17, 1929.

Mr. G. E. Stowell, Assistant Mining Engineer, Oregon Bureau of mines and Geology, examined the Queen of Bronze group of mines and reported on them as of November 1919. The Stowell report goes into location, topography, geology, climate, timber, water, transportation, etc., in considerable detail, and further references are given, on the geology of the district. The Stowell report goes into the geology quite in detail and is good reasoning. Incidently, the only record of ore widths and grade are recorded, so far as the caved East stopes are concerned: The area beneath the East stope is of more interest than any other undeveloped area of the Queen of Bronze, north mine.

Briefly the Queen of Bronze, north and south mines, Waldo, Lyttle, Mabel and Cow Boy mines, are strung along the west slope of a north-south low lying ridge, south and east of Takilma, Oregon. These mines, with the exception of the Waldo constitute the Queen of Bronze group. The land holdings constitute about eight hundred acres of patented claims and ranches. The mines are named in their order from north to south, a prospect with some ore production each half mile of distance. The distance from the Queen of Bronze to the Cow Boy is two miles. The Queen the most northerly and the Cow Boy the most southerly. The Queen is partially open and is now producing some shipping ore. The Cow Boy is producing a limited tonnage of eleven per cent copper ore from bunchy ore shoots. The Queen is and has been the principal producer and is the one here under consideration.

While all of the mines have a production record, the Queen has the best both in quantity and quality of ore. The north mines seem to carry the larger ore shoots and the south ones the smaller. All, with the exception of portions of the Queen and Cow Boy, are closed and the workings inaccessible. The production entirely of high grade shipping ore-ten per cent or better copper-and from a trace to five dollars gold: The average gold perhaps three dollars per ton, shipping ore.

The rocks are serpentine varying from mushy black muddy material in the faulted areas to shiny black slickensided lumps and finally a more compact massive semi-basic rock. The ore outcrops are patches of iron gossan showing copper carbonates and at shallow depth pyrrhotite-chalcopyrite sulfides. The veins strike east-west and show a flat southeast dip. Block faulting, probably due to rock expansion tho it may be a later movement or movements, is as much responsible for the poorly defined ore shoots as the natural serpentine, metamorphic contact type, of ore deposit.

It has been advanced that the Queen of Bronze and other neighboring prospects show a decrease or impoverishment of the copper content of the ore shoots as depth is gained. The evidence in the mine do not bear out that idea. If there is a decrease in the copper content it is because of contraction in the soze of the shoots and not in grade. Development work as it now stands does not prove or disprove either contention.

The Queen of Bronze displays ore ranging from less than one percent copper and a trace of gold, carried in a heavy pyrrhotite, pyrite, chalcopyrite sulphide, to an almost solid chalcopyrite ore carrying up to eighteen percent copper. Both grades of ore, high and low, show from the bottom of the glory hole down to the bottom level, a vertical distance of two hundred feet. The lower grade ore is now in place near the top of number one raise and virtually in the bottom of the surface workings from which copper carbonate ores were mined. Eighteen

percent chalcopyrite ore is in place in the Hampshire stope at the point of deepest development. Vertically andhorizontally both grades show wherever the workings are open, in ore country, from the top of the mine to the bottom.

Low grade irony ore shows in A level forty feet west of the Number One raise near the bottom of the glory hole, and again in the westerly margin of the Messinger stope. The deepest showing of the same class of ore is in the bottom of the Twohy stope at a point a few feet above D level. The same ore shows seventy feet up Number One raise. It seems that there is a block of low grade ore from top to bottom in this section of the mine. The higher grade ore produced was mined from the McAuley and East stopes. The Messinger stope is the westerly extension of the McAuley. While the East stope is caved it is said that the higher grade ore was being produced from that point when the works were closed. Samples taken and recorded in the Stowell report substantiate the statement. The mining was down to the sill of C level when stopped. Work recently driven easterly on the Leach level, forty feet below C level, has encountered a bunch of ore showing a mixture of the two grades. An extension of the Leach work easterly and a raise connection from the lower tunnel level into the downward extension of the East ore would settle the point conclusively.

The best evidence of high grade ore at depth is that now showing in the sill of the Hampshire stope at bottom of the deepest development. The shoots are short and are not to exceed two feet thick at the best. Further prospecting should be done on the Hampshire ore showing.

The deepest development at the Queen of Bronze, north workings, the D tunnel level, was driven under the Tutt management and is said to have caused suspension of work as ore of shipping and smelting grade was not found. Later attempts at reopening the mine were confined to the area above D level. Short bights were taken immediately underneath known ore shoots and without making an attempt to locate ore at greater depth. Most of the lower tunnel workings are either beneath the ore shoots or out of the ore areas. There is not much reason to expect ore except at the one point where work was pushed into ore ground. The only ore showing on the lower level is some irony low grade stuff immediately south of the Twohy stope and in the back of the cross cut, and, in the sill in and around the Hampshire stope. The Hampshire ore is a good grade of chalcopyrite in short shoots.

D level, from the portal into the northeast face and the most easterly cross cut to the south are underneath all of the known ore shoots and in the foot wall country. The Hampshire cross-cut, to the south, passes under the Twohy irony ore shoot and into the Hampshire ore shoot. Neither the Twohy nor the Hampshire ore has been prospected on the level sufficiently to define the extent. The Hampshire ore shows a different dip and strike from the Twohy ore. The Hampshire may be an entirely new ore shoot not known on the surface of elsewhere. The fact that ten thousand tons of sulphide ores were mined from the south workings, a thousand feet south of the Hampshire ore showing, is sufficient reason for continuing the prospecting on to the south.

A cross-cut was driven ver a hundred feet to the north from the most southerly and easterly extension of the D tunnel workings, the object to pick up the downward extension of the East ore shoot. The East stope is caved and inaccessible. It is known that the ore still remains in the sill of C level. The North cross-cut is

under the East ore. A raise, if put up under the dip, would no doubt cut the ore in from fifty to sixty feet and locate the shoot sufficiently for further prospecting on the tunnel level. A bunch of ore has recently been found in the east face of the Leach cross-cut, a hundred feet to the west and is probably the west end of the East or McAuley ore shoots.

Block A, maps No. 1 and No. 2 shows the only accessible developed block of ore in the mine. Block B shows a partially developed block and the most promising section for ore production; This block should be opened. Block C deserves further prospecting. The sulphide ore dumps show fifteen thousand tons of profitable grade ore.

The block of ground marked A, map No. 1., has been opened on A,C,D, levels and in intermediate work, drifts, cross-cuts and raises. The openings were driven thru block A and on into the easterly ore shoots where a shipping grade of ore was recovered. In driving thru block A lower grade irony shoots were encountered. Parts of the ore shoots are now open and it is possible to form some idea of the ore extent and grade. In making the estimate considerable attention was given to records of former sampling and test lots. Recent sampling verify, to a reasonable extent, theformer record.

Some parts of the west end of the Messinger stopes are open, and short lengths of ore are exposed where indicated on the accompanying maps. It is admitted that an estimate, for tonnage, within block A is based on information gained from a well known characteristic of the Queen of Bronze shoots as well as the broad difference of copper content of the ore. From smelter shipments produced from known areas it is possible to make a reliable estimate of tonnage from an adjoining area especially when ore exposures are in evidence in the block under consideration, and smelter shipment records are available from the block under estimate. In making the estimate recent smelter shipments and sampling and records of former shipments and sampling of ore from block A are used. It is evident that a consistent ore shoot is not carried in the block; numerous ore showings thru and around the block make the estimate a reasonably safe one.

Block B represents the area of most probably ore. Ore of shipping grade was mined from the ground immediately above and the downward extension of that ore has not been opened. Smelter shipment records and sampling of the ore at the deepest point opened, C level, are given in following pages. The future of the Queen of Bronze is largely dependent on the opening of block B.

Block C, the Hampshire stope shows a good grade of chalcopyrite ore. Ore has been opened for a hundred feet north-south and east-west. The dip of the Hampshire ore is 45° east, the strike north-south, differing from that of the other known ore shoots. The Hampshire stope has produced enough ore of shipping grade, from short shoots, to pay the development cost, certainly the development should be continued.

Sulphide Ore Dumps.

The dumps contain roughly 30,000 tons of all classes of material. On half of the dumps are diluted oxidized materials of doubtful value tho they do carry copper of undetermined quantity. There are fifteen thousand tons of sulphide ores of a copper-gold content sufficient for profit, if a satisfactory saving can be accomplished.

The sulphide dumps are made up of three classes of ore; Large lumps of heavy iron weighing up to a hundred pounds or better. Intermediate sizes averaging the size of a base ball. Partially oxidized fines. The three sized are about equally divided; one third, or five thousand tons of each. The copper content varies with the sizes.

Samples of the dump materials were taken by trench-around the contours of the dump and from toe of slope to top. Chips were cut from the larger lumps and the smaller lumps were broken down. The fines were selected in equal proportions along all cuts. Roughly twenty tons of material was cut down to a one ton sample lot, divided as follows: Eight sacks 800 lbs. coarse lumps; seven sacks, 700 lbs. intermediate lumps; five sacks 500 lbs. fines. The ton lot of samples were assayed at the Tacoma smelter, the results as follows:

Materials.	No.	au. oz.	ag. oz.	cu. %
Coarse lumps	One	.03	.08	2.12%
Small "	Two	.055	.12	3.80
Fines	Three	.02	.08	1.78
Average		.029	.09	2.57

Some other sampling of the same dumps are of record. Mr. Roy Clarke is said to have sampled the dumps and arrived at a copper content of 2.60% copper for all of the dump material. Mr. G. E. Stowell, report of November 1919, reports 2.42% copper after rejecting waste in the form of barren serpentine.

The average grade of ore could be materially raised by sorting. One fourth of the total is clearly waste serpentine. It is probable that an additional tonnage could be recovered by sorting some of the more dilute dumps. The estimate of fifteen thousand tons of 2.60% copper ore is a fairly safe one.

Mine Ore-Block A.

Block A offers ten thousand tons of mine run ore. The basis of estimate made on ore showings now in evidence at the margins and thru the center of the block. Records of former test samples, of record, recent smelter shipments and sampling of mine run. An assay chart cannot be prepared as the ore exposures are limited, because of caving ground and the fact that former workings do not expose the ore across dip, or show lengths. C level should be opened and the ore exposures made available for sampling, and if desired, mining. The cost would be small and the result probably profitable..

Six samples were cut across the ore at the six points of ore showings within the block. The results as below:

Location.	No.	Width	Au.	Ag.	Cu.%
Twohy stope	M-13	6 ft.	.20	selected	7.40
E. Drift	M-14	4 "	.04	"	4.60
No. 2 Raise	M-15	8 "	.08	"	3.20
Messinger Stope	M-16	6 "	.16	"	7.80
C Level	M-17	4 "	.02	"	2.80
A Level	M-18	5 "	.06	"	7.90
Leach, east face	M-19	Mine Run	.80	20 tons	3.00

During September, 1928, two 50 ton lots of sorted ore were shipped, from the Twohy stope, to the Tacoma Smelter. The mine run sampling, after sorting, and the smelter lots establish a fair basis of estimate for the lowest grade ore showing in the mine:

	Lot No.	au.	ag.	Cu.%
Smelter lot, 50 tons	2034	.06	.22	6.49
" " " "	2207	.05	.24	5.35

Roughly 500 tons of mine run were sorted to get the above 100 tons of shipping ore. Sampling of the rejected ore gave the following:

Sample No.	Au.	Ag.	Cu.%
M-3			2.30
M-4			1.50
M-6			1.90
M-10	.22	.44	1.80
M-12	.02	tr	1.40
			1.78

The rejected waste from sorting is roughly one half of the total mined. It appears that the ore shall have to be sorted in any event and that sorting offers the best chance for a profitable operation. The grade of mill ore could be roughly doubled thru sorting and the above rejected material raised to 3% copper instead of the larger tonnage of 1.78% stuff. The tonnage reduced by one half as well.

The locations of ore showing within block A are indicated on Map No. 1. The widths show from 5 to 7 feet across the dips. Length exposures are limited. It is fairly certain that the continuity is at least that of the length and depth of the block. A ton of sample ore was taken from the exposures in the Messenger stope and from A and C levels, and cut down to a hundred pounds. The hundred pound sample forwarded to the Tacoma Smelter for assay. The result as below:

No.	Width	Au.	Ag.	Cu.%
5	5 ft.	.065	.13	5.05

The ore here represented that of the better grade sulphide and very similar to that open in the Twohy stope.

A Faust jig test lot, R. E. Stowell report of June, 1917, shows 7000 pounds of 3.4% copper ore. The lot from the Messinger iron stopes, R. E. Stowell report of November, 1919, shows a sample taken across 12 feet of .06 oz. gold and 5.4% copper ore. The same report shows a five foot sample of 1.3% copper ore. These last two samples indicate the range of copper values. It is evident that a properly constructed and operated sorting plant would eliminate the waste serpentine and less than 1% iron ore, and recover the shipping ore and the $2\frac{1}{2}$ to 3% mill ore.

Mine Ore-Block B.

No tonnage estimate is made for block B. The ore area is inaccessible. Ten thousand tons of .15 oz. gold and 10% copper ore was mined and shipped from the ground immediately above block B. It is assumed that the downward extension of the

ore shoots are within block B. Two hundred feet of development drifts, raises, and cross-cuts would open the ground and if the downward extension proven there should be shown a tonnage equal to that mined, 10,000 tons, of shipping grade ore and an equal tonnage of mill ore. The only record is that contained in the 1919 report of G. E. Stowell. Stowell shows the result of five sample cuts, as below.

Location	Width	Au.	Ag.	Cu. %
East Stope	5 ft.	.04	-	4.8
" "	8 "	1.00	-	4.0
" "	6 "	.08	-	4.7
" "	7½ "	.06	-	7.0

MINE Ore-Block B.

Messinger and Cameron, leasers at the Queen mine, shipped a 61 ton lot of ore from the East stope. The ore was mined during 1926, and is the last shipment from that stope. The lot returned as below:

Place	Smelter	au.	ag.	cu. %
East stope	Tacoma	.09	.39	10.13

Mine Ore-Block C. Hampshire Stope.

The Hampshire ore is of a better grade chalcopyrite than that showing in block A. A 50 ton car shipped to Tacoma Smelter during September 1928, returned:

Place	Smelter	au.	ag.	cu. %
Hampshire Lot 11	Tacoma	.19	.64	11.42

Cow Boy Mine.

The Cow Boy mine has been given little attention except by leasers. Small shipments have been made at intervals from a broad slickensided serpentine structure. The vein dips about 45% easterly and has not been developed except for shallow and short depths. The ore is uniformly of better than 10% copper. The shoots show from two to six feet of ore in shoots of twenty to thirty feet length. Nothing is known of the downward extent. Two cars of ore were sent to the Tacoma Smelter during September, and October 1928. There are no reasons why shipments should not be continued and at a profit.

Place	Smelter	au.	ag.	cu. %
Cow Boy Lot #8	Tacoma	.02	.28	11.11
Cow Boy Lot #10	"		.31	12.03

Other Mines.

Little is known of the Lyttle and Mabel mines of the Queen of Bronze group. Some production was made during the early history. The workings are caved and closed. The surface exposures show much the same character as the other mines. They no doubt deserve some attention, but after the Queen problem is worked out.

Respectfully,

(sgd) Byron Wilson

REPORT ON THE PROPERTY

of the

QUEEN OF BRONZE MINING CO.,

TAKILMA, JOSEPHINE COUNTY,

OREGON.

Made for
Pohlman Brothers,
Spokane, Wash.

By

F. Cushing Moore,
Spokane, Wash.

December, 1929.

F. CUSHING MOORE
MINING ENGINEER
SPOKANE, WASHINGTON

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F. CUSHING MOORE
MINING ENGINEER
SPOKANE, WASHINGTON

REPORT ON THE PROPERTY

of the

QUEEN OF BRONZE MINING CO.,

Waldo District, Josephine County, Oregon.

By F. Cushing Moore,
Spokane, Wash.
December 1929.

I. Location, Transportation, Topography, Climatic Conditions. See Map No.2.

A. Location & Topography:

The property is located on the west slope of a prominent Ridge between Althouse Creek and East Branch of the Illinois River, about one mile southerly from Takilma, 40 miles, west of Grants Pass, Josephine County, Oregon, and 50 miles northeast of Crescent City on the California Coast. Elevation at Takilma is 1567 ft. above sea level,

"	"	Queen of Bronze, 2200 to 2450	do
"	"	Cowboy, 2600 to 2800	do
"	"	Waldo 2250 to 2400	do

B. Climatic Conditions:

The climate is very mild; snow NEVER remains for any great length of time, and the average rainfall is about 40 inches.

C. Roads, Railroads, Transportation:

The Redwood Highway passes within about six miles of Takilma, and a good road is available from the mine to the Highway. At the Cowboy workings it is necessary to haul the ore on sleds to the lower tunnel, which is in process of construction. This entails a cost of 50¢ per ton, which will be eliminated when the lower tunnel is connected with the upper workings. Truck hauling from the mine to Water Creek railway station, 27 miles distant, is contracted for at $2.75 per ton.

II. Extent of Property, Area, Titles. See Map No. 1.

The total area of the patented ground owned by the Queen of Bronze Mining Co., is 382.2 acres, comprising an irregular-shaped block extending over three miles north and south, by two miles east and west. Within this block of ground is located the property of the Waldo Copper Company, comprising 480 acres of patented ground, which is held under lease and bond by the Queen of Bronze Mining Co., at a total purchase price of $50,000 with very favorable terms.

In addition to the patented ground, mining claims have been located to the extent of 220 acres, giving a total area held by the company of 1582.2 acres. The county records have recently been searched by Roy H. Clarke, and title is clear as above stated.

III. History:

The district was first discovered in 1860, but practically no development was undertaken until 1903. A 100-ton smelter was built on the premises under the Tutt ownership in 1904, and operated for five years, handling 20,000 tons of 8% Copper ore from the Queen of Bronze, besides considerable from the Cowboy, Lyttle, Mabel, and other properties. This ore was reduced to matte and hauled over 40 miles of rough road to Grants Pass. Coke cost $20.00 per ton delivered, and other supplies in proportion.

From 1910 to 1916, the property was handled by leasors who shipped a few thousand tons of ore to the Tacoma Smelter. In 1916 the property was taken over by the Twohy interests, under the management of Roy H. Clarke, who operated it until the drop in the price of Copper, at the close of 1918. During these three years, 9,992.3 tons were produced from the Queen of Bronze, having a net smeltery value of $432,440.88, and 1644.46 tons from the Cowboy, with a net smeltery value of $73,562.91.

During the following few years, leasers made intermittent shipments from the property.

In 1928 and the spring of 1929, John Hampshire of Grants Pass, Oregon, operated the property under lease, producing up to the time of the incorporation

of the Queen of Bronze Mining Co., (May 31, 1929) about $39,994.20 net smeltery value. Shipments from June 1st to November 1, 1929, netted $23,589.30, giving a total for 1928 and 1929 of $63,994.20

The gross value of all ore shipped to date from the Queen of Bronze and other properties in the Queen of Bronze ore zone is approximately $1,500,000 with net operating profit of about $500,000.

IV. Economic Value of Ore:

A. Valuable Metals in Ore:
The metals of commercial value occurring in the ore given in the order of their importance, are:- Copper, Gold and Silver.

B. Stability of Metal Prices:
Copper averaged about 15¢ per lb. for many years, and, during the War, it was much higher, but up to this spring has been lower. In April of this ye ar, it advanced to 24¢ per lb., and now is quoted at 18¢. I feel that it is safe to assume that the price of Copper will stay around 18¢ per lb., and all estimates of value in this report are based on this price.

Gold is paid for by the smelters at $20.00 per ounce; Silver is worth about 50¢ per ounce at present, and is likely to remain about that.

C. Character of Ore; Amenability to Reduction; Values:

1. Ore Minerals:
The principal ore mineral is Chalcopyrite, and intimately associated with it is found Pyrrhotite and Pyrite, accompanied by some quartzand Calcite. In the upper areas, the Chalcopyrite has oxidized to Malachite, Azurite, Cuprite and Chalcocite. Native Copper is found occasionally.

2. Sampling:
I made no attempt to do any hand sampling, but from samples taken by Roy H. Clarke, G. E. Stowell and Byron Willson, and from the mining and shipment records to date, it is safe to estimate that the ore, as sorted for shipment, will average about 10% Copper and .06 ounces in Gold, per ton. The Silver is below pay value. Recent shipments from the Cowboy have run as high as 14% Copper. The last car of 54 tons netted $1868.49.

3. Sorting:
As operations are being conducted at the present time, one ton of hand-sorted shipping product is the resultant of about 5 tons of ore removed from the mine, and some waste is sorted out under ground and put back as filling. Metallurgical tests made on the mine run ore would indicate that probably after the construction of the mill, sorting will be confined to the elimination of waste.

4. Milling:
Milling tests made by the Federal Mining & Smelting Co. on samples taken from various parts of the mine and dump indicated that a saving of 95% of the Copper and 85% of the Gold could be made when a concentrate of 8-1/2% Copper content was produced. This, of course, was a high extraction, but the grade of the concentrate was too low. Recently other tests made by the Bunker Hill & Sullivan M. & C. Co., and the General Engineering Co., of Salt Lake, indicate that a much higher grade of concentrate is possible, but the extraction is slightly lower. The results of these tests are as follows:

Bunker Hill & Sullivan M. & C. Co.-
Heads, Copper 5.2%; Gold .0325 oz.
Copper recovery 92%; Gold recovery 75%.
Concentrate produced Copper 18% carrying .10 oz Gold per ton.

General Engineering Co., Salt Lake,-
Heads, Copper 5.345%; Gold .025 oz.
Copper recovery 92%; Gold recovery 75%.
Concentrate produced, Copper 18% carrying .12 oz.Gold per ton.

The second test just being made by the Bunker Hill & Sullivan on a different sample, indicates that probably the most efficient grade of concentrate to be produced by the mill will contain at least 20% Copper.

5. Smelting:
The most favorable smelter schedules under which shipments can be made are those offered by the American Smelting & Refining Co. plant at Tacoma, which are as follows:

Crude Ore Schedule:
Pay for 100% of the Copper at the market price, less 2¢ per lb., where Copper is under 14-1/4¢ per lb., and 2-1/2¢ per lb. when Copper is selling for over 14-1/4¢ per lb. Deduct 10 lbs. Copper when Copper is under 14-1/4¢ per lb., and 15 lbs. when over 14-1/4¢ per lb. Treatment $2.50 per ton.

Pay for 95% of Gold at $20.00 per oz.
Pay for 95% of Silver at New York market.

Flotation Concentrate Schedule:
Pay for 100% of Copper, less 2-3/4¢ per lb.
Deduct 20 lbs. Copper from assay.
Pay for 95% Gold at $20.00 per oz.
Pay for 95% Silver over half an oz.
Base charge for treatment, $2.75 per ton, for concentrates containing 12% Copper or less. Add to base charge 10¢ per unit over 12% up to a maximum additional charge of $2.25 a ton. Add to base charge 25¢ for each 1¢ increase in Copper quotation over 17¢.

Freight from Waters Creek to Tacoma-

Under $20.00 valuation		$4.41	per	ton
$20. to $30.	"	$4.75	"	"
$30. to $40.	"	$5.10	"	"
$40. to $50.	"	$5.45	"	"
Over $50.00	"	$5.80	"	" (assumed)

D. Power:
No water power has been developed, although I am informed that power can be had on the Klamath River. For the present, gasoline engine driven compressors are used. Steam power is too expensive and therefore out of the question.

When the mill is constructed, either water power will have to be developed, or electric power brought in, or Diesel engine power used. The latter is probably the most feasible, as the initial cost will be very much less, and the fuel oil can be handled in drums by the trucks as a back-haul from Waters Creek Station. To minimize the expense of handling the oil, the power plant should be located conveniently to the loading bins at the mill, and a storage tank provided at Waters Creek Station.

The cost of power generated by Diesel engines compares very favorably with electric power, where fuel oil can be delivered cheaply. Gasoline is being delivered in tank trucks by the oil companies at

Takilma and Grants Pass from ship cargoes landed at Crescent City, California at a very reasonable figure, but I see no reason why fuel oil cannot be delivered via Crescent City for less money than via Waters Creek Station. In any event fuel oil can be delivered at Waters Creek Station at 5-1/2¢ per gallon and contracted for as a back-haul on the hauling of concentrate shipments for not to exceed 1¢ per gallon. This would furnish fuel oil at the plant for 6-1/2¢ a gallon. This would mean a fuel cost of 7¢ per hp. hour including labor. This cost compares very favorably with that of electric power at the smaller properties in the Coeur d'Alene Mining District.

Estimate of cost of operation of a 275 h.p. Diesel-Electric power plant:

Fuel oil consumption @ .42 lbs.per h.p. hour - 15 gals
15 gals.@ 6-1/2¢ - 97¢ per hour, or about 4/10¢ per h.p. hour.

Labor, 3 men @ 5.00 per day,	$15.00
Lubricating oils, & incidentals	2.50
Fuel oil, 15 gals per hour for an average of 20 hours per day @ 6-1/2¢	19.50
Total cost per day for 275 h.p.=	$37.00

Total cost per h.p. hour - 7/10¢
Cost per h.p. year - $48.50
Cost per ton mined on 100-ton-per-day basis - 37¢

E. Ore Extraction - Mining & Milling:

1. Mining:

Ample labor for mining purposes has been available at less wages than prevail in most of the larger mining camps.

Working conditions have been ideal for economic operation, as all the miners have lived in the vicinity of Takilma and go back and forth in privately owned autos. When a large crew is required it may possibly be necessary to operate a company-owned boarding and bunk house adjacent to the mine workings.

Early mining was done by hand, but recently drilling underground has been accomplished by two portable gasoline-driven compressors, one each at the Queen of Bronze and the Cowboy workings.

Ample timber for mining purposes for many years to come is available on the premises. Water for milling and domestic purposes is also in abundance.

Supplies are procurable at Grants Pass and there is a small store at Takilma.

Practically all of the ore will be taken from tunnel workings. Most of the ore thus far mined has come from the Queen of Bronze workings, although the Cowboy is now a material producer.

Extraction cost for hand-sorted ore has been quite definitely established during the past year at $15.75 per ton, made up as follows:

Mining	$10.00	per ton
Development	3.00	" "
Hauling to Waters Creek	2.75	" "
Total extraction cost,	$15.75	per ton

This figure is based upon one ton of shipping product averaging 10% Copper and .06 oz. Gold per ton. It includes all underground costs, surface sorting, and transportation to the railroad. The cost for each ton as taken from the mine averages $2.60 per ton.

Under the present method of sorting, for every ton of shipping product, four tons of low-grade ore are put on the dump. When a mill is available, this sorting will be done on a belt, and the rejects go directly to the mill; Thus materially reducing the cost per ton of the shipping product, as well as converting the rejects from the sorting into mill feed, with no additional mining expense.

As nearly as can be determined from samples taken and records of operations to date, the mine ore should average 6% Copper, and, after having one-fifth sorted out as high-grade (10% Copper), the remaining mill feed would average 5% Copper, i.e.-

5 tons Mine ore averaging 6% Copper = (1 ton hand-sorted ore (10% Copper)
(4 tons mill feed (avg. 5% Copper = (1 ton conc. (18% Copper)
(3 tons tails to waste.

2. Milling:
It will be advisable to centralize the equipment by locating the mill and power plant at some convenient point, and transport the ore to the mill by short tramways. The necessity for this is apparent, as ore will be mined from a number of points along

the course of the ore zone, which extends over two miles. The topography of the country lends itself admirably to this arrangement. A mill site has been selected immediately west of the west end of the Annie Claim on patented ground. See Maps No. 1 and 14.

A surface tram, constructed from the "D" tunnel of the Queen of Bronze to the proposed mill site would be 1665 feet long. Map No.17 shows a profile of the tram line.

Estimate on Operating Cost of Mill, 100 tons of Mine Ore Per Day, Straight Concentration - No Sorting.

	Per Day	Per Ton
2 men in mill per shift @ 5.00 each	$30.00	.30
1 man in sorting plant sorting waste	5.00	.05
Reagents per ton - 15¢		
Power - 40¢		
Supplies - 25¢		.80
Total cost per ton		$1.15

In all computations following, I have used the figure, $1.50 per ton, to be on the safe side, but in actual practice it will probably be nearer the above estimate of $1.15 per ton, as the indicated flow sheet is extremely simple and the power cost low.

3. Calculations for Net Value of the Ore.

Calculations based on the present grade of shipments taken in combination with the concentrate to be produced by the mill and upon concentrates of various grades exclusive of any hand-sorted product are as follows:

(a) A combination of hand-sorted and milling ore where the hand-sorted product taken from the picking belt prior to milling will be of the average grade of shipments from the mine during the past year, and the concentrate produced from the rejects from this sorting will be equal to the results obtained from the recent metallurgical tests on such ore:

Mine run ore - Copper 6%.
Hand-sorted product (one-fifth of mine run Ore)
Copper 10%; Gold .06 oz.
Copper concentrate produced from rejects,
Copper 18%; Gold $2.00, per ton.

Hand-sorted ore, Copper 10%; Gold .06 oz. per ton.
Copper @ 18¢ less 2-1/2¢ = 15-1/2¢ net per lb.

Copper 200 lbs. less 15 lbs @ 15-1/2¢ =		$28.67
Gold .06 oz. x 95% @ $20.00 per oz	=	1.14
Pay for copper and gold		$29.81
Freight 4.75		
Treatment 2.50		7.25
Net smelter return (hand-sorted ore)		$22.56 per to
Mining 5 tons - 10.00		
Development - 3.00		
Truck haul - 2.75		15.75
Net operating profit on each ton of hand-sorted ore		$ 6.81

As the milling ore is the reject from the sorting of the crude ore, all expenses have been deducted except milling expense and the transportation of the concentrates. Therefore we have four-fifths of the original mine run of ore as mill feed.

Assumed flotation concentrate, Copper 18%, Gold .11 oz. per ton.
Copper 18¢, less 2-3/4¢ = 15-1/4¢ per lb.

Copper 18%; 360 lbs. less 20 lbs = 340 lbs @ 15-1/4¢		$51.85
Gold .11 oz x 95% @ $20.00		2.00
Silver 1 oz. less 1/2 oz = 1/2 oz @ 50¢		.25
Pay for Copper, Gold, Silver, per ton		$54.10
Hauling - 2.75		
Freight to smelter - 5.80		
Treatment charge - 3.60		
Milling 4 tons - 6.00		18.15
Net operating profit on concentrate		35.95
Net operating profit on hand-sorted ore		6.81
Total operating profit on 2 tons ore		42.76 per ton

On one ton each of crude ore and concentrate produced from 5 tons of mine run.
Combined profit per ton of ore as mined- $8.55

(b) Concentration without sorting of Crude Ore.
Copper 18% concentrate carrying $2.00 in Gold.
Mill feed, Copper 6% - Recovery 92%. Copper 18¢.
Ratio of concentration - 3.26 to 1.

Copper 18% - 360 lbs.less 20 lbs @ 15-1/4¢		$51.85
Gold		2.00
Silver		.25
		54.10
Hauling to RR - 2.75		
Freight - 5.80		
Treatment - 3.60		
Milling @ 1.50- 4.90		
Mining @ 2.50- 8.15		25.20
Net value of 1 ton of concentrates		28.90
Net operating profit 1 ton mine ore		8.87

(c) Concentration without sorting - 6% Copper mill feed.
Copper 20% concentrate, recovery 90%, Gold $2.25.
Ratio of concentrate 3.7 to 1.

Copper 20% - 400 lbs.less 20 lbs, 380 lbs. @ 15-1/4¢		$57.95
Gold $2.50, Silver 30¢		2.55
		60.50
Hauling - 2.75		
Freight - 5.80		
Treatment - 3.80		
Mining @ 2.50 - 9.25		
Milling @ 1.50- 5.50		27.10
		33.40
Net operating profit 1 ton mine ore		9.00

(d) Concentration without sorting - 6% Copper feed.
Copper 20% concentrate, recovery 92%, Gold $2.25.
Ratio of concentrate 3.62 to 1.

Copper 20% - 400 lbs, less 20 lbs. 380 lbs @ 15-1/4¢		$57.95
Gold $2.50, Silver 30¢		2.55
		60.50
Hauling - 2.75		
Freight - 5.80		
Treatment - 3.80		
Mining @ 2.50- 9.05		
Milling @ 1.50- 5.44		26.84
		33.66
Net operating profit 1 ton mine ore		9.30

(e) Tabulations of Net Values Per Ton of Mine Ore, Based upon a Concentrate Carrying 18% Copper and $2.00 in Gold Per ton. Range of ore values 3% to 6% Copper. Range of Copper prices 0 to 25¢ per lb. Overhead expense 50¢ per ton.

Mine Run 6% Copper:

Price			Operating Profit			Net Profit
Copper price	22¢	per lb. -	$12.35	less 50¢	overhead	$11.85
" "	20¢	" " -	10.70	" "	"	10.20
" "	18¢	" " -	8.87	" "	"	8.37
" "	15¢	" " -	5.43	" "	"	4.93
" "	12¢	" " -	2.61	" "	"	2.11

Mine Run 5% Copper:

Copper price	22¢	per lb. -	$10.12	less 50¢	overhead	$ 9.62
" "	20¢	" " -	8.30	" "	"	7.80
" "	18¢	" " -	6.67	" "	"	6.17
" "	15¢	" " -	4.09	" "	"	3.59
" "	12¢	" " -	1.77	" "	"	1.52

Mine Run 4% Copper:

Copper price	22¢	per lb. -	$ 6.90	less 50¢	overhead	$ 6.40
" "	20¢	" " -	5.82	" "	"	5.32
" "	18¢	" " -	4.53	" "	"	4.03
" "	15¢	" " -	2.29	" "	"	1.79
" "	12¢	" " -	0.41	" "	" Loss-	.09

Mine Run 3% Copper:

Copper price	22¢	per lb. -	$ 4.18	less 50¢	overhead	$ 3.68
" "	20¢	" " -	3.36	" "	"	2.86
" "	18¢	" " -	2.40	" "	"	1.90
" "	15¢	" " -	.72	" "	"	.22
" "	12¢	" " Less-	.54	" "	" Loss-	1.04

See Graph on next page.

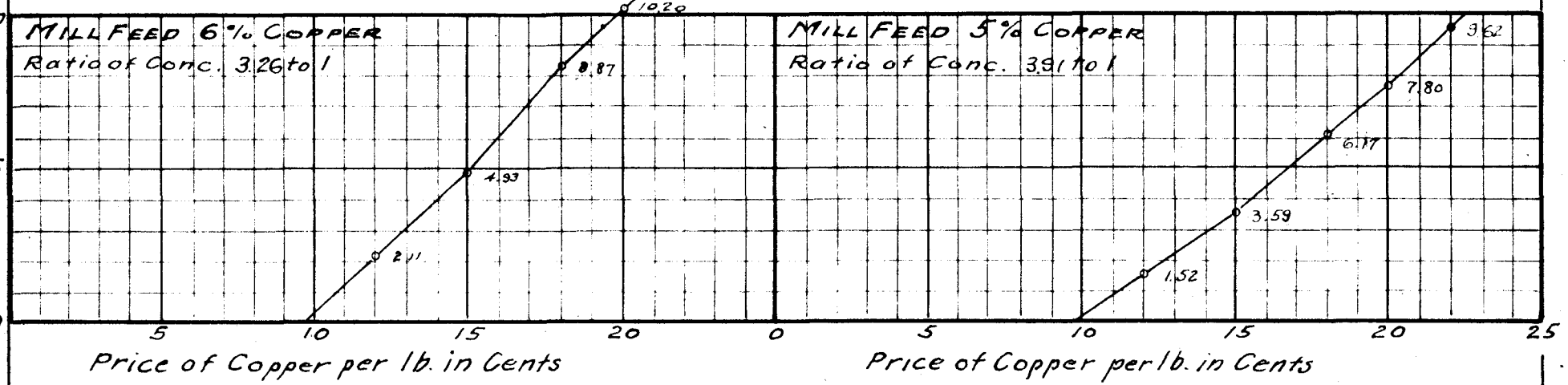
(f) Record of Shipments of Queen of Bronze Mining Co. Between August, 1928, and November 1st, 1929.

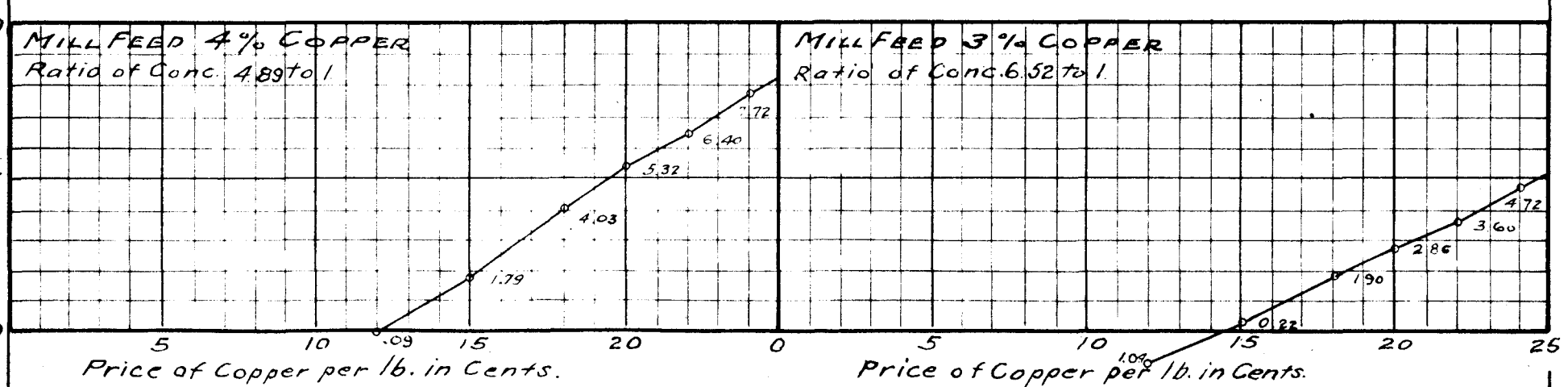
No. Cars	54
Net Tons Shipped	2798.86
Average Copper content	10.29%
Average Settlement price	14.42
Average Gross value Copper per ton	28.98
Average Gross value Gold per ton	1.18
Total net value shipments after all deductions, freight and smelter.	63583.50
Average Net value per ton	23.72

QUEEN OF BRONZE MINE
JOSEPHINE CO., OREGON

PROFIT PER TON RUN OF MINE ORE = INCLUDING MINING, MILLING & OVERHEAD EXPENSE.
FLOTATION CONCENTRATION = MILL RECOVERY 92% COPPER
GRADE OF CONCENTRATE = COPPER 18% - GOLD $2.00 PER TON.

NET PROFIT PER TON MINE RUN ORE IN DOLLARS.





F Cushing Moore.

12 A.

F. Overhead Expense:
Up to the present time, the overhead expense on the property has consisted of a manager, drawing $300 a month, and an office girl, with intermittent engineering expense and traveling expense for officers of the Company. From now on, the overhead expense will be considerably greater, and will include the salaries of a manager, an assayer-engineer, and a bookkeeper at the property. This with traveling expense of officials, and other incidentals will probably amount to $1,000.00 a month. For the sake of conservatism, I am placing the overhead at $1,500.00 per month, or 50¢ a ton on a 100-tons-per-day basis.

V. Extent and Permanency of Ore Deposits:

A. Geology: Ore Genesis:
At the Cowboy, Lyttle and Mabel Mines, the ore bodies are found in ill-defined mineralized zones in or associated with the serpentine, closely related to the diorite or andesite. The ore occurs in irregular bodies and fragments varying in size from several thousand tons to a few pounds, showing broken angular shapes and slickensided surfaces, crushed into the softer enclosing rock with no apparent order or connection. But at the Queen of Bronze the ore has been less disturbed and less serpentine is in evidence.

Quoting from a report by G. E. Stowell, Mining Engineer with the Oregon Bureau of Mines & Geology, who investigated the property several years ago when many of the upper works of the mine were accessible, which have since been closed by caving:
"Briefly the rocks of the district include both sedimentary and igneous types. The sedimentaries are represented by older slatey rocks, quartzites and limestones, and more recent gravels and sands. Associated with these are the igneous intrusives, practically all being classified as andesite, peridotite, augite, gabbro, diorite, greenstone and serpentine, etc.

"Doubtless the alteration of such rocks as peridotite, pyroxenite and gabbro have produced the serpentines so common in the region, while alteration of the diorite and andesite, etc., accounts for the greenstones. The Copper deposits are associated with one or more of the above named igneous rocks, and so far as is known, have no

direct relation to the sedimentaries other than the ore deposits are found in a belt of country extending east of north in the direction which conforms roughly with the strike of the sedimentary rocks in the vicinity.

"The most common rock associated with the ore is serpentine, although the ore is not confined to this rock alone, but is found in adjacent rocks less altered and less basic in character."

In all geological publications, the theory of descending solution with resultant shallow deposits, has been advanced, but recent development has established the fact that the ores do extend to considerable depth. The theory that mineralized waters arising from deep-seated rock intrusions are responsible for ore deposits conforms more closely to present observed conditions. The evidence is in the mineral associations and character of the ore deposit. Pyrrbotite especially is a mineral found in deposits formed at considerable depth where comparatively high temperature and pressure was the rule. It is found also in magmatic segregations and contact metamorphic deposits, but rarely, if ever, in solution deposits at or near the surface.

As stated, at the Cowboy Mine and in one of the stopes at the Queen of Bronze Mine, the original form of the ore bodies is poorly preserved, and only the larger masses have withstood the crushing of the rock movements. In places less disturbed blocks of ground have protected to some extent the enclosed sulfids, so that one may observe stringers and veins of various sizes running through the country rock.

Briefly stated, intrusions of various rocks caused a system of shear zones to be developed across the country. These shear zones offered a means of circulation for the deep-seated mineralized waters coming from heated igneous masses. When these solutions reached points where conditions of temperature, pressure and chemical composition were favorable, the sulfids were deposited along with the silica and lime. Subsequently some of the rocks were altered to serpentine. In the formation of serpentine, an increase of from 30% to 50% of volume of the rock is involved. This increase in volume was sufficient to account for the movement indicated in the vicinity of the ore bodies, and

could easily have accounted for the crushed and dragged appearance of the ore.

Erosion has progressed to a point that some of the ore bodies have been exposed on the surface with their consequent development of gossany outcrop. As surface oxidation has progressed more rapidly than the erosion, we find the oxidized copper minerals in the upper horizon of the ore bodies.

All of the ore thus far mined has been taken from comparatively small areas, that at the Queen of Bronze being the most extensive. At this point, the ore bodies are contained within an area of approximately 400 feet in diameter, although other ore bodies have been mined in the same property at a little greater distance.

The Cowboy workings cover a very small area, and those in the Lyttle and Mabel even less.

Undoubtedly many other ore bodies will be found on the property, as the territory is not only geographically well located, but iron stained float and gossan material are exposed at various points along the line of the other ore deposits.

B. Ore Reserves:
Owing to the irregular character of the ore bodies and the defiency of development work, it is impossible to technically calculate much ore, but from the past history of the property it is safe to assume considerable reserves, although not conclusively proven.

Cowboy; Map No.13: The Cowboy has been a very material producer during the past year. The last carload of ore shipped during the month of November ran over 14% Copper. The ore shoot has an approximate length of 150 feet on the Intermediate Tunnel level, and a width of about 10 feet. On the No.2 Tunnel Level, the ore as developed is not quite so long, but possibly the northern extension of the ore has not yet been encountered.

The southern extension of the ore is in question, but as the drift encountered heavily serpentinized rock and the identity of the vein destroyed, possibly the ore has been displaced to the east. Recently a new showing of oxidized ore, carrying some high

values in Chalcocite with native copper has been opened east of the old workings. This exposure is shown on Map No.13. Tunnel No.4 has undercut the old shaft at about 26 feet and some high grade ore is being sacked for shipment. This ore appears to follow the old fault, as some chunks of unaltered sulfid ore were found in the fissure. The development of this showing to the east may possibly lead to the location of the southern extension of the Cowboy ore body.

I have estimated roughly 8000 tons of positive ore; 8000 tons of probable ore, and 8000 tons of possible ore of average grade of 6% Copper in the Cowboy.

Lyttle; Map No.12: The Lyttle Mine produced considerable high grade ore years ago but the workings have been caved until recently. A raise is being driven from No.2 Tunnel to tap the bottom of the incline shaft from No.1, where, according to best authority, considerable ore was left by former operators.

I have estimated 5000 tons of probable ore and 5000 tons of possible ore of 6% Copper grade for the Lyttle.

Queen of Bronze; Maps Nos.3 to 11, inc.

Stevens Ore Body: The Stevens ore body was mined from the "E" level through to the surface. These workings are all closed, but from information obtained from former operators and from a study of the maps, it is probable that this ore body should be encountered on "C" Tunnel level a short distance west of the top of the Rynski raise. Again this ore body should be found south and west of the southwest drift of the Hampshire workings on "D" Tunnel level.

I have estimated 1000 tons of probable ore and 6000 tons of possible ore of average grade of 6% Copper for the Stevens ore body.

Messenger-Twohy Ore Body: The Messenger ore body above "C" Tunnel level b broke into the old glory hole workings and below that level is represented by the Twohy ore body partially stoped between the "D" and Leach levels. This ore body connects with the McCauley ore body above "C" level, so that more than likely it will

connect with the Leach ore body, which is the downward continuation of the McCauley ore body on the Leach level.

Driving of the crosscut at transit point 96 on the Leach level should cut this ore.

On "D" level, the Twohy ore body was developed by drift east and west for a distance of about 50 feet, but the drift at the east end turned to the left away from the ore. This face should be advanced, as the Twohy-Messenger ore should be found to the south and east. The ore body should be approximately 100 feet long, ten feet wide, with 40 feet of backs between the Leach and "C" levels, giving 2200 tons of positive ore, 2200 tons of probable ore and between the "D" and Leach levels 7800 tons of possible ore.

McCauley-Leach Ore Body: The Leach stope on the Leach level has the downward continuation of the McCauley ore body, which was the greatest producer of any of the Queen of Bronze ore bodies. This ore body was followed through practically to the surface, is about 100 feet long, 10 feet wide, and has above the Leach level 2200 tons of positive ore, between the "D" and Leach levels 7800 tons of probable ore, and below the "D" Tunnel level 11,000 tons of possible ore.

Cameron Ore Body: Above the Cameron stope at the east end of "C" level is a block of virgin ground with the exception of the tunnel workings on the extreme east end of "A" level. This ore body is at least 50 feet long by 20 feet wide, and should extend to the surface.

I have estimated 5500 tons of positive ore, 5500 tons of probable ore, and between "D" and "C" levels 12,000 tons of possible ore.

Pohlman Ore Body: The Pohlman ore body is that ore which has recently been encountered in the top of No.7 raise above the Pohlman crosscut on the Leach level. This ore is undeveloped, in virgin ground, and may either connect with the Cameron ore body or may be an entirely independent deposit. I have estimated 2800 tons of probable ore and 2800 tons of possible ore.

Hampshire Ore Body: The Hampshire ore body was discovered on the "D" level and has been encountered by no other workings. The ore body is 100 feet long by about 15 feet wide, has been mined at the west end about 40 feet high but practically no stoping has been done at the east end. This ore should extend through to the surface and to depth. I have estimated for this deposit 8300 tons of positive ore, 16,600 tons of probable ore and 20,000 tons of possible ore.

South Workings: Former operators removed considerable ore from the South Workings of the Queen of Bronze, and a large glory hole is in evidence, but as the workings beneath this glory hole are inaccessible, and as I have no authentic data on them, I am making no inclusion of reserves in my estimates.

Dumps: At the Queen of Broze workings there are large dumps of ore which have accumulated from the shipping of high grade ore during the past years. From estimate of volume and trenching it has been determined that there are from 25,000 to 40,000 tons of ore in these dumps which will average approximately 3% Copper. The copper sulfid has oxidized to some extent in these dumps, but flotation tests have shown that very satisfactory extraction can be made. Possibly it might pay to leach them prior to flotation concentration.

RECAPITULATION OF ORE RESERVES.

	Tons of Ore Reserve		
	Positive	Probable	Possible
Cowboy Mine	8000	8000	8000
Lyttle Mine	-	5000	5000
Queen of Bronze:			
Stevens, above "D" level	-	1000	6000
Twohy-Messenger, above "D" level	2200	2200	7800
McCauley-Leech " " "	2200	7800	-
" " below "D" level	-	-	11000
Cameron, above Cameron Stope	5500	5500	-
" above "D" level	-	-	12000
Pohlman, above "D" level	-	2800	2800
Hampshire " "D" level	8300	16600	-
" below "D" level	-	-	20000
	26,200	48,900	72,600
Ore in dumps, 3% Copper	25,000	15,000	
	51,200	63,500	72,600

Positive ore 51,200 tons
Positive & Probable ore 115,100 tons
All estimated ore reserves 187,700 tons

Positive			
In Mine	26,200 tons @ $8.00	$209,600.00	
Dumps	25,000 tons @ $4.00	100,000.00	
	Positive ore value		$309,600.00
Probable			
In Mine	48,900 tons @ $8.00	391,200.00	
Dumps	15,000 tons @ $4.00	60,000.00	
	Probable ore value		451,200.00
	Positive and Probable-		760,800.00
Possible			
In Mine	72,600 tons @ $8.00	580,800.00	580,800.00
	T o t a l		$1,341,600.00

This estimate includes only known ore bodies, and does not consider the prospective value of ore bodies partially indicated by surface manifestations.

In ore reserve calculations, a ton of ore is figured as nine cubic feet of ore in place.

VI. Summary, Conclusions, Deductions and Recommendations:

A. Ore Deposits: The rocks in the immediate vicinity of the Queen of Bronze ore bodies are igneous intrusions or flows, principally andesite, gabbro, diorite, etc., altered in places to serpentine and greenstone. The ore bodies were undoubtedly deposited by ascending mineralized solutions related to the intrusion of igneous rocks at great depth.

Subsequent to the ore deposition, erosion lowered the surface to the proximity of the ore deposits. Oxidization has been more rapid than erosion, causing oxidized zones above the sulfid ore bodies. Rock movement along shear zones has caused alteration of the basic rocks and alteration of the basic rocks to serpentine may have been responsible for the movement. In any event, some of the ore bodies are much disturbed and the ore in places occurs as boulders surrounded by a serpentine coating. The ore has the appearance of being squeezed rather than crushed or rolled by movement.

The ore is a chalcopyrite associated with pyrrhotite and pyrite. In the oxidized zone are found asurite, malachite, cuprite, chalcocite and native copper. Gold and Silver are associated with the Copper. Average milling ore will carry approximately 6% Copper and .06 oz. Gold, and .30 oz. Silver per ton. The Gold values are very erratic, but

F. CUSHING MOORE
MINING ENGINEER
SPOKANE, WASHINGTON

the average of all shipments made to date ran $3.07 per ton in a hand-sorted ore averaging 10.84% Copper ,-exclusive of early shipments, of which we have no detailed record.

The ore bodies thus far developed are within areas of no great dimensions, but the mineralized belt extends through the property in a northerly and southerly direction for over two miles. Undoubtedly the Queen of Bronze ore zone is in the northern extension of the mineralized copper area of Northern California.

From the nature of the deposit and associated minerals, it is safe to assume that the ore bodies will extend to depth.

B. Ore Reserves: Owing to the complex character of the ore bodies and the lack of conclusive development, it is necessary, in estimating the ore reserves in some instances, to rely upon the history of past production rather than upon a strictly technical determination, but the estimates include only known ore bodies, and do not consider the prospective value of ore bodies partially indicated by surface or underground manifestations.

Ore Reserve Estimates.

Positive			
In mine	26,200 tons @ $8.00	$209,600.00	
Dumps	25,000 tons @ $4.00	100,000.00	
Positive ore value			$309,600.00
Probable			
In Mine	48,900 tons @ $8.00	391,200.00	
Dumps	15,000 tons @ $4.00	60,000.00	
Probable ore value			451,200.00
Positive and Probable ore value . . .			760,800.00
Possible			
In Mine	72,600 tons @ $8.00	580,800.00	580,800.00
Total-	187,700 tons		$1,341,600.00

C. Development: The development work on the property consists of several thousand feet of tunnel and raises. A considerable portion of the surface workings are still closed by caving, but the principal tunnels have been opened. The Queen of Bronze mine is developed to a depth of about 250 feet, and the Cowboy to a depth of about 120 feet. A tunnel 200 feet deeper is being driven at the Cowboy, but deeper development at the Queen of Bronze can be accomplished by sinking to better advantage.

The large unexplored area within the property, likely to carry sulfid ore bodies lends itself admirably to geophysical prospecting, followed by drilling, sinking or tunneling.

D. Equipment: At the present time the equipment on the property consists of two small gasoline-driven compressors, one each at the Queen of Bronze and Cowboy workings. For extensive work, the compressors should be larger at each property.

A mill of 100 tons daily capacity should be constructed at a convenient central point, probably the one shown on Map No.17, about 1700 feet westerly from the "D" tunnel dump, and a surface tramway constructed between these points. Other tramways can be brought to the mill from ore producing points as needed.

A 100-ton modern flotation mill can be built for approximately $100,000. including a power plant and necessary surface tramway. There is on hand at the property surface tramway equipment, which, with some repair, can be utilized for this purpose. Either a Diesel-electric central power plant will have to be constructed at the mill and electric energy transmitted to the various mine workings, or else separate Diesel-driven compressors will be necessary at the Queen of Bronze and Cowboy workings. The two portable gasoline-driven compressors now in use can be used to good advantage in prospect work at other points on the property.

E. Recommendations: Practically all the recommendations made in my former report have been complied with. They were in substance as follows: Drive a lower crosscut tunnel at the Cowboy Mine; survey the Queen of Bronze Mine for possible lower tunnel site; build a mill and make other correlated improvements; make a complete survey of the property; acquire the Waldo Group; drive a raise from "D" tunnel level to intersect the downward continuation of ore bodies mined in the upper workings, and extend the Leach level to an intersection with the raise.

The lower Cowboy tunnel is being driven. The survey of the Queen of Bronze surface indicated that the lower tunnel is not feasible, and that sinking can be done to better advantage. The construction of a mill has been decided upon. The property has been surveyed and maps prepared. The Waldo property is held under option. The raise was driven from "D" level, and the Leach level connected with it; in the process of this development, the Leach and Pohlman ore bodies have been opened.

I now recommend at the Queen of Bronze:

(1) That the crosscut on the Leach level, starting at transit point 96 be extended to the south to prospect the downward continuation of the Messenger ore body mined above "C" level.

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SPOKANE, WASHINGTON

(2) Drive the Twohy drift on "D" level easterly to develop the Messenger and Leach ore bodies on that level.

(3) Drive "C" level west from a point at the top of the Rynski raise to locate the downward continuation of the Stevens ore body.

(4) If the work outlined above under (3) proves successful, drive a crosscut south from the extreme southwest drift of the Hampshire workings to expose the Stevens ore body on "D" level.

(5) Explore the country from the top of No.3 Raise, "D" level.

(6) Follow ore found in top of No.6 Raise, "D" level.

(7) Crosscut south at the foot of No.6 Raise on "D" level.

(8) Attempt to find downward continuation of Pohlman ore on Leach level.

(9) Explore ore exposed in back of Cameron stope and east end of "A" level.

(10) Continue lower tunnel (No.3) of the Cowboy.

(11) Follow ore found in No.4 Tunnel of the Cowboy.

(12) Continue raise to connect with incline shaft and explore ore at the Lyttle.

(13) Make geophysical survey of the surface of the attractive unexplored portions of the group, to be followed by drilling or underground work, if mineralized area is indicated.

(14) Build a modern flotation mill of 100-ton daily capacity following flow sheet indicated by metallurgical tests.

(15) Install Diesel electric power to operate mill and mines at the mill site, or install Diesel power for driving the mill, and separate Diesel compressor plants at the Queen of Bronze and Cowboy Mines.

(16) Use the surface tram equipment now on the ground, or similar device for transporting the ore from the Queen of Bronze Mine to the mill.

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F. Conclusions: Climatic conditions, accessibility to a good highway, the present low hauling cost to the railway - $2.75 per ton, the prospect of being placed on direct rail transportation to the coast in the future, the large available reserves of profitable ore, together with the large unexplored area included in its holdings, make the Queen of Bronze a very attractive mining venture.

The profit made on the ore contained in the dumps should pay for mill construction and mine equipment on 100-tons-per-day basis, and the profit from the mine reserves can be used for dividends and expansion.

The demand for copper is so great, and the price so well regulated, that there is reasonable assurance of the price remaining about where it is at the present time, 18¢. With 6% mill feed, which is also reasonable to expect from the mine, the price of copper will have to drop below 10¢ per pound before operations will become unprofitable, and at the present price of copper, the ore will return a net profit of over $8.00 per ton.

In the body of this report, I have made no attempt to estimate the probable ultimate daily capacity for production of the property, but feel it is safe to assume that the 100-ton per day mill now to be constructed will be materially enlarged as unexplored territory is developed. However, the present indicated ore reserves assure not only a large return on the money already expended, but an equally profitable return on the expense necessary to bring the property to the contemplated production basis.

Respectfully,

F. Cushing Moore

Spokane, Washington,
December 17, 1929.

REPORT ON THE PROPERTY

of the

QUEEN OF BRONZE MINING CO.

Waldo District, Josephine County, Oregon.

By F. Cushing Moore,
Spokane, Wash.,

May 1929.

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Outline of Report

and

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MINING ENGINEER
SPOKANE, WASHINGTON

REPORT ON THE PROPERTY

of the

QUEEN OF BRONZE MINING CO.

Waldo District, Josephine County, Oregon.

By F. Cushing Moore,
Spokane, Wash.
May 1929.

I. History: The District was first discovered in 1860, but practically no development was undertaken until 1903. A 100-ton smelter was built on the premises under the Tutt ownership in 1904, and operated for five years, handling 20,000 tons of 8% copper ore from Queen of Bronze, besides considerable from the Cowboy, Little and other properties.

From 1910 to 1916 the property was handled by leasers who shipped a few thousand tons of ore to the Tacoma Smelter. In 1916 the property was taken over by the Twohy interests under the management of Roy H. Clarke, who operated it until the drop in the price of copper at the close of 1918. During these three years, 9,992.3 tons were produced from Queen of Bronze, having a net smeltery value of $432,440.88, and 1644.46 tons from the Cowboy with a net smeltery value of $13,562.91. The total net value of the shipments (183 carloads), outside of actual mining costs, was $350,000. The ore shipped ran from 5.16% to 16.33% Copper, with .04 to .44 ounces of Gold per ton.

During the following few years leasers made intermittent shipments from the property.

In 1928 and spring of 1929, John Hampshire of Grants Pass, Oregon, operated the property under lease, netting up to the time of the incorporation of the Queen of Bronze Mining Co. (May 1, 1929), about $33,000 net smeltery value.

The total gross value of all ore shipped to date from Queen of Bronze and other properties in Queen of Bronze ore zone is approximately $1,500,000, with a net operating profit of about $500,000.

II. Economic Value of Ore:

A. Valuable Metals in Ore: The metals of commercial value occurring in the ore, given in the order of their importance are - Copper, Gold and Silver

B. Stability of Metal Prices: Copper averaged about 15¢ per lb. for many years, and during the War it was much higher, but up to recently has been lower. In April of this year it advanced to 24¢ per lb., and now is quoted at 18¢. I feel that it is safe to assume that the price of Copper will stay around 18¢ per lb., and all estimates of values in this report are based on this price.

Gold is paid for by the smelters at $20.00 per ounce; Silver is worth about 55¢ per ounce.

C. Character of Ore- Amenability to Reduction - Ore Values:

1. Ore Minerals: The principal ore mineral is Chalcopyrite, and intimately associated with it is found pyrrhotite, marcasite and pyrite, accompanied by some quartz and calcite. In the upper areas, the chalcopyrite is oxidized to malachite, azurite, cuprite, etc.

2. Sampling: I made no attempt to do any sampling, but from samples taken by Clarke, Stowell and Willson, and from the shipment records, it is safe to estimate that the ore as sorted for shipment will average 10% Copper and .15 ounces per ton in Gold. The Silver is below pay value.

3. Sorting: One ton of shipping product is the resultant of about 5 tons of ore removed from the mine, and some waste is put back as filling underground.

4. Milling: From this remaining four parts another ton of concentrates could be made by milling the rejects from the hand sorting, giving 2 tons of 10% Copper carrying .15 ounces of Gold from the 5 tons of ore mined. Milling tests made by the Federal Mining &

Smelting Co. on samples taken from various parts of the mine and dumps indicated that a saving of 95% of the Copper and 85% of the Gold can be made. Sorting could be done more cheaply and efficiently on a sorting belt than by hand, as is now practiced.

5. Smelting: The most favorable smelter schedule under which shipments can be made is that offered by the A. S. & R. Co.'s plant at Tacoma, Washington, which is as follows:

Pay for 100% of the Copper at the market price, less 2¢ per lb. where Copper is under 14½¢ per lb. and 2½¢ per lb. when Copper is selling for over 14½¢ per lb. Deduct 10 lbs. Copper when Copper is under 14½¢ per lb., and 15 lbs. when over 14½¢ per lb.
Treatmenr $2.50 per ton.
Pay for 95% of the Gold at $20.00 per oz.
" " 95% " " Silver at New York market.
Freight from Waters Creek to Tacoma-

Under $20.00	valuation	$4.10 per ton
$20. to $30.	"	$4.75 " "
$30. to $40.	"	$6.50 " "
$40. to $50.	"	$7.24 " "

D. Topography and Transportation:

1. Location; Altitude; Topography: The property is located on the West slope of a prominent Ridge between Althouse Creek and East Branch of the Illinois River, about one mile southerly from Takelma. 40 mi. ≤ W. of GrantsPas
Elevation at Takelma is 1570 above sea level,
" " Queen of Bronze, 2300 above sea lev.
" " Cowboy 2600 " " "
" " Waldo 2500 " " "

2. Climatic Conditions: The climatic conditions are very mild; snow never remains for any great length of time at the mine. The average rainfall is about 40 inches.

3. Trails; Roads; Railroads; Costs: The Redwood Highway passes through Takelma and a good truck road is available to the Queen of Bronze Mine. A truck road also leads to the foot of the hill below the Cowboy, but the ore is

brought down to that point from the mine by sleds on a steep road or trail about one half mile long. It costs $1.50 per ton to pull ore down from the Cowboy to the truck loading station and $2.75 per ton from there and the Queen of Bronze to the railroad at Waters Creek, 27 miles distant.

E. Power:

No water power has been developed, although I am informed that power can be had on the Klamath River. For the present gasoline engine driven compressors are used. Steam power is too expensive, and therefore out of the question.

When the mill is constructed, either water power will have to be developed, or electric power brought in, or Diesel engine power used. The latter is probably the most feasible, as the initial cost will be very much less and the fuel oil can be handled in drums by the truck as a back-haul from Waters Station. To minimize the expense of handling the oil, the power plant should be located conveniently to the loading bins at the mine, and a storage tank provided at Water's Creek Station. The cost of power generated by Diesel engines compares very favorably with electric power where fuel oil can be delivered cheaply.

F. Ore Extraction:

Ample labor for mining purposes has been available at less wages than prevail in most of the larger mining camps.

Working conditions have been ideal for economic operation, as all the miners have lived in the vicinity of Takelma and go backward forth in privately owned autos. When a large crew is required it will be necessary to operate a company-owned boarding and bunk house adjacent to the mine workings.

Early mining was done by hand, but recently drilling underground has been accomplished by two portable gasoline-driven compressors, one each at the Queen of Bronze and the Cowboy workings.

Ample timber for mining purposes is available on the premises. Water for domestic purposes is in abundance. Supplies are procurable at Grants Pass and there is a small store at Takelma.

To take the greatest advantage of existing conditions a mill should be constructed to handle the low grade ore now put over the dump or left in the mine. It will be advisable to centralize the equipment by locating the mill, power plant and compressor plant at some convenient point and transport the ore to the mill by short tramways. The necessity for this is apparent, as ore will be mined from a number of points along the course of the ore zone, which extends over two miles. The topography of the country lends itself admirably to this arrangement.

Practically all of the ore will be taken from tunnel workings. Most of the ore thus far mined has come from the Queen of Bronze workings, although the Cowboy is now a material producer. No maps are available of the Cowboy workings, but attached hereto is a map showing most of the Queen of Bronze workings.

Extraction costs during the past years' production has quite definitely established a figure of $15.75 per ton, made up as follows:

Mining	$10.00	per ton
Development	3.00	" "
Hauling to Waters Station	2.75	" "
Total extraction cost-	$15.75	per ton.

This figure is based upon one ton of shipping product averaging 10% Copper and .15 oz. Gold per ton. It includes all underground costs, surface sorting, and transportation to the railroad.

Under the present method of sorting one ton of shipping product, four tons of lowgrade ore is put in the dump. When a mill is available, this sorting will be done on a belt, and the reject go directly to the mill, thus materially reducing the cost per ton of shipping product.

G. Overhead Expense:

At the present time overhead expense is very light, consisting of a manager at $300 per month, an office man, and intermittent engineering services.

III. Extent and Permanency of Ore Deposit: Paraphrasing from Stowell's Report.

A. Geology: Ore-Genesis: "Briefly the rocks of the district include both sedimentary and igneous types. The sedimentaries are represented by older slatey rocks, quartzites and limestones, and more recent gravels and sands. Associated with these are the igneous intrusives, practically all being classified as andesite, peridotite, augite, gabbro, diorite, greenstone and serpentine, etc."

"Doubtless the alteration of such rocks as peridotite, pyroxenite and gabbro have produced the serpentines so common in the region, while alteration of the diorite and andesite, etc., accounts for the greenstones. The Copper deposits are associated with one or more of the above named igneous rocks, and so far as is known, have no direct relation to the sedimentaries other than the ore deposits are found in a belt of country extending east of north in the direction which conforms roughly with the strike of the sedimentary rocks in the vicinity."

"The most common rock associated with the ore is serpentine, although the ore is not confined to this rock alone, but is found in adjacent rocks less altered and less basic in character."

The ore bodies are found in ill-defined mineralized zones, in or associated with the serpentine, closely related to the diorite or andesite. The ore occurs in irregular bodies and fragments varying in size from several thousand tons to a few pounds, showing broken angular shapes and slickensided surfaces, crushed into the softer enclosing rock with no apparent order or connection.

In all geological publications, the theory of descending solution with resultant shallow deposits, has been advanced, but recent development has established the fact that the ores do extend to considerable depth. The theory that mineralized waters arising from deep-

seated rock intrusions are responsible for ore deposits conforms more closely to present observed conditions. The evidence is in the mineral associations and character of the ore deposit. Pyrrhotite especially is a mineral found in deposits formed at considerable depth where comparatively high temperature and pressure was the rule. It is found also in magmatic segregations and contact metamorphic deposits, but rarely, if ever, in solution deposits at or near the surface.

The original form of ore bodies is poorly preserved, only the larger masses have withstood the crushing of the rock movements in places less disturbed blocks of ground have protected to some extent the enclosed sulfids, so that one may observe stringers and veins of various sizes running through the country rock.

Briefly stated, intrusions of various rocks caused a system of shear zones to be developed across the country. These shear zones offered a means of circulation for the deep-seated mineralized waters coming from heated igneous masses. When these solutions reached points where conditions of temperature, pressure and chemical composition were favorable, the sulfids were deposited along with the silica and lime. Subsequently some of the rocks were altered to serpentine. In the formation of serpentine, an increase of from 30% to 50% of volume of the rock is involved. This increase in volume was sufficient to account for the movement indicated in the vicinity of the ore bodies, and could easily have accounted for the crushed and dragged condition of the ore.

Erosion has progressed to a point that some of the ore bodies have been exposed on the surface with their consequent development of gossany outcrop. As surface oxidation has progressed more rapidly than the erosion, we find the oxidized copper minerals in the upper horizon of the ore bodies.

All of the ore thus far mined has been taken from comparatively small areas, that at the Queen of Bronze being the most extensive. At this point, the ore bodies are contained within an area of approximately 400 feet in diameter, although other ore bodies have been mined in the same property at a little greater distance.

The Cowboy workings cover a very small area, and those in the Little and Mable are of small dimension.

Undoubtedly many other ore bodies will be

found upon the property, as the territory is not only likely, but iron-stained float and gossan material are found at various points along the line of these other deposits.

The development has consisted entirely of tunnels and raises, and the ground lends itself admirably to this means of development, and deeper tunnels can be driven to advantage. Surface drilling or drilling from underground workings, could be used to advantage in the search for other ore bodies.

B. Ore Reserves: Owing to the irregular character of the ore bodies and the defiency of development work, it is impossible to technically calculate much ore, but from the past history of the property it is safe to assume considerable reserves, although not conclusively proven.

For convenience in estimating ore zones, the Queen of Bronze Mine has been divided into blocks, as is shown on the accompanying map.

Ore Calculations.

Crude ore, mining and sorting costs per ton		$10.00
Trucking to railroad	" "	2.75
Development	" "	3.00
Total-		$15.75

10% Copper; .15 oz. Gold per ton,- Average grade shipping product.

Copper @ 18¢ less 2.5¢ - 15.5¢ net per lb.		
Copper 200 lbs. less 15 lbs. deducted @ 15.5¢ -		$28.67 per ton
Gold .15 oz per ton x 95% @ $20.00 -		2.85 " "
Pay for Copper & Gold		$31.52
Freight	$4.75	
Treatment	2.50	7.25
Crude Ore - Net Smelter Returns - - - - - - - - - -		$24.27
Mining, sorting, development costs		15.75
Net operating profit - - - - - - - - - - - - - -		$ 8.52 per ton

Milling ore made from reject in sorting averaging 2.6 Copper .05 oz. Gold per ton. Ratio of concentration 4 to 1, with milling efficiency of 95% on Copper and 80% on Gold. Resultant concentrate

approximately 10% Copper, .15 oz. Gold per ton,-same ashandsorted ore.

Milling 4 tons @ $1.50 per ton	$6.00
Trucking from mill to railroad	2.75
Cost per ton concentrate at railroad	$8.75
Net smeltery value of 1 ton concentrates	$24.27 per ton
Cost per ton	8.75
Profit per ton of concentrates	$15.52
Profit per ton Crude Ore	8.52

Combined profit with mill and sorting - - - - - -$24.04 per ton of shipping product.

Profit on 5 tons mined - profit on combined process $4.80 per ton
Profit on 5 tons mined - profit on hand sorting alone 1.70 " "

Five tons average ore as mined will produce one ton of shipping ore of 10% Copper and .15 oz. per ton Gold, netting over and above all operating expenses $8.52 per ton, and one ton of Concentrates of 10% Copper and .15 oz. Gold per ton having a net value - taken in connection with the shipping ore - of $15.52 per ton. giving a combined profit of $24.04, or $4.80 per ton mined.

Queen of Bronze:-	Mill Ore	Ratio	Conc.	Profit	Ship. Ore	Profit
Block A - Partiallyproven	10,000	4 to 1	2500	$15.52		$26,300
Block B - Probable	10,000	"	2500	15.52	10M	124,000
Block C - Partiallyproven	5,760	"	1440	15.52	1440	34,617
Cowboy - " "	12,800	"	3200	15.52	3200	76,928
Sulfid Dumps-sampled	15,000	"	3750	15.52	--	58,200
Totals-	53,560		13,390		14,640	$320,045

Shipments available by hand sorting, 14,640 tons - Profit $124,732.
Shipments available, mill in conjunction with hand sorting - 13,390 tons concentrates - Profit 195,313.

Total- $320,045.

At the Queen of Bronze, there are large dumps of oxidized ore of undetermined value, which should be sampled and tested for leaching possibilities. Possibly a small leaching plant, constructed at no great expense, would pay a good profit on the investment.

IV. Extent of Properties: The total area of ground held by the Queen of Bronze Mining Co. is over 880 acres, and comprises a block of ground over two miles long by about a quarter of a mile wide. This block of ground is separated by a group of

claims held by the old Waldo Mining Co., in which the Queen of Bronze has no equity at the present time. There is no reason why this Waldo property should not be attractive for development purposes, as it is in the direct line of the ore zone, but as far as known has no developed ore bodies. See Map No. 1.

Title to the ground is held by patent either as mineral claims or through government filings.

V. Conclusions, Deductions, and Recommendations:

A. Ore Deposits: The rocks in the immediate vicinity of the Queen of Bronze ore bodies are igneous intrusions or flows, principally andesite, gabbro, diorite, etc., altered to serpentine and greenstone. The ore bodies were undoubtedly deposited by ascending mineralized solutions related to the intrusions of igneous rocks at great depth.

Subsequent to the ore deposition, the surface was eroded, placing the ore bodies in the proximity of the surface. Oxidization has been more rapid than erosion, causing oxidized zones above the sulfid ore bodies. Rock movement along shear zones has caused alteration of the basic rocks or alteration of the basic rocks to serpentine may have been responsible for the movement. In any event, the ore bodies are much disturbed and the ore in many places occurs as boulders surrounded by a serpentine coating.

The ore is a chalcopyrite associated with pyrrhotite, marcasite and pyrite. In the oxidized zone are found azurite, malachite, cuprite, etc., Gold and Silver are associated with the Copper, averaging milling ore that will carry 2.6% Copper, .05 oz. Gold and .20 oz. Silver per ton.

The ore bodies thus far developed are within an area of no great dimensions, but the mineralized belt extends through the property in a northerly and southerly direction for over two miles. Undoubtedly the Queen of Bronze ore zone is the northern extension of the mineralized area of Northern California.

From the nature of the deposit and associated minerals, it is safe to assume that the ore bodies will extend to depth.

B. Ore Reserves: The irregular character of the ore bodies is such that it is impossible to make accurate estimates of reserves, but as far as development permits, we may assume that there are about 14,640 tons of shipping ore which will average 10% Copper and .15 oz. Gold per ton, netting $8.52 per ton, at 18¢ Copper, or $124,732. When a mill is available, an additional 13,390 tons of concentrates of similar grade can be shipped, netting $15.52 per ton, or $195,313.00, making a grand total of all estimated reserves of $320,045.00.

Queen of Bronze - Proven and Partially Proven Ore:

	Milling Ore	Concentrates		Profit Per Ton	Shipping Ore	Operating Profit
Block A -	10,000	2500	@	10.52	--	$26,300.
Block C -	5,760	1440	@	15.52	1440	34,617.
Sulfid Dumps	15,000	3750	@	15.52	--	58,200.
Cowboy -	12,800	3200	@	15.52	3200	76,928.
	43,560	10890			4640	$196,045.

Probable Ore:

Block B -	10,000	2500	@	15.52	10000	124,000.
Totals -	53,560	13390			14640	$320,045.

C. Development: The development work on the property consists of several thousand feet of tunnels and workings, a considerable portion of which is closed by caving. The Queen of Bronze is opened to a maximum depth of 200 feet beneath the outcrop, and the Cowboy to about 100 feet.

At both properties additional depth can be had by lower tunnels. This is particularly true at the Cowboy, where the country is very steep.

The large unexplored area, within the property, likely to carry sulfid ore bodies, lends itself admirably to geophysical prospecting, followed by drilling and tunneling.

D. Equipment: At the present time the equipment consists of two small gasoline-driven compressors, one each at the Queen of Bronze and Cowboy workings. For extensive work, the compressors should be larger at

A mill of 50-ton per day capacity should be constructed at a convenient central point, and tramways built from the ore-producing workings to it. A 50-ton flotation mill can be built for $50,000., two tramways for not to exceed $10,000, and surface plants at the mine for about $15,000. A boarding house will also be required if extensive work is undertaken.

E. Recommendations: I recommend that a lower tunnel be driven at the Cowboy Mine to expose the ore body at greater depth. This work can be contracted at about $8.00 per foot, exclusive of rails, pipes and timbers.

A survey should be made at the Queen of Bronze Mine surface to ascertain the feasibility of a lower tunnel.

Sufficient ore is available to justify the construction of a mill and other necessary surface plants commensurate with an energetic exploration and operation of the property.

I also recommend that a complete survey of the property be made, and accurate and complete maps constructed therefrom.

The Waldo property should be acquired, if possible, as it is likely territory and bisects your holdings.

The first development work undertaken at the Queen of Bronze mine should be the driving a raise from the S. E. workings of "D" Tunnel to explore the downward continuation of the "East Stope" ore-body. Also the "Leach" Level shpuld be extended to intersect this Raise and the vacinity thoroughly explored.

Respectfully,

H Cushing Moore

Spokane, Wash.
May, 29, 1929.

REPORT

on

THE ELDER MINING COMPANY.

The Elder Mining Co. is an American corporation, organized in 1906 under the laws of the State of Arizona. Its principal place of business is in Phoenix, the capital of this State, and its administration office is in San Francisco, Union Trust Building.

The corporation has for its object the exploitation of a bed of copper and gold ores, already known to exist for several years in the neighborhood of the village of Waldo in Josephine County, State of Oregon.

The nominal capital of the Elder Mining Co. is $500,000. (Frcs.2500000.-), divided into 25000 shares of $20. (100 francs) each, issued to stockholders or Bearer at their option.

Of these 25000 shares, 10000 still remain in the Treasury, and the 15,000 shares alone participate in the results.

These 15,000 shares are considered sufficient to erect a smelter and to develop the mine. The company ultimately proposes to issue the 10,000 shares, which are still in the Treasury, as funds may be required and in particular with the view to erect an electric converter, for the refining of copper ore and the extraction of precious metals.

The American Company had already started development work, when the catastrophe in San Francisco, shortly followed by the financial stringency in the United States, prevented further operations by the owners.

Reports received on the quality of the mine and the reputation of its promoters were of such a favorable nature, that they justtified the continuance of the work. However, before engaging new capital, it was considered justifiable, to send a French engineer

to the mine, to determine its value, and to map out an economical plan for working the same. Further on, the report in detail, of this engineer will be found, showing the possibilities of this enterprise, and the results, which can be expected.

According to this report a net profit of about $15. per ton for every ton of ore mined, may be counted upon. It is proposed to mine 50 tons per day, meaning a profit of $750. or 3750 francs, or an annual profit of $225,000. or 1,125,000 francs.

Reserving one-half of the above amount for a sinking fund, there will still remain $112,500. equal to 562,500 francs, available for dividends; in other words $7.50 or 37.50 francs for every share of the 15,000 shares issued of the value of $20. or 100 francs each.

ADMINISTRATIVE COUNCIL

California

Jas. O'B. Gunn, President of Mechanics Savings Bank, San Francisco.
T. Waln Morgan Draper, former scholar of the College of Freiberg, former Colonel of the American Militia, Engineer of Mines.
M. S. Wilson, of the firm of Wilson & Wilson, Attorneys, S. F.

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REPORT

on

THE ELDER MINING COMPANY

(Copper & Gold Mine)
Oregon,
U.S.A.

Charged with the examination of the properties of the Elder Mining Co., in the State of Oregon, (U.S.A.) we visited the premises during the month of February, 1909, and the report of our visit follows herewith:

1. LOCATION.

In the U.S.A., near the Coast of the Pacific Ocean, a great deal of gold has for a long time been found, and it is but recently, that attention has been drawn to numerous deposits of copper ore, which extend from north to south in the States of Oregon and California.

Since the discovery of gold mines, half a century ago, copper ore has been found in this section, and wherever auriferous alluvions have been found, the presence of pyrites of iron more or less rich in copper values have been observed.

One may say, that the copper mineral, constitutes more or less or to a great extent, the mother of the gold ore, which has been discovered in an abundance hardly ever equalled.

The mineralized section of this territory may be divided into two zones, forming two parallel belts, running north and south, and corresponding with the mountain range, near the coast of the Pacific Ocean.

The zone most remote from the coast, has at its base, the railroad, with Portland as its northern and Los Angeles as its southern terminals, after having passed through San Francisco. This zone is a very active one, and today shows a great number of

The second zone is nearer the coast, but as the means of communication are almost nil, its mineral deposits are much less exploited. However, at the present time, attention is very strongly drawn to this side, because it is acknowledged, that the mineral deposits found here are very rich. That is to say, that while the ores of the first mentioned zone show an average of between 3 and 6% of copper, those of the second zone show at least 10% and above. The quantity of gold contained per ton, is generally 3 grammes and runs as high as 12.5 grammes.

These favorable conditions have led to searches for ore. To tell the truth, up to the present time attempts have not been very numerous, because the mineral will pay only under certain conditions; to make it pay, the cost of transportation must be reduced; as the cost of installing new improvements involves a considerable outlay, a period of inactivity naturally [illegible] mine. Snow, not very abundant.

2. COMMUNICATIONS.

The mine which we went to examine, and which is called the Elder Mine, is located in the second zone. It is in such a condition that one can readily see, that its operation will prove extremely remunerative, as the ore can be reduced to 40% as I shall show later on.

The property, Elder Mine, is located near the village of Waldo, celebrated in the annals of auriferous alluvions. Close by is the Queen of Bronze Mine, operated for more than three years. At the foor of the ore-bed, there is a good road, passable all the year around. This road from one side, puts Waldo into communication with Grants Pass, distance 41 miles, and on the railroad line between Portland and San Francisco, and from the other side with Crescent City, a harbor on the Pacific Coast, distance 50 miles to the West.

One may therefore easily connect with the station at Grants Pass, which has connection with the great copper smelters of California.

3. COUNTRY.

The country is entirely timbered, scarcely here and there one may see some patches of agriculture on entirely virgin and very fertile soil.

The forest, which covers the country, consists almost entirely of pines, such as pitch pines, sugar pines and others. One sometimes meets giant trees, the wonder of the world, the Sequoia Gigantia.

South of Waldo, the Government has reserved the forest to prevent its destruction, and this measure has had some influence to keep out exploitation.

The climate is very moderate, the winters are very mild, characterized principally by rains. Snow, not very abundant, remains on the ground but a short time, except on the mountains exceeding 1000 metres in height. The summers are dry, particularly in the month of September. The climate is very healthy, there are no mosquitoes and therefore no fever. The population is thinly scattered. Here and there one finds some houses, some occupied by farmers, some by gravel miners and some by prospectors.

Postal and telephone service as well as schools are everywhere extablished, and it is remarkable with what facility one can replenish supplies and find the help of mechanics.

4. GEOLOGY.

The Waldo District is particularly rich in mineral showings. Everywhere, in passing through the country, one finds indications of mineralizztion, consisting of spongy oxyde of iron, in almost all of the ravines made by rivers. In ascending the mountains, one almost invariably again finds the oxyde of iron, which forms the iron cap, immediately under which the copper ore is found.

Geological study of the country shows, that the subsoil is constituted of a great mass of eruptive rock. It is a diorite, almost always transformed, near the surface, into serpentine, and it is near the contact of the serpentine with the diorite that one finds mineral showings. To speak correctly, one does not find oneself in the presence of a true vein, for no one can say that there have not been breaks, which have been filled by metalliferous debris.

One finds oneself rather in the presence of a rock (diorite) covered by another rock (serpentine) and on the surface of these two rocks, there has been some melting process or condensation of the sulphurets contained in one(diorite). These sulphurets are also lodged between the two rocks, forming a true continuous sheet.

The mineral does not only show itself on the side of the mountains, but also in the valleys and [illegible] manner, that one may safely say, that it will surely extend to the hydrostatical level. Some work has already demonstrated that it exists in depth, in fact below the bottom of the valley.

The copper bed of Waldo has already been known for a long time, It is more than half of a century since a Frenchman established himself in the country, and having commenced work with the most rudimentary methods, contented himself with mining the oxidized ore from the surface, and shipping same to Crescent City. This individual thus dug a number of holes which have since permitted to follow up the vein or bed.

After him followed a number of men, who did superficial work but as they had neither money nor ability, these tentative workings resulted in nothing.

Finally, and some time ago, a Mr. Tutt, from Colorado, came to examine these beds, and brought considerable property. He immediately started to open up the Queen of Bronze Mine in a business-like manner. Used to mining, he at once understood that it

was impracticable to ship the crude ore and he built a furnace, capable of treating from 100 to 125 tons of ore per day. Success crowned his enterprise, and even during the economical crisis, he did not stop operations.

5. NATURE OF ORE AND ANALYSIS

The development work done by Mr. Tutt shows in a very exact manner the nature of the ore, its value as well as the manner of treatment necessary to make the ore merchantable.

The ore shows itself in pyrites of iron very rich in copper; it is compact without impurity, other than a little calcite in small veins, and it contains from 10 to 18% of copper.

The neighboring mother-rock is also mineralized, and may perhaps be considered as ore; its extent is variable and diminishes gradually as one leaves the vein.

[illegible] been DRM. This well known fact, resulting from three year's work, is of great importance to the Elder Mine, which is a continuation of the Queen of Bronze Mine, and exactly presents the same ore under the same conditions. The quantity of gold contained in the ore mined by Mr. Tutt varied from 4 to 12.5 grammes per ton.

We have compared the industrial results with analyses made of the ore from the Waldo Mine. Samples of the ore have been taken by us with the greatest of care and in a manner to avoid possible errors and to give an average result to conform to that given by the practical work.

The four samples thus chosen have been analyzed by Messrs. Morris Bros, and give the following results:

The difference in the contents of copper in samples 1 and 2 comes from the fact that the rock was knocked down by dynamite, and that we took the samples in the dark without picking the ore, just as it came to hand. In practice, only the ore would be taken

leaving the sterile rock behind. However, it may be, one may state that the ore on the average contains 14.23% of copper and 4.75 in gold per ton: this is magnificent.

The composition of the ore in itself is remarkable, in as much as it contains no antimony, no arsenic, and that the amount of silica contained, is almost sufficient to make the ore "self-fluxing."

The ore of the Queen of Bronze Mine contains on an average 10% of copper, less than we show. The difference arises from the fact that the work in the Queen of Bronze Mine leaves much to be desired.

In our calculation, for caution's sake, we only consider that the ore contains from 10 to 12% of copper.

6. TREATMENT OF ORE.

The ore being in a compact state, without gangue and impurities, cannot be worked mechanically. Thus Mr. Tutt had a furnace built, which transformed the ore into matte of 40% copper, It would be possible to concentrate the ore still more and thus save in transportation costs, but the refineries for special technical reasons, prefer matte of 40% copper. The furnace installed by Mr. Tutt is called a water jacket, with a capacity of from 100 to 125 tons of ore per day. The fuel used is coke, hauled in wagons from Grants Pass to the mine, the cost of this fuel delivered at the mine is 150 francs ($30). In spite of this high price, Mr. Tutt has made considerable money.

There being considerable wood in the country, and which can be had from outside of the National Reservations, it would be advantageous to use it for fuel, the same as is done in Montana.

The manufacture of matte in this latter state is principally done in reverbatory furnaces, which gives a better result, than water jacket furnace.

This method should therefore be followed. It may be stated, that the Company owns sufficient wood on its own property, to allow of the operation of a furnace for a long time to come.

7. ELDER MINING COMPANY.

This Company possesses concessions on both sides of the Queen of Bronze Mine, extending from North to South, that is to say in the direction and over the mineralized zone. The portion to the South of the Queen of Bronze Mine is called Waldo, and the one to the North "Railroad Sections." In addition to this mineralized territory, the Company also possesses a certain number of claims, on which copper has been found, chromite and auriferous alluvions. The surface, which the Company possesses is as follows:

Waldo Mine	160 acres
Railroad Sections.	320 "
Monarch	7.5 "
New Discovery.	20 "
Accident	20 "
Hortense	20 "
Ida	20 "
Copper Belle	20 "
	578.5 "

It is on the Waldo property that all the work has been done, on the Railroad Sections, and the other claims, only assessment work has been done and such work as the mining laws demand.

As we did, one may follow the mineralized zone from the extreme North of the Railroad Sections to the Southern limits of Waldo; everywhere one sees outcroppings of hematite, which forms the iron cap of the vein. Here and there, one finds excavations, probably half a century old, probably made for the purpose of looking for gold, and which are a valuable guide today.

The presence of the iron cap continues to show itself for several miles to the North of Waldo, where Mr. Tutt and others have acquired the mining rights. One may say, that the territory owned by the Elder Mining Company and Mr. Tutt, is intermixed.

It is in about the center of the Waldo property, that the work has been done. It at first consisted in sinking, which proved that the ore existed under the iron cap. But it was necessary to prove that the ore really extended in depth, which all the geological observations indicated. A tunnel driven about 40 metres below, where the first gallery cut the vein, has again encountered the mineralized zone, exactly at the point indicated by the incline of the vein. One finds, that the mother rock gradually becomes richer and it all at once becomes a mass of pure mineral. I had eight charges of dynamite set off in the side of this tunnel and I could measure 12 feet of compact ore, without other gangue, than very small veins of calcite. From the only passage of this gallery across the mineralized zone, about 65 tons of ore have been taken. From there a chimney, by which one can reach the surface, [illegible] assuring ventilation. The ore, gained by this work, together with the above mentioned, amounts to about 250 tons, of which about 50 tons have been sent to the smelter for assay purposes, and the balance of about 200 tons today remains at the entrance to the gallery.

In the face of this fine showing we advised the immediate resumption of work, which since done, we find on our return fully justifies the good opinion we had of the ore-bed.

8. RICHNESS OF MINE. TONNAGE.

In this Mine there is ore mined, ore in sight, ore in reserve and probable ore.

a. The ore mined consists of about 200 tons, placed on the mine platform at the mouth of the gallery.

b. The ore in sight, is that which has been opened up by the tunnel and by the two galleries running to the right and to the left of the wall of the deposit, from the beginning of the mineralized part.

The ore is in sight for a distance of 9 metres with a height of 40 metres; it experimentally shows a thickness averaging 3m.30 of compact ore. This ore weighs at least 4.000 kilogrammes per cubic metre, the tonnage will then be:

9 x 40x 3.30 x 4 = 4.752 tons

from which must be deducted the ore already taken out, say from 250 to 300 tons. There then remains actual ore in sight, exposed by work, from 4000 to 4500 tons in round numbers.

c. The ore in reserve is that which exists in the whole mineralized zone of the Waldo Section, from the top of the iron cap to the level of the actual gallery.

The length of the zone is 800 metres, the average known height is 40 metres. As to thichness, instead of taking this at 3 m.30, the thickness in the present workings, we shall figure it at but 3m.

Under these conditions, the calculation is as follows:

800 x 40 x 3 x 4 = 384,000 tons,

but to be conservative, we shall take but half of this amount, and say that the tonnage in reserve amounts to about 192,000 tons of ore.

d. The probable ore, is that which is likely to exist below the actual or present level, and the ore which may be found on the other side of the Queen of Bronze Mine, on the Railroad Sections.

One may figure that the entire Waldo ore-bed, from below the present level, could be worked to a height of 130 metres.

Accepting the same characteristics, the result in tonnage would then be:

800 x 130 x 3 x 4 = 1,248,000 tons,

of this we should take out but 1/3. say 416,000 tons

As to the Railroad Sections, we shall not take them into account, as they never have been explored to any depth; but by

virtue of their close proximity to the Queen of Bronze Mine, we must not forget that very likely there will be as much ore found there as at Waldo . In fact, the superintendant of the Queen of Bronze Mine, after having asked the permission of the Elder Mining Co., for his own satisfaction, made some investigations on the Railroad Sections near the Queen of Bronze Mine, and in his opinion, there are good copper ores in the neighborhood.

To sum.up, there are:

Ore on dump	200	tons
Ore in sight	4400	"
Ore probable	192000	"
Ore in reserve	419000	"
	612000	"

The conclusion from these very moderate estimates is, that there is a great quantity of ore in the Elder Mining Co's properties, sufficient to [illegible] years.

9. WORK.

The mining of the ore is very simple. The tunnel on the side of the hill cuts the vein on either side of which a gallery has been run in such a manner that four ore shoots (?) can be installed.

The work of mining presents absolutely no difficulties. Timbering is not necessary, as the rock is very solid. Virtually during the month of April, the output should be 25 tons per day, and that simply in the work of preparing the ore-shoots. End of May everything will be ready for a daily output of 50 tons. The output could be forced still more by running a gallery above the present working level, but this is not necessary at present. The mined ore must be lowered below the mouth of the tunnel, to a place, where it is proposed to build the furnace. A road is now being built.

The site of the furnace has been selected, with the view to ultimately allow of the development of the entire mine, by driving a tunnel which will open up a level with the furnace, this tunnel it will not be necessary to build just yet, because there certainly is sufficient ore above the present level to insure work for long months to come.

The furnace will be a reverberatory one, heated with wood, with a daily capacity of 50 tons, and a rock crusher.

Definite plans are now under consideration, and construction work can begin immediately.

10. NET COST.

Considering that it will be necessary to produce a copper matte for the market of about 40%, we shall give the details of the net cost of this material. This price includes the cost of mining of the ore, its transportation to the refinery:

a MINING. The ton of ore delivered at the furnace, together with carting and supervision, will cost $2.00 for a daily output of 50 tons.

b. The matte produced in a reverberatory furnace, output 50 tons per day, from the time of the entrance of the raw material until it is taken out as ingots, will cost for labor, fuel motive power, analyses and supervision, $2.44 per gross ton treated.

c. Transportation of the matte. The transportation of a ton of matte from the mine to the Station at Grants Pass by wagon will cost $5.25; the transportation by rail from Grants Pass to the refinery, will average $3.25; the total cost of transportation of a ton of matte will be $8.50.

Fifty tons of ore treated daily for conversion into matte of 40% will in proportion to the richness of the ore, yield a larger quantity of material, that is to say:

50 tons of ore at 10% will yield 12.5 tons of matte at 40%.
50 " " " " 12% " " 15 " " " " "

The transportation of these 50 tons of ore, converted into matte, will be in proportion to the amount of copper contained.

The result is that a ton of gross ore converted into matte will cost for transportation as follows:

Per ton of gross ore at 10%	$2.13
" " " " " 12%	$2.55

d. REFINING. The cost of refining will most likely be $3.00 per ton of matte at 40%. To make sure, we shall figure that the cost of treatment of a ton of matte at 40% will be the same as the cost of refining a ton of ore, say $6.00.

In reducing to units the ton of ore mined we find the cost

of refining a ton of ore at 10%	$1.50
" " " " " " 12%	$1.80

To sum up, the following table will show the net cost of a ton of ore, sold and delivered at the refinery:

	Ore at 10%	Ore at 12%
Mining	$2.00	$2.00
Manufacture of Matte	2.44	2.44
Transportation of Matte	2.13	2.55
Refining of Matte	1.50	1.80
	$8.07	$8.79

e. GENERAL EXPENSES. As the work will be in full activity, we believe that the general expenses may be placed at $4,250. per month. In these expenses are included the cost of management and accounting, the preparatory work, the axes, journeys, administrative expenses, etc.

For a monthly output of 1500 tons, this would mean $2.82 per ton. From this follows, that the total cost of ore will be:

Ore at 10%	$10.89
" " 12%	11.61

II. SALE.

A ton of matte sells by the units contained. Usually the buyer deducts a unit and one-half, but to provide for small expenses we shall deduct two units, and figure that the matte of 40% contains but 38%, for which we receive payment.

The price of copper in the U.S. is not fixed, as in Europe, by the ton but by the pound. The American ton contains 2000 pounds, from which follows, that a ton of matte contains 760 pounds of merchantable copper.

Taking the price of copper at lowest it has yet reached during the late crisis, say 12 cents per lb., corresponding to 54 lbs sterling per ton, the ton of matte, delivered at the refinery, will have a value of

"	760	$0.12	$91.20

In applying the price to the ton of ore, we find that the ore is worth of possible business opportunities and profits.

Per ton at 10%	$22.80
" " " 12%	27.36

As the ore contains gold, say $2.50 per ton, the total value of the ore will be:

	Ore at 10%	Ore at 12%
Copper	$22.80	$27.36
Gold	2.50	2.50
	$25.30	$29.86

This is the merchantable value of a ton of ore.

12. EXPENSES OF DEVELOPMENT AND ROLLING FUND.

What is to follow is subject to the mine being put into full operation, that is to say:

1st. The mining of 50 tons of ore per day.

2nd. The treatment of ore for matte.

1st. Mining of 50 tons of ore per day. This production should be attained very soon and before the furnace is finished. It is also necessary, during this work, that the production be restricted and that ore only should be taken from the two galleries in the direction which they are now being advanced.

For this work, as well as for putting the buildings, etc. in order, $17,500. should be provided for, to be distributed during the months of March, April, May, June, July and August.

2. INSTALLATION OF FURNACE. The Furnace, with all the appurtenances and a road to the entrance of the gallery will cost at most $35,000. This amount cannot be divided into monthly payments, at least one-third must be at once available for the ordering of material and mechanical parts.

To these amounts, representing the expenses, there must be added a rolling fund, necessary to keep the work going for about a month and a half, and to allow of the payment for merchandise, the payment of which is customary , one month after delivery. Returning to above figures, it will be seen that $30,000. must be provided for, but this sum may be materially reduced by taking advantage of possible

To sum up, it is necessary to place at the disposal of the mine.

1st.	For the monthly expenses of March April, May, June, July & August	$17,500.
2nd.	For expenses to construct furnace and accessories	35,000.
	Total expenses	$52,500.

and hereto must be added a maximum rolling fund of $30,000.

Under these conditions it is indisputable that the mine show the results which we have indicated.

It is certain that this mine will show results as good, if not better, than the California mines,- Mammoth, Balaklala, Afterthought, Penn Chemical Co., etc.

13. BENEFITS OR PROFITS.

The profit from a ton of ore is shown by the preceding calculations by taking the difference between the merchantable value and the net cost.

	Copper at 10%	Copper at 12%
Merchantable value	$25.30	$29.86
Net cost	10.89	11.61
Difference	$14.41	$18.25

From the above figures, we deduct 5% to provide for loss in the treatment of ore, for loss in transportation of ore and matte and for general expenses not foreseen. Thus finally the net profit of a ton of ore refined will be:

Ore at 10%	$13.69
" " 12%	17.34

This profit, in our opinion, is certainly obtainable, for all the costs and expenses have been figured excessively high. The profit to be expected may therefore be placed at from $13.00 to $17.00 per ton of ore.

The price of copper, at the mine, with ore at 10% would be probably no longer pay a profit, is 6-1/4 cents per lb., or 30 pounds sterling per ton.

CONCLUSIONS.

We conclude by saying:

1st. The properties of the Elder Mining Co., throughout their entire length show a rich and strong ore-body, the ore being of a compact mixture of pyrite of iron and chalcopyrite, from three to four metres in thickness.

2nd. The average contents in copper are 14% and there are 4.75 grammes of gold per ton.

3rd. The work done on the territory of the Elder Mining Co. as well as that done on adjoining property, has proven that there is a large quantity of ore, sufficient to insure a daily output of 50 tons, for more than 30 years to come.

4th. Mining is of the easiest, because it is done on the side of a hill under most excellent conditions, without timbering nor waste rock.

5th. The sale of the ore is subject to its reduction to matte of 40% as it is done in the adjoining mine.

6th. It is necessary to expend about $52,500. to assure the mining and treatment of about 50 tons of ore per day, and this will leave a net profit of from $13.00 to $18.00 per ton, the price of copper being 54 pounds sterling per ton.

7th. In six months the improvements can be completed, and the product be ready for market.

8th. The mine, in every respect, is in most excellent condition, and a brilliant future may be predicted, like that of similar enterprises, started not long since, in the Northern part of California.

Waldo, February, 1909.

Paris, April, 25, 1909.

(Signed) A. FONVILLE,
Engineer, E.C.P.

COPPER GOLD

Queen *of* Bronze Mining Company



A mining enterprise based on a producing and rapidly expanding property in Southwestern Oregon with early dividend possibilities



Queen of Bronze Mining Company

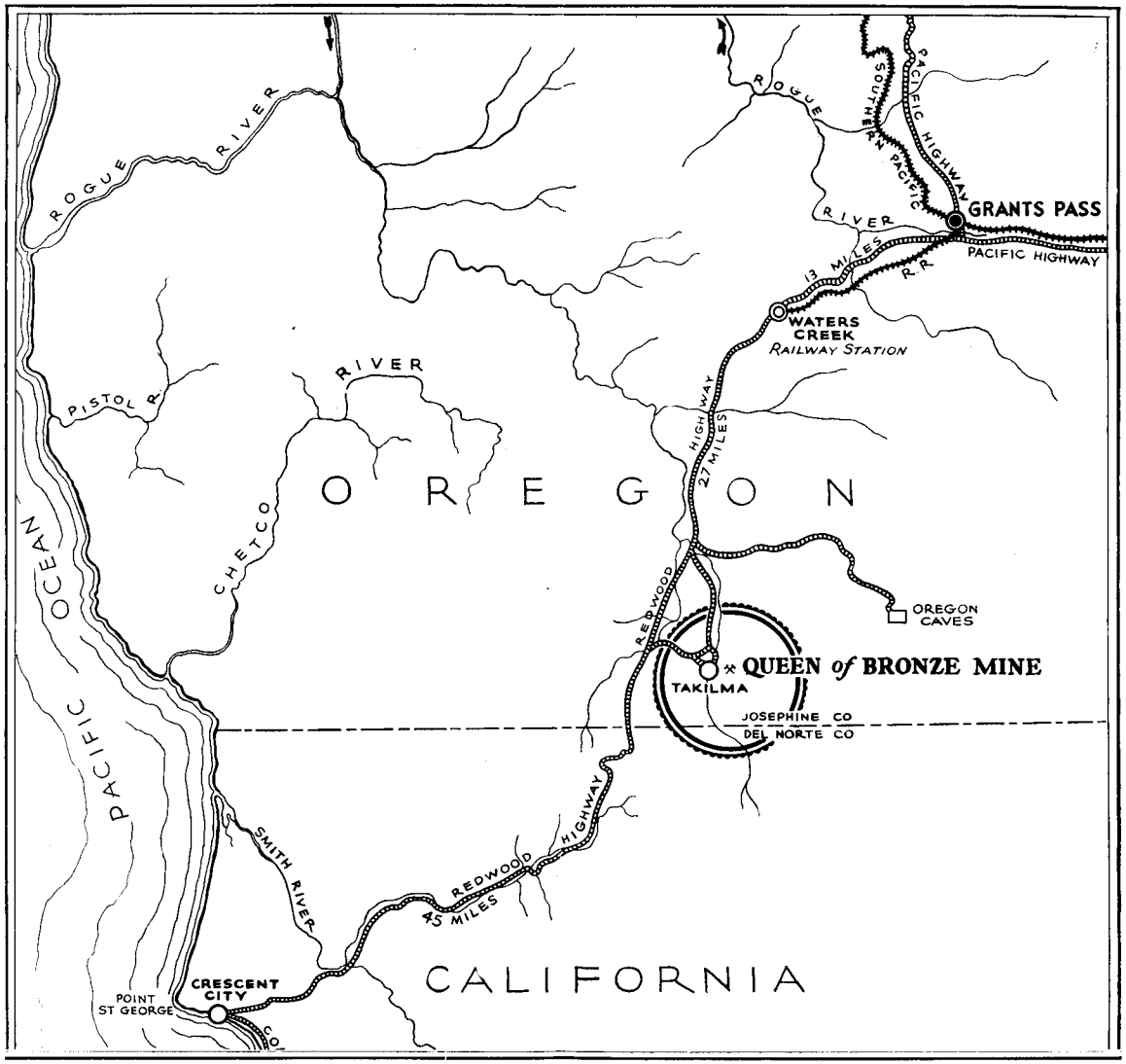
AREA, TITLE AND LOCATION The Queen of Bronze Mining Company, organized in 1929, owns undisputed title, free and clear of any indebtedness, to the producing COPPER-GOLD mines known as the "Queen," "Cowboy" and "East Cowboy," and a large area contiguous thereto, consisting of 882 acres of patented ground, and 220 acres not yet patented. It also holds a lease and bond on the adjoining Waldo group (a former important producer) consisting of 480 acres. This entire holding, aggregating 1582 acres, covers a mineralized area extending over a distance of two miles, located near the town of Takilma, Josephine County, Oregon, 40 miles southwest of Grants Pass, and adjacent to the Redwood Highway.

HISTORY AND PRODUCTION RECORD Operations were first commenced in 1904, and since conducted intermittently by owners and leasers, who concentrated entirely on the production of hand-sorted ore, from ore bodies most easily accessible, without attempting any systematic development work, or equipping of the property with a concentrator for treatment of the lower grade ores.

According to well authenticated records, the total production derived from hand-sorted ore by former operators aggregated approximately $1,500,000, all of which was mined from a small area to no greater depth than 250 feet at any point on the properties, at which horizon the ore is entirely in sulphides.



QUEEN MINE—Showing Sulphide Ore Dumps

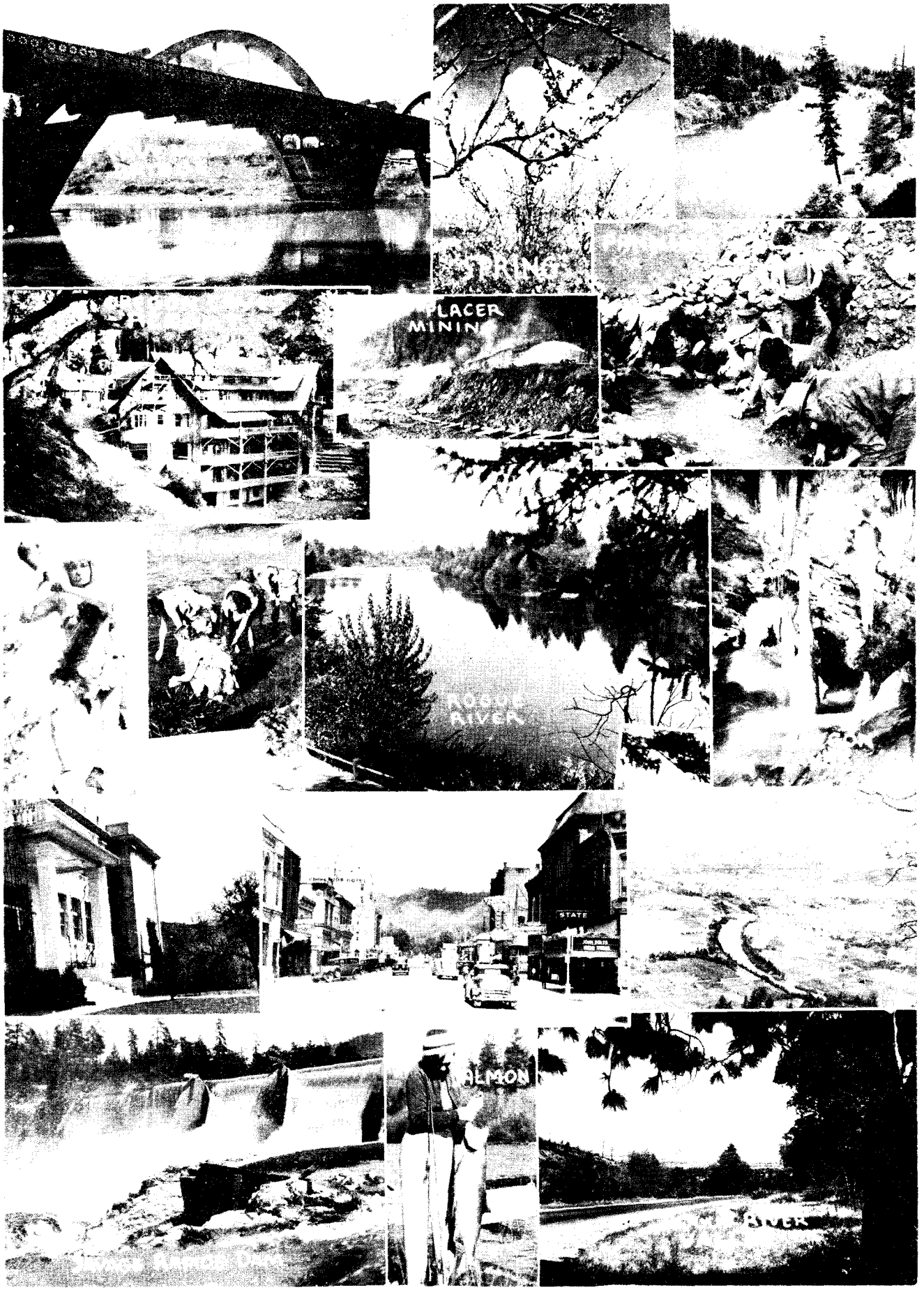


Map Showing Location and Transportation Facilities

ORE EXPANSION Expert opinion, based on the geology of the district, classifies the ores as of deep-seated origin, which increases the potential value of this property enormously. Recent exploration of the ore bodies is only now beginning to reveal their strength, continuity and permanence at increased depth. Lateral exploration is also disclosing new ore bodies beyond the confines of the area hitherto mined.

ORE RESERVES NET VALUE AND POSSIBLE PROFITS According to report by F. Cushing Moore, E. M., made in December, 1929, the ore reserves then developed and partially developed were given as *187,000 TONS;*

Grants Pass Set in Heart of Scenic Wonderland



COURIER LITHO, GRANTS PASS, ORE.

Ref. No. A-202

REPORT

ON

SINK AND FLOAT TESTS

FOR

QUEEN OF BRONZE MINE
Takilma, Oregon

by

The Sink and Float Corporation
Empire State Building
New York 1, N. Y.

October 24, 1947

SINK AND FLOAT TESTS

Ref. No. A-202

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Summary 1
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Method of Separation 3
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SINK AND FLOAT TESTS

Summary

Name of Client: E. R. Waite

Name of Mine: Queen of Bronze mine
Takilma, Oregon

Grade of Sample: Gold - 0.06 oz. per ton
Copper - 2.92 per cent

Objectives

1. To test the amenability of the sample to separation by the H. H. Sink and Float process, with the object of producing a shipping-grade product by rejecting uneconomic waste from the ore.

2. To determine the optimum operating conditions and metallurgical results obtainable by such treatment.

Conclusions

1. The sample of Queen of Bronze copper-gold ore tested proved to be amenable to separation by the H. H. Sink and Float process.

2. A product of shipping-grade was made, as below:

Weight of shipping product:	57.00 per cent
Assays of shipping product:	0.091 oz. Au per ton
	4.61 per cent Cu

3. This was accomplished by rejecting 43.00 per cent of the total ore as an uneconomic waste product, as below:

Weight of reject:	43.00 per cent
Assay of reject:	0.017 oz. Au per ton
	0.68 per cent Cu

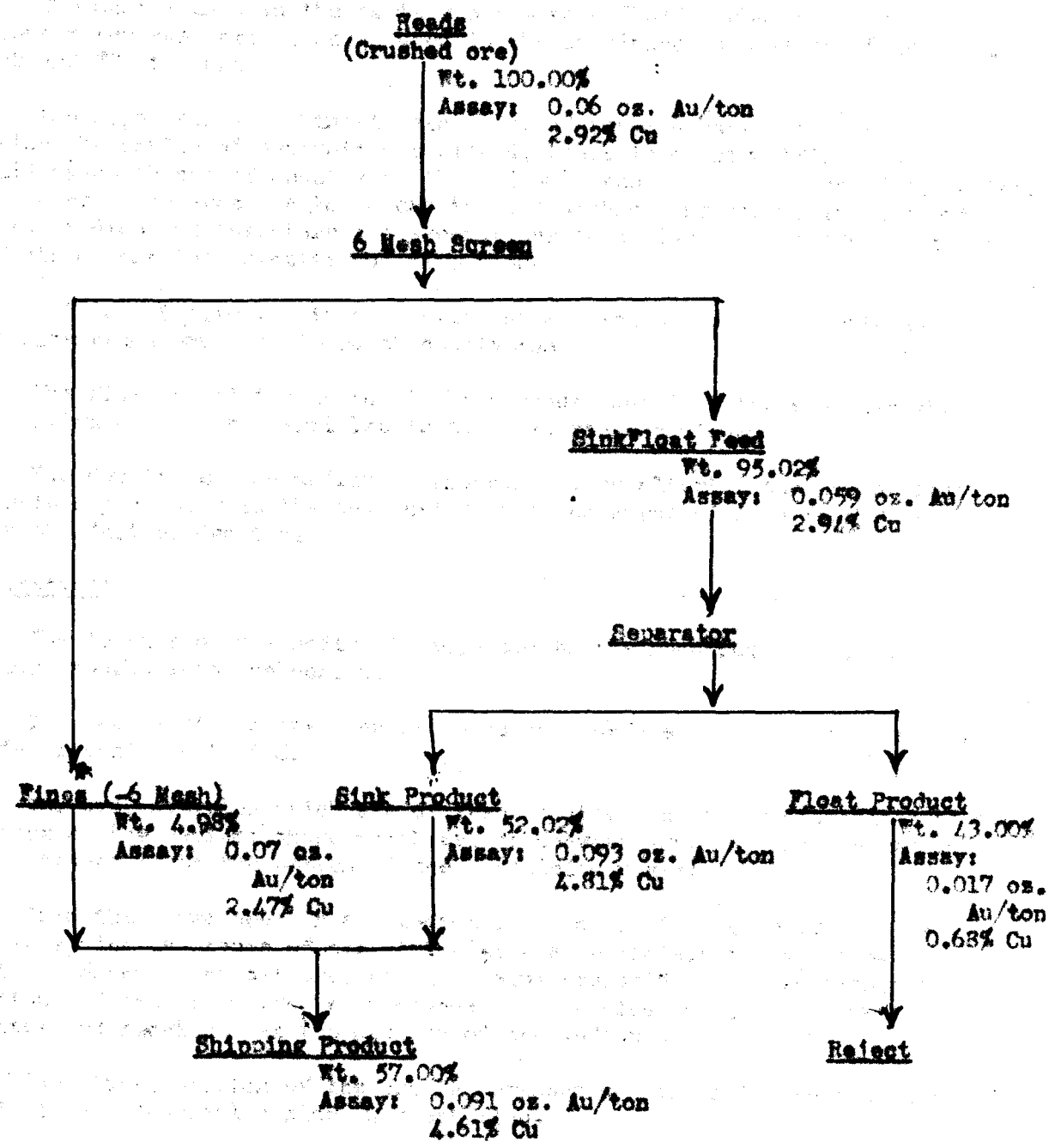
4. The recoveries obtained in the above separation were high, as follows:

Gold	- 87.51 per cent
Copper	- 90.02 per cent

5. Crushing to a size of 1½ inches is ample for efficient SinkFloat treatment of the ore.

SINK AND FLOAT TESTS

FLOWSHEET



Method of Sink-Float Separation

Medium Used:

The medium used in the tests was the same fluid, substantially stable galena-water suspension, as is used in the continuous operation of an H. H. Sink and Float plant.

The high degree of kinetic stability of the suspension is such that, during the period of separation employed, there is no stratification (which could cause change of density of the medium) even in the absence of agitation. This permits separations to be carried out without any disturbing currents in the medium and therefore the separations take place in accordance with the effects of particle density and time alone.

Sufficient time is given to allow separation of the ore particles in the size range employed to reach completion.

The fluidity of the medium is high enough not to interfere with the separation of the ore particles in the size range employed.

The density of the medium is controlled, by altering the proportions of galena and water in the suspension, with an error of less than 0.01 from the desired density.

Separation:

The density of the medium is adjusted to the required figure and the medium brought into suspension.

The ore particles are then fed onto the surface of the medium and complete immersion effected.

The ore particles which are of less gravity than the medium float at the surface of the medium; those which are of heavier gravity than the medium sink through it to the bottom of the containing vessel.

When they have been in the medium long enough for separation to be complete, the two types of ore particles are separately removed from the medium. Known as "float product" and "sink product", they constitute those portions of the ore which, when crushed to the size used, are respectively lighter and heavier than the density of the medium used.

The finest portion of the ore is screened out and not otherwise treated, as it is uneconomically slow in separation.

This method of testing has been found from operating experience to give metallurgical results which are duplicated or bettered in full-scale commercial operation of H. H. Sink and Float plants on ores of which the samples tested are representative.

Description of Sample

Amount Received:

Approximately 260 lb. was received as a sample.

Grade of Sample:

Gold: 0.06 ounces per ton
Copper: 2.92 per cent

Size Analysis:

The sample was given a size analysis before and after crushing prior to SinkFloat testing, as below:

Size	Wt. % Before Crushing	Wt. % After Crushing
+ 3"	14.37	
-3" + 2"	34.11	
-2" + 1½"	16.99	
-1½" + 1¼"	6.76	22.40
-1¼" + 1"	7.40	27.73
-1" + 3/4"	8.49	21.69
-3/4" + ½"	3.97	9.18
-½" + 3/8"	2.07	5.96
-3/8" + ¼"	1.87	3.77
-¼" + 4 M	1.20	2.67
-4 M + 6 M	0.76	1.66
-6 Mesh	2.01	4.94
	100.00	100.00

Mineralogical Description:

Economic Minerals: Chalcopyrite, gold.

Gangue Minerals: Pyrite.

Country Rock: Dark-colored, fine-grained igneous rock of the basalt family, with some calcareous felsites.

Description of Tests

Preparation of Sample for Testing:

The sample was divided into portions representative as to grade and size in the manner described below:

It was divided into four size fractions of nearly equal amount. Each size fraction was split by coning and quartering into a number of equal parts. A recombination was made by adding together one of these parts from each size fraction. This process of recombination was repeated until all fractions were used up. In this way each recombination was a smaller sample representing the original one and each could be used in a series of parallel tests.

Crushing:

The sample was crushed in a jaw crusher of the Blake type, with an open setting of 1½ inches.

Screening:

A size analysis was made on the sample before and after crushing (see page 4).

The minus 6-mesh material was screened out and was not separated by SinkFloat. The plus 6-mesh ore was then subjected to separation.

Separation:

Several small-scale preliminary tests showed that optimum separation should occur in a medium of density 2.95. The sample was then separated, and the following products resulted:

1. A float product containing all particles of density less than 2.95;
2. A sink product containing all particles of density greater than 2.95;
3. A "fines" portion containing all minus 6-mesh material, and which was not separated by SinkFloat.

Assaying:

The above three products of testing were dried, weighed, and assayed for gold and copper.

Tabulation:

Weight percentages and recoveries of various products were calculated. All results of testing are tabulated in Table I, "Separation of -1½" + 6 Mesh Ore in a Medium of Density 2.95", page 6.

/s/ W. H. Love
The Sink and Float Corporation

SINK AND FLOAT TESTS

TABLE I

SEPARATION OF -1½" + 6 MESH ORE IN A MEDIUM OF DENSITY 2.95

PRODUCT	WT. %	ASSAYS		% METAL DISTRIBUTION	
		Au	Cu	Au	Cu
Heads (Calc.)	100.00	0.06	2.92	100.00	100.00
Fines (-6 Mesh)	4.98	0.07	2.47	5.88	4.22
S. F. Feed	95.02	0.059	2.94	94.12	95.78
Float (Reject)	43.00	0.017	0.68	12.49	9.98
Sink	52.02	0.093	4.81	81.63	85.80
Reject	43.00	0.017	0.68	12.49	9.98
Product*	57.00	0.091	4.61	87.51	90.02

*Sink product plus fines.

AMERICAN CYANAMID COMPANY
Ore Dressing Laboratory

T.I. 242
Problem 463

December 1, 1947

Heavy-Media Separation Tests on Gold-Copper Ore
From Queen of Bronze Mine Submitted by Mr. E.R. Waite.

(This report is based on metallurgical results obtained in the ore dressing laboratory of American Cyanamid Company on a sample of material submitted by the subject company, and all recommendations and opinions expressed herein apply only to the treatment of material conforming to the sample submitted.)

Introduction:

Mr. E. R. Waite of Grants Pass, Oregon shipped a 500 pound sample of dump ore from the Queen of Bronze Mine, Takilma, Oregon, to our Mineral Dressing Laboratory for Heavy-Media Separation tests. Mr. Waite advised that a substantial tonnage was available but that it was too low grade to ship to a smelter. It was his thought that concentration by Heavy-Media would produce a shipping grade concentrate.

Object of Investigation:

We were asked to determine the amenability of the sample of dump ore to concentration by Heavy-Media Separation.

Description of Shipment:

Four kegs having a gross weight of 500 pounds were received at the Mineral Dressing Laboratory in good condition on October 7, 1947. The sample consisted of pieces ranging in size from 6" to fines.

Preparation of Sample:

The contents of all four kegs were screened in 1". The screen oversize was crushed to pass 1" and the entire sample was then screened on 10 mesh. The minus 1" plus 10 mesh ore was mixed and representative 50 pound charges were cut out for testing. The minus 10 mesh fraction was weighed and sampled for assay. The above treatment resulted in the following weight distribution.

Table I

Weight Distribution

Prepared Ore

Product	% Weight
Original	100.00
-1" +10-mesh	89.95
-10-mesh	10.05

Heavy-Media Test Procedure:

Each 50 pound charge of -1" +10-mesh ore was wet screened on 10 mesh to remove adhering slime. The wet minus 10 mesh fraction was dried, weighed and assayed. The -1" +10-mesh ore was then fed continuously at the rate of 50 pounds per hour to a 20" diameter Heavy-Media Separatory cone. Preliminary examination of the ore had indicated the optimum specific gravity of separation to be slightly below 3.00. The Heavy-Media Separatory cone was therefore charged with a medium consisting of a mixture of finely ground ferrosilicon and magnetite suspended in water. The specific gravity of the medium was adjusted to 3.00 and the feed was started.

The ore particles having a specific gravity lower than 3.00 floated on the surface of the medium and were discharged from the cone over a weir. The heavier fraction of the ore sank and was discharged from the ore continuously by means of an air lift. Both products were drained free of medium on a vibrating screen. The drained medium was returned to the cone. The separated ore products were washed free of medium on the lower part of the screen. Each product was dried, weighed and assayed.

Separating mediums having specific gravities of 2.93 and 2.88 were employed in subsequent tests.

The metallurgical results of these tests are presented in Tables II, III and IV.

AMERICAN CYANAMID COMPANY
Ore Dressing Laboratory

Table No. II
Test No. 1

Tech. Invest. No. 242 - 463
Sample -1"

PRODUCT	% WEIGHT		ASSAYS	
	In Size	In Total	oz/ton Au	% Cu
Ore		100.00	0.044	2.41
Total -10 mesh		10.25	0.050	1.98
Cone Feed -1" +10 mesh	100.00	89.75	0.043	2.46
Sink in 3.00	32.58	29.24	0.090	5.26
Float on 3.00	67.42	60.51	0.020	1.11

Remarks:

% DISTRIBUTION

PRODUCT	In Size		In Total	
	Au	Cu	Au	Cu
Ore			100.00	100.00
Total -10 mesh			11.78	8.41
Cone Feed -1" +10 mesh	100.00	100.00	88.22	91.59
Sink in 3.00	68.51	69.60	60.44	63.75
Float on 3.00	31.49	30.40	27.78	27.84

AMERICAN CYANAMID COMPANY
Ore Dressing Laboratory

Table No. XII Tech. Invest. No. 242 - 463

Test No. 2 Sample -1"

PRODUCT	% WEIGHT		ASSAYS	
	In Size	In Total	oz/ton Au	% Cu
Ore		100.00	0.050	2.54
Total -10 mesh		10.25	0.050	1.98
Cone Feed -1" +10 mesh	100.00	89.75	0.050	2.61
Sink in 2.93	53.41	47.94	0.076	4.11
Float on 2.93	46.59	41.81	0.020	0.88

Remarks:

% DISTRIBUTION

PRODUCT	In Size		In Total	
	Au	Cu	Au	Cu
Ore			100.00	100.00
Total -10 mesh			10.28	7.99
Cone Feed -1" +10 mesh	100.00	100.00	89.72	92.01
Sink in 2.93	81.34	84.27	72.98	77.53
Float on 2.93	18.66	15.73	16.74	14.48

AMERICAN CYANAMID COMPANY
Ore Dressing Laboratory

Table No. IV Tech. Invest. No. 242 - 463

Test No. 3 Sample -1"

PRODUCT	% WEIGHT		ASSAYS	
	In Size	In Total	oz/ton Au	% Cu
Ore		100.00	0.045	2.30
Total -10 mesh		10.25	0.050	1.98
Cone Feed -1" +10 mesh	100.00	89.75	0.044	2.33
Sink in 2.88	65.23	58.54	0.060	3.18
Float on 2.88	34.77	31.21	0.015	0.74

Remarks:

% DISTRIBUTION

PRODUCT	In Size		In Total	
	Au	Cu	Au	Cu
Ore			100.00	100.00
Total -10 mesh			11.42	8.84
Cone Feed -1" +10 mesh	100.00	100.00	88.58	91.16
Sink in 2.88	88.24	88.96	78.17	81.09
Float on 2.88	11.76	11.04	10.41	10.07

Discussion of Results:

The metallurgical results given in Tables II, III and IV show that as the specific gravity of separation was decreased the ratio of concentration decreased and metal recoveries increased.

The highest grade concentrate was made in Test 1 in which a medium having a specific gravity of 3.00 was employed. The sink product amounted to 29.24% weight of the original ore and assayed 0.09 oz. Au per ton and 5.26% Cu. Recoveries of 60.44% of the gold and 63.75% of the copper contained in original ore were made.

Conclusion:

Heavy-Media Separation tests conducted on a sample of dump ore from the Queen of Bronze Mine have shown the amenability of this material to concentration. The optimum specific gravity of separation will depend largely on the cost of marketing the concentrate. Heavy-Media Separation at 3.00 specific gravity produced the highest grade concentrate. If shipping costs are moderate the use of a lower specific gravity of separation might be indicated so as to obtain higher metal recoveries even though at the expense of concentrate grade.

AMERICAN CYANAMID COMPANY

/s/ R. A. Pickens

Mineral Dressing Laboratory

INVENTORY OF QUEEN OF BRONZE AREA MAPS

OBTAINED FROM U.S. BUREAU OF MINES

No.	Map title	Scale	Date
1	Geologic map of Takilma-Waldo Copper Belt (Fig. 2 of USBM RI 5187, at original scale)	1" = 1,320'	1951
2	Topography, geology, and assays of Cowboy mine	1" = 40'	1951
3	Cowboy mine (geologic map of mine workings)	1" = 40'	1951
4	Plan of surface, Cowboy mine (Fig. 15 of USBM RI 5187, at original scale)	1" = 40'	1951
5	Plan of surface, Queen of Bronze mine (Fig. 5 of USBM RI 5187, at original scale)	1" = 40'	1951
6	South workings, Queen of Bronze mine (plan view)	1" = 20'	?
7	Lyttle mine (plan and profile)	1" = 40'	1929
8	Lyttle mine (profiles of projected tunnels)	1" = 100', 40'	1902
9	Lyttle mine (plan and profile)	1" = 20'	1902
10	Plan of surface, Lyttle mine (Fig. 14 of USBM RI 5187, at original scale)	1" = 40'	1951
11	North workings, Queen of Bronze mine (plan view)	1" = 20'	1916
12	Waldo Copper Mine (plan and profile)	1" = 200', 10'	1901

MCP
mineral commodity profiles

UPDATE
JULY 1977 ISSU

COBALT



BUREAU OF MINES ★ UNITED STATES DEPARTMENT OF THE INTERIOR
OCTOBER 1979

ERRATA

The following reference list has been revised to conform with reference numbers that appear in the text of the Mineral Commodity Profile, "Cobalt" which was mailed recently. Please insert this errata page in your copy of "Cobalt."

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COBALT

MINERAL COMMODITY PROFILES
October 1979

This current report on cobalt has been prepared by the Bureau of Mines, U.S. Department of the Interior to:

1. Provide the latest available data and information on cobalt.

2. Invite comment, revisions, or additional information on the subject.

Please direct communications to the author:

Scott F. Sibley
Bureau of Mines
2401 E. St. NW.
Washington, D.C. 20241
Telephone number (202) 634-1024



UNITED STATES DEPARTMENT OF THE INTERIOR

A single copy of this publication is available without charge from: Publications Distribution Branch, Bureau of Mines, 4800 Forbes Avenue, Pittsburgh, PA 15213

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Bibliography. p.
1. Cobalt industry--United States. 2. Cobalt--Statistics. I. United States. Bureau of Mines. II. Title. III. Series.
HD9539.C463U57 1979 338.2'7'48 79-21719

COBALT

By Scott F. Sibley[1]

Cobalt, a little-known but strategic metal, has been developed through the centuries from an obscure coloring additive into an essential element in many alloys and an important ingredient in chemical compounds. Moreover, cobalt is one of several vital alloying elements in the aerospace and electrical-product industries. In most of its alloying applications, cobalt imparts essential qualities such as heat resistance, high strength, wear resistance, and superior magnetic properties. Major uses include cutting tools, jet engine parts, electrical devices, permanent magnets, catalysts, and pigments and dryers for paints and allied products.

The basic U.S. problem regarding cobalt is excessive dependence (95 percent) on foreign sources of supply, combined with concentration of the higher grade deposits in only a few areas of the world. For example, although Zaire supplied the United States with 42 percent of cobalt imports in 1978, and Belgium supplied 18 percent, all of the Belgian material originated in Zaire. Zambia, adjacent to Zaire, supplied 11 percent of the U.S. imports. In addition, Botswana and the Republic of South Africa each supplied about 2.0 percent, raising the total from a single geographic area to 75 percent for 1978. In some years, this percentage has been even higher. To insure an adequate supply in case of emergency, sales of then—considered excess cobalt from the national stockpile were halted in September 1976, and on October 1 a new stockpile goal of 85.415 million pounds was announced by the Federal Preparedness Agency (FPA), of the General Services Administration (GSA). The effect of the discontinuance of sales of cobalt from Government stockpiles in 1976 was a drawdown of industry stocks in 1977. When demand turned sharply upward in 1978, producers were unable to adjust by increasing their output, industry stocks were quickly depleted, and most consumers were put on allocation by the major supplier. Although the United States is almost entirely dependent on directly imported material, the development of resources, such as the cobalt-bearing manganese nodules on the ocean floor and additional land-based copper and nickel deposits, may substantially reduce domestic dependence over the next 20 years.

If another serious domestic supply problem should develop, it is doubtful that it will result from either a lack of world resources or technologic difficulties. The potential problem lies in the existence of relatively insecure supplies owing to the small number of high-grade cobalt deposits in the world. The shortages that occurred in mid-1976 and again in mid-1978 further demonstrated the precarious nature of supply. Transportation problems resulting from the civil strife in Angola occurred in 1976. Insurgent attacks in the mining region of Shaba Province, Zaire, occurred in both 1977 and 1978, and the threat of a recurrence of such attacks, and occupation of greater duration remains a contingency. Although the shortage of cobalt at the producer price in 1978 was primarily the result of unusually high demand and lack of efficient recovery combined with generally limited production worldwide because of depressed markets for copper and nickel, the incidents underscored U.S. vulnerability to supply disruption. In addition, highly centralized sources of supply provide the producing countries with leverage whereby prices may be artificially increased. Development of domestic and foreign laterites, sulfides, and ocean floor resources, establishment of adequate substitutes for cobalt, and accumulation of sufficient stockpiles would be ameliorating measures.

Cobalt is usually mined as a byproduct of either nickel or copper. Because of this relationship, cobalt supply is relatively inelastic with respect to price. Therefore, another important problem that could develop is insufficient mine capacity for cobalt production resulting from lack of expansion of nickel and copper operations. There is a degree of supply elasticity, but the amount of copper and nickel produced ultimately places an upper limit on the amount of cobalt that can be produced. This problem can pose as serious a threat as insecure supply channels. As demand for cobalt continues to expand, new projects or expansion of existing ones becomes a necessity. Some expansions were brought on-stream in 1979. Although the current situation may be temporary, the relative price of copper versus cobalt has elevated cobalt to the

[1] Physical scientist, Ferrous Metals Section, Bureau of Mines.

status of a genuine coproduct, rather than by-product, of copper at some locations in the relatively short period of 1 year. This may have significant implications for new mine development.

Between 1973 and 1978, copper production in Zaire remained relatively constant, while production of cobalt progressively declined. The lack of priority given to cobalt production by Zaire during this period resulted in inefficiency and lower than normal recovery rates. The current much higher value of cobalt relative to copper is likely to change this situation, but only if Zaire can recover financially and fully utilize its capacity for refined cobalt production. An encouraging sign was a reported increase in production in 1978. With cobalt generating as much as one-third of export revenue of all mineral exports from Zaire and Zambia, and approaching copper in terms of total value of production, every possible effort to improve operations is likely. Zambia, through its expansions, will take up some of the slack created by lowered production in Zaire.

U.S. total annual demand for cobalt, based on contingency assumptions for end uses under various technologic and economic conditions, is forecast to grow at 3.5 percent and to reach 45.6 million pounds in 2000. Annual total demand for the rest of the world is forecast to grow at 3.6 percent, reaching 70 million pounds in 2000, compared with an estimated 31 million pounds consumed in 1977 and 33 million pounds consumed in 1978.

INDUSTRY STRUCTURE

Background

Cobalt was first used as a pigment about 2000 B.C., when naturally occurring cobalt compounds were used to color glass. This remained its primary use until 1742, when it was isolated as a metal by the Swedish scientist G. Brandt. About 40 years later, in 1780, Bergman established cobalt as an element.

The use of cobalt metal began in 1907, when Elwood Haynes received a patent on a series of cobalt-chromium alloys, which were named Stellites. The patented Stellites were forerunners of today's superalloys (*5*).[2]

In Japan, Honda and Saito (*10*) first published reports in 1920 on the magnetic properties of steels with high cobalt content. With this research as a foundation, Mishima, in 1930, discovered that adding cobalt to certain alloys of iron, aluminum, and nickel enhanced their outstanding permanent-magnet properties.

Subsequently, a series of permanent-magnet alloys containing aluminum, nickel, and cobalt (Alnico) were developed. The first alloy in the Alnico series was announced by General Electric Co. in 1936. The cobalt content of this alloy was found to improve both its magnetic properties and its response to heat treatment . Basic research in the 1950's included investigation of the binary, ternary, and quaternary alloys of cobalt. Since 1960, attention has focused on the use of cobalt in steel, permanent magnets, paint dryers, catalysts, and high-temperature alloys.

Size and Organization

Cobalt mine production in the United States ceased at the end of 1971. All production since that time has been from relatively minor amounts of secondary material derived from recycled alloys.

Averaged over the period 1974–77, about 65 percent of U.S. imports originated in Zaire, where cobalt was produced as a byproduct of copper. About 23 percent of these imports were shipped via Belgium for refining and processing, while 42 percent were imported directly from Zaire. In 1978, Zaire's share of total world cobalt refined metal production was estimated at 57 percent. In the same year, Zaire (directly and through Belgium) supplied 60 percent of total U.S. imports. Imports from Zambia were 11 percent of total U.S. imports in 1978. One company, the Government-owned Générale des Carrières et des Mines (Gécamines), mines most copper and cobalt in Zaire. To assist in mining, refining, and maintenance, Gécamines maintains a technical cooperation agreement with the Belgian company Sté. Générale des Minerais (SGM), a subsidiary of Sté. Générale de Belgique. Société Zairoise de Commercialisation (SOZACOM) is the official marketing agency for Zairian cobalt. A U.S. company, African Metals Corp. of New York, holds exclusive rights for sale of Zairian and Belgian cobalt in the United States. This relationship may change if, as anticipated, SOZACOM takes over more of the marketing functions now handled by SGM.

African Metals announced April 28, 1978, that cobalt metal orders would be accepted on an allocation basis only beginning May 1, 1978. The allocation was effectively 70 percent of the

[2] Italicized numbers in parentheses refer to items in the list of references at the end of this report.

customer's average monthly purchases during the calendar year 1977. Nearly 2 years earlier, in September 1976, because of defense considerations, sales of cobalt from then-existing U.S. government stockpile excesses had been halted. This meant the withdrawal of a major source of supply to U.S. industry. Foreign producers did not anticipate the impact that this loss of supply would have on the market when demand turned sharply upward. The allocation action was precipitated by a number of factors, including the relatively high demand for cobalt during the first quarter, hedge buying in anticipation of price increases resulting from currency realinments, and reduced production because of inefficiency of cobalt recovery and generally weak markets in copper and nickel. Exacerbating the situation, only about 2 weeks after the allocation began, insurgents disrupted cobalt mining and refining activity by invading and occupying parts of Shaba Province, Zaire, and most expatriate technical personnel were evacuated from the Kolwezi area, scene of the fighting. The facilities at Kolwezi account for at least half of Zaire's cobalt metal production, which totaled about 22.5 million pounds in 1977. Production at the mines was restored relatively quickly to preinvasion levels after the occupation, which lasted only about 2 weeks.

Transportation is a continuing problem in southern Africa, and is particularly acute for Zaire because its copper-cobalt mining district is located distant from any seaport. Normally, cobalt is shipped out of Zaire from the port of Matadi at the mouth of the Zaire River, after a 1,587-mile rail-barge-rail journey from the Kolwezi refinery in Shaba Province. However, at current relatively high prices, a significant percentage of cobalt is being transported to Europe by air. Most of Zaire's copper production is shipped south through Zambia, Southern Rhodesia, and Botswana to ports in the Republic of South Africa. This route is highly vulnerable to disruption, and if it were severed, cobalt supplies could be effectively limited because copper, of which cobalt is a byproduct, might not be produced if it could not be shipped.

Sulfide and oxide concentrates produced from the ores mined by Gécamines are processed to cobalt metal at the Shituru and Luilu electrolytic plants in Likasi and Kolwezi, respectively. Siliceous oxides are fed directly into the primary leaching circuit. Sulfide concentrates are heated in a fluosolid reactor in which sulfur dioxide is driven off, and the roast product, containing copper, cobalt, and other metals, is fed into the primary leaching circuit. After clarification of the solution, copper is recovered electrolytically, and cobalt is precipitated as cobalt hydroxide. The cobalt hydroxide is then redissolved in sulfuric acid for electrolytic deposition on stainless steel starting plates. The Shituru and Luilu plants both produce cathodes and pellets. The Shituru plant also produces granules. In addition, cobalt is smelted at the Panda-Likasi complex to produce cobalt-rich "white alloy" (matte), which is refined in Belgium by Metallurgie Hoboken Overpelt (7).

Cobalt is also a byproduct of Zambia's copper industry. It is initially recovered by selective flotation of sulfide ores from the N'Kana, Chibuluma, and Baluba deposits. There are two major state-owned mining companies in Zambia: Nchanga Consolidated Copper Mines, Ltd. (NCCM) and Roan Consolidated Mines, Ltd.(RCM). The Rokana Division of NCCM operates a mine, concentrator, and refinery at N'Kana. Cobalt concentrates are roasted and leached, and the cobalt hydroxide recovered from the leach solution is treated to permit electrolytic recovery of cobalt. RCM operates concentrators at Chibuluma and Baluba. Concentrates are shipped to RCM's Chambishi leach plant, and the cobalt hydroxide is converted to metal at the new Chambishi refinery. All of the recovery operations in Zambia are basically of the "roast-leach-electrowin" type. Future development of ore discovered at Chingola is planned. Significant recovery of cobalt from slag may also take place.

Mining for cobalt, copper, and nickel was begun in Botswana at the Selebi-Pickwe deposit in 1973. The ore is smelted in Botswana, and matte is shipped to the United States for refining. Bamangwato Concessions, Ltd ., a subsidiary of Botswana, RST, Ltd., reported production of cobalt-bearing matte in 1978 at more than half of rated capacity. When all startup problems have been overcome, 0.5 million pounds per year of cobalt could be extracted from matte produced at this mine.

A fourth major African producer is Morocco. Deposits assaying as high as 1.2 percent cobalt are found in the Bou Azzer and El Graara areas on the eastern edge of the Atlas Mountains. Most of the cobalt produced in Morocco is exported to France. Reserves are adequate for at least 10 years at the present rate of production.

In Canada, INCO, Ltd., produces commercial grade cobalt oxide at Port Colborne, Ontario, and Thompson, Manitoba . Refined cobalt oxides and salts are produced at INCO's refinery at Clydach, Wales (using oxide shipped from Canada). Falconbridge Nickel Mines, Ltd., ex-

DISTRIBUTION OF ESTIMATED WORLD REFINED COBALT METAL AND OXIDE PRODUCTION IN 1978

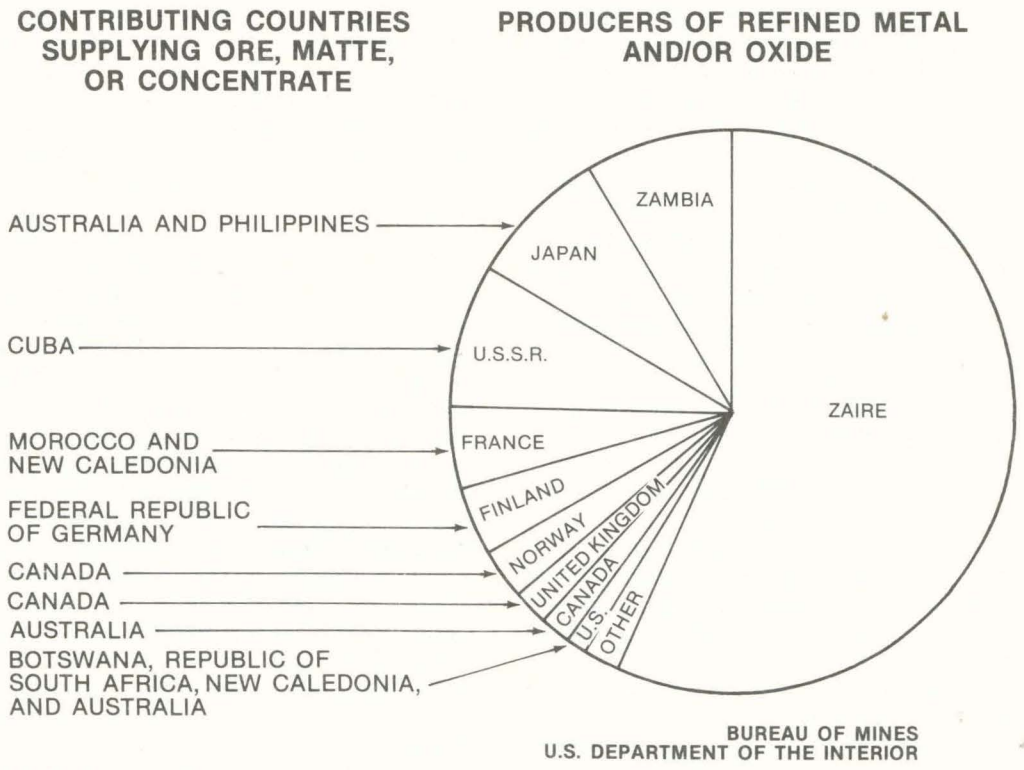

Figure 1.—Distribution of estimated world refined cobalt metal production in 1978. Belgium is not listed because it is mainly a processor of refined metal.

ports cobalt bearing matte from its mines and smelters in Canada to its refining operation in Kristiansand, Norway. Cobalt is also recovered as a byproduct of nickel and copper concentrates by Sherritt Gordon Mines, Ltd., at its refinery in Fort Saskatchewan, Alberta. Sherritt Gordon's Lynn Lake mine, a former source of feedstock for its refinery, was shut down in 1976. Through 1978, a large portion of feedstock for the plant came from Australia. In 1978 an agreement was signed with INCO, whereby Sherritt would refine some concentrate produced at Thompson. At the Agnico-Eagle Mines, Ltd., in Ontario, some cobalt has been recovered as a byproduct of the smelting and refining of complex silver-cobalt concentrate. New discoveries may result in significant expansion of this operation.

In the Philippines, Marinduque Mining and Industrial Corp. experienced technical difficulties through 1978 with its Nonoc Island nickel-cobalt refinery, which began production in 1973. When and as the problems are overcome, 3.3 million pounds of cobalt in mixed sulfides is scheduled to be produced annually. Marinduque announced in 1977 that it planned to construct a cobalt refinery in the Philippines to refine 7 million pounds per year of cobalt contained in a mixed sulfide concentrate. If Marinduque decides to go ahead with the project, the facility may be on-stream in the early 1980's. Currently, all of the concentrate produced is refined in Japan. In 1978, Marinduque and Sumitomo Metal Mining Co. changed their agreement so that one half (50 tons per month) of the cobalt produced would be toll-refined, rather than purchased by Sumitomo, and would be marketed by Marinduque.

Cobalt is produced in Australia by Freeport Queensland Nickel, Inc. (a subsidiary of Freeport Minerals, Inc., of New York) and Metals Exploration Queensland, Pty., in Greenvale, Australia. Repairs and redesign of ore-processing units at the Greenvale nickel-cobalt metallurgical plant in Queensland have brought the operation close to its design capacity of 2.75 million pounds per year of cobalt. The mixed sulfide concentrate is shipped to Japan for re-

Table 1.—Principal world cobalt producers

Country	Company
North America:	
Canada	Sherritt Gordon Mines, Ltd.
	Inco, Ltd.
	Falconbridge Nickel Mines, Ltd.
Cuba	Cubaniquel (State owned)
United States	AMAX Nickel, Inc.
Europe:	
Belgium	Métallurgie Hoboken-Overpelt, S.A.
Finland	Outokumpu Oy (Government-owned).
France	Pechiney Ugine Kuhlmann.
	Société Le Nickel.
Germany, Federal Republic of	Duisburger Kupferhutte.
	Gesellschaft fur Electometallugie, GmbH.
	Herman C. Starck.
Norway	Falconbridge Nikkelverk A/S.
United Kingdom (Wales)	International Nickel, Ltd.
U.S.S.R.	State plants located in Balkhash, Touva (Khozu-aksy), Norilsk, and Verkni-Oufalei.
Africa:	
Botswana	Botswana RST, Ltd.
Morocco	Campagnie de Tifnout Tiranimine.
Zaire	Générale des Carrières et des Mines (Gécamines).
Zambia	Roan Consolidated Mines, Ltd.
	Nchanga Consolidated Copper Mines, Ltd.
Asia:	
Japan	Nippon Mining Co.
	Sumitomo Metal Mining Co.
Oceania:	
Australia	Freeport Queensland Nickel, Inc.
	Metals Exploration Queensland, Pty.
	Western Mining Co.
New Caledonia	Société Le Nickel (SLN).
Philippines	Marinduque Mining and Industrial Corp.

fining. Also, Western Mining Corp. smelts a concentrate produced by Great Boulder Mines, Ltd., which Western Mining acquired in 1976, and by its Kambalda operation (17). A portion of the matte is exported to Canada for refining at Sherritt Gordon's Fort Saskatchewan plant. This arrangement was altered at yearend 1978, and the material now is to be toll-refined for Western Mining.

AMAX Inc. began significant production at its refinery in Braithwaite, La ., in 1975 and greatly improved operating capacity in 1978. About 643,000 pounds of cobalt was produced in 1978; this was about 40 percent greater than that reported for 1977. Nickel-cobalt and nickel-copper-cobalt matte was supplied to the plant from Botswana, New Caledonia, and the Republic of South Africa. In March 1978, AMAX announced that it would purchase the entire output of matte from Botswana Concessions, Ltd. (BCL) rather than toll-refine the material, which is to supply the Port Nickel, La., refinery with 50 percent of its feedstock requirements. In 1979, the facility also began processing matte from the Agnew, Australia, nickel project.

Other major producing countries are Cuba, Finland, New Caledonia, and the U.S.S.R. In Finland, Outokumpu Oy mines cobalt-bearing pyrite deposits, which may become depleted in the next 15 years. The exact quantity of refined cobalt metal produced from New Caledonian ores is unknown, but it is relatively small. Most cobalt that is mined there either is not recovered in processing or is included in various ferronickel products as an impurity. Société Le Nickel in France will have the capacity in 1980 to refine as much as 1 million pounds per year of cobalt from New Caledonian sources. Pechiney Ugine Kuhlmann in France refines cobalt produced in Morocco. In Indonesia, PT Pacific Nikkel Indonesia (PNI) had announced plans to develop a large cobalt-bearing laterite deposit, but these plans were postponed early in 1979. Other major refining and processing operations are located in West Germany, Belgium, and Norway. In a move toward greater Japanese independence in refining capability, Nippon Mining Co. and Sumitomo Metal Mining Co. established domestic refining operations that now satisfy at least 50 percent of Japan's refined cobalt requirements. Production at these plants reached design capacity in 1978. Major world cobalt producers are listed in table 1.

Geographical Distribution

The top five producers of cobalt in terms of world mine output (estimated) in 1978 were Zaire (41 percent), New Caledonia (13 percent), Australia (11 percent), Zambia (6 percent), and Canada (3 percent). In terms of estimated cobalt metal produced by refining, the largest producers for which data are available were Zaire (57 percent), U.S.S.R. (8 percent), Zambia (8 percent), Japan (8 percent), and France (5 percent) (fig. 1). Belgium does not publish production figures. However, with an estimated annual production of 14 million pounds per year, Belgium was probably the largest producer of processed metal. Most of this material was previously refined to metal in Zaire. Little information is available concerning 1978 cobalt production in Cuba, Eastern Europe, the U.S.S.R., or China. South Africa, Bulgaria, Cyprus, the German Democratic Republic, Poland, Spain, and Sweden are also known to produce ores that contain recoverable quantities of cobalt. Production capacities of Botswana, Cuba, Indonesia, New Caledonia, the Philippines, Zambia, and Zaire are expected to increase significantly over the next 10 years as a result of expanded facilities and the development of newly discovered resources. One impediment to expansion of facilities in Zaire has been suspension of construction of the Inga-Shaba power transmission line, which was to supply power to the proposed Tenke-Fungurume project. Resumption of construction in 1979 appeared likely. Cobalt production in 1978 and capacity in 1978, 1979, and 1985 for various areas of the world are estimated in table 2.

Table 2.—World cobalt mine production, 1978, and capacity, 1978, 1979, and 1985

(Thousand pounds)

	Production in 1978[e]	Capacity[e]		
		1978	1979	1985
North America:				
Canada	2,000	4,400	4,500	4,600
Cuba	3,600	3,800	3,800	4,000
Total	5,600	8,200	8,300	8,600
Europe:				
Finland	3,000	3,100	3,100	3,500
U.S.S.R.	4,400	5,000	5,000	6,000
Total	7,400	8,100	8,100	9,500
Africa:				
Botswana	400	500	500	500
Morocco	4,000	4,000	4,100	5,000
Zaire	29,000	30,000	32,000	40,000
Zambia	4,000	5,400	6,400	10,000
Total	37,400	39,900	43,000	55,500
Oceania:				
Australia	7,600	7,600	8,000	10,000
New Caledonia	9,200	9,200	9,200	10,000
Philippines	2,400	2,500	2,500	3,000
Total	19,200	19,300	19,700	23,000
World total[1]	70,000	76,000	79,000	97,000

[e] Estimated.
[1] Data may not add to total shown because of independent rounding.

Definition, Grades, Specifications

Cobalt is a silvery gray metal with an atomic number of 27 and an atomic weight of 58.94 (*20*). Its melting point is 2,723° F, its density is 8.9 grams per cubic centimeter, and it exhibits a faint bluish tinge when polished (*5*). Moreover, when magnetized, cobalt retains its magnetic properties up to 2,050° F, making it very useful in pemanent magnet materials. About 70 percent of the cobalt consumed in the United States is in some form of the metal, such as briquets, granules, squares, broken cathodes, rondelles, fines, or powder. More than 10 percent of the cobalt produced in the world is consumed in various chemical compounds other than oxide, and more than 5 percent is used as an oxide, principally gray cobaltous oxide (CoO) and black cobaltic oxide (Co_3O_4). Cobalt salt producers generally market about six cobalt compounds, including acetate, carbonate, chloride, nitrate, and sulfate. The term "specification-grade" has been frequently used in reference to GSA stockpile sales. This material is 99.7 percent cobalt (minimum), with less than 0.2 percent nickel and less than 0.03 percent impurities each of iron, carbon, copper, sulfur, and phosphorus. This specification may be changed to conform with the purity of material currently being produced (99.9 percent Co).

USES

Cobalt is used principally in heat- and wear-resistant materials, cutting tools, high-strength materials, permanent magnets, and various chemical applications. Cobalt is also used as an alloying element in tool and die steels. In metallic form, cobalt is used primarily in permanent magnets, cemented carbides, and various alloys. More than 25 percent of cobalt consumed is processed into nonmetallic compounds. These compounds are used as dryers in paints, and in pigments, enamels, rubber, and catalysts (*18*). Major end uses in 1978 were transportation (aircraft), 23 percent; electrical (magnets), 22 percent; machinery (principally cutting tools), 16 percent; paints (mainly dryers), 16 percent; ceramics, 13 percent; and chemicals 8 percent (fig. 2).

Electrical Equipment and Supplies.—Permanent magnets are widely used in the electrical-equipment industry. Cobalt additions raise the saturation magnetization and Curie temperatures to higher values than those obtained with pure iron. Because of their high coercivity, cobalt-containing magnets are used in telecommunications, magnetic couplings, meters, loudspeakers, permanent magnet motors, and repulsion devices.

Of the cobalt used in magnets, Alnico (aluminum-nickel-cobalt) magnets represent the largest consumption category. However, samarium-cobalt and mischmetal-cobalt magnets may become very strong competitors. These magnets have definite weight advantages in certain applications because of their higher coercivity. An approximate breakdown of the use of cobalt magnets follows, in percent: loudspeakers, 25; rotating electric machines, 25; indicating devices, 25; telecommunications, 20; all other uses, 5.

Aircraft and Surface Engines and Parts.—Much of the cobalt that is used in superalloys is for jet engine parts, which are subjected to stress at high temperatures. These alloys contain from 1 to 65 percent cobalt. Cobalt-base alloys can withstand temperatures up to 1,600° F under conditions of low stress (about 10,000 psi) and can withstand moderate stress (about 20,000 psi) at temperatures to 1,800° F for adequate life in turbine components, however, under conditions of high stress (30,000 psi) at 1,900° F they will rupture in a few hours. Cobalt is also used in some nickel-base superalloys with similar characteristics (table 3). Expansion of usage has occurred recently in industrial turbine engines, aircraft turbines, natural gas transmission pipe-

Table 3.—Cobalt-bearing superalloys

Designation	Cobalt content, percent
Cobalt base:	
Wrought:	
Haynes 25 (L605)	53.4
Haynes 188	41.3
MP35N	35.3
Case:	
Stellite 21	64.5
Stellite 31	56.0
WI–52	63.8
MAR-M 302	57.4
MAR-M 509	54.8
X–40	54.0
FSX–414	52.0
Haynes alloy 1002	47.0
Nickel base:	
Wrought, nonhardenable: Hastelloy X	1.5
Wrought, age-hardenable:	
Waspalloy	13.5
Astroloy	17.0
Udimet 700	18.5
Inconel 100	15.0
MOD IN 100	18.5
Case:	
Hastelloy X	1.5
Waspalloy	13.5
Udimet 700	15.3
IN 100	15.0
MAR-M 200 & Hf	10.0
B–1900	10.0
Inconel 738	8.5
Inconel 792	9.0
FA 375	10.0
CH–71	7.5
Udimet 500	18.0

Sources: Pratt & Whitney Aircarft Division of United Technologies Corp. (July 31, 1972); Inco, Ltd. (July 1977).

lines, and marine equipment (*16*). As much as 30 percent of the cobalt used in superalloys may be utilized in industrial turbines for power generation.

Tools.—Cobalt is an important constituent in tools requiring very high strength and abrasion resistance. It is used in cutting tools as the metal matrix or "cement" for tungsten carbide, titanium carbides, and alloyed tungsten carbides containing tantalum, titanium, or vanadium. These are referred to as cemented carbides and generally contain 10 to 15 percent cobalt. Cobalt is also used in carbide drawing dies (up to 30 percent cobalt) and masonry saw teeth. Several high-speed steels used in machining iron, steel, and nonferrous metals require cobalt.

Construction and Mining.—Cemented carbide bits are used in mining and drilling operations. In sour-gas applications, an alloy containing 35.3 percent cobalt has found limited use for parts such as logging tools, valve stems on wellheads, springs, and safety-valve systems. The use of extruded seamless tubing made of this alloy may expand cobalt usage considerably in this area. Coating with a cobalt alloy, or "hardfacing," provides greater resistance to abrasion, heat, impact, and corrosion. For example, valves used in automobiles and the petrochemical industry, chain saws, dipper teeth of power shovels, and the surfaces of plowshares may be treated in this manner. Cobalt alloys may also be thermal-sprayed in powder form on worn construction machinery parts to provide a protective coating. The increasing use of plasma-arc hard facing reportedly is resulting in greater efficiency in production of valve seats and other surfaces that require protective coating of cobalt-base alloys.

Paints and Related Products.—Cobalt oxides and organic compounds are used in paints, ceramics, and allied products as decolorizers, dyes, dryers, pigments, and oxidizers. Cobalt used as a ground coat frit also promotes the adherence of enamel to steel. The rubber industry uses organic cobalt compounds to promote the adherence of steel to rubber in steel-belted radial tires. Cobalt content of these compounds is in the 12- to 18-percent range. Total quantities used are unknown.

Miscellaneous Chemical Products.—The principal use for cobalt in chemical processes is as a catalyst in hydrogenation. Other chemical uses are in hydration, desulfurization, oxidation, reduction, and synthesis of hydrocarbons. A cobalt-molybdenum-alumina compound is the common petroleum desulfurization catalyst (*7*).

Other Uses.—Cobalt is added to soils so that crops produced for animal feed will contain enough of the element to serve as a nutritive agent. This need was discovered in 1935 by Eric J. Underwood and John F. Filmer of Australia's Department of Agriculture. It was found that a shortage of cobalt, though only in trace amounts, causes what is known as "wasting disease" in sheep and cattle. Cobalt is also essential to humans as a component of vitamin B_{12}, which can prevent pernicious anemia. It is one of a group of elements, including copper, manganese, molybdenum, selenium, zinc, and chromium, for which a need has been established but not yet quantified. Dietary sources for vitamin B_{12} are meat and dairy products. The isotope cobalt-60 provides a relatively inexpensive source of gamma radiation and is used in physical, chemical, and biological research and treatment, and in industrial radiography for the investigation of physical strains and imperfections in metals. Cobalt is used in significant quantities in corrosion-resistant prosthetic devices such as hip and arm joints, and in alloys used in the dental field.

RESERVES-RESOURCES

United States and World

Most cobalt resources are only available as byproducts of mining for more abundant ele-

Table 4.—Identified world cobalt resources

(Million pounds)

	Estimated reserve base[1]	Other identified resources[2]	Total identified resources[3]
North America:			
United States	0	1,700	1,700
Canada	66	480	550
Cuba	240	2,100	2,300
Total	300	4,300	4,600
Europe:			
Finland	40	10	50
U.S.S.R.	460	40	500
Total	500	50	550
Africa:			
Botswana	58	12	70
Morocco	28	2	30
Zaire	1,000	500	1,500
Zambia	250	520	770
Total	1,300	1,000	2,400
Oceania:			
Australia	110	540	650
New Caledonia	600	250	850
Philippines	420	30	450
Total	1,100	820	2,000
World total, land-based[3]	3,200	6,200	9,600
World total, seabed nodules[4]	0	500,000	500,000

[1] Reserve base is defined as that part of the identified in-place resource meeting minimum physical and chemical requirements as related to current mining and production practices.
[2] Derived in consultation with the U.S. Geological Survey.
[3] Data may not add to totals shown because of independent rounding.
[4] Based on Holser's paramarginal resource estimate of 76 billion dry short tons of sea nodules and the NMAB–323 estimate of average nodule composition of 1 percent copper, 1 percent nickel, 0.35 percent cobalt, and 24 percent manganese. (Holser, F., Manganese Nodule Resource and Mine Site Availability, Ocean Mining Administration, U.S. Department of the Interior, August 1976, p. 28; National Materials Advisory Board, NMAB–323, 1976)

ments. Production of cobalt as a principal metal requires a relatively high-grade, extensive ore body such as the Bou-Azzer district in Morocco. The largest profitable mining district for cobalt in the United States was the Blackbird district of Lehmi County, Idaho, about 20 miles southwest of Salmon. This district was one of the rare locations where cobalt has been mined as the principal product, but the mine was closed in June 1959. If cobalt mining resumes in the Unites States, the Blackbird district is the most likely candidate, partly because of its relatively high-grade ore (0.6 percent cobalt). Plans were made in 1979 for reopening the mine, with initial shipment of the ore to an overseas processor. Mining for cobalt as a byproduct has taken place at two principal locations: Fredericktown, Mo., from a copper-nickel-cobalt mine closed in mid-1961; and Cornwall, Pa., from iron ores mined until mid-1971. One company announced plans in 1979 to reopen the former Madison mine, located near Fredericktown. No cobalt was produced in the United States in 1978, and none of the extensive domestic resources that exist are presently considered to be reserves. Minnesota possesses the largest identified resource in the United States—approximately 250 million pounds of cobalt in the form of sulfide in the Ely area. Missouri and California are second, with 150 million pounds each; deposits are located in the southeast Missouri lead district and the northern California laterite zone, respectively. Oregon, Alaska, and Pennsylvania also have large identified resources.

Much of the world's identified resources are in the form of lateritic nickel ores in tropical regions, such as the Philippines, Indonesia, and New Caledonia. Australian production from laterites has become an increasingly significant source. Nevertheless, most cobalt currently comes from sulfide and oxide deposits in Zaire, Zambia, Finland, and Canada. Expansions were planned in 1979 in each of these countries. Moroccan deposits may become more significant in the future. Moroccan cobalt is mined as the principal product, and rising cobalt prices may result in production of previously uneconomic material. However, the best long-term prospect for abundant supplies may be the development of cobalt-bearing manganese nodules on the Pacific Ocean floor.

Table 4 shows the areas possessing identified resources of cobalt. Speculative resources in both the United States and the world amount to many billions of pounds more, but cobalt mining is only likely if the associated metals, such as nickel, copper, iron, lead, zinc, and silver are in high demand and are mined. In terms of estimated reserves, Zaire ranks first, followed by New Caledonia, the U.S.S.R., the Philippines, Zambia, and Cuba.

To provide current resource information, the Bureau of Mines maintains a data bank under its computerized Minerals Availability System (MAS) in which reserve and resource data on cobalt and other mineral ore deposits in the United States and throughout the world are being compiled.

Geology

Average crustal abundance of cobalt is 20 parts per million (ppm). The greatest crustal concentration of cobalt occurs in mafic and ultramafic igneous rocks, with a smaller concentration at the silica-rich end of the differentiation series. The average concentration of cobalt in ultramafic rocks is about 270 ppm. Concentration of cobalt in sedimentary rocks is not well known, but a variety of sources indicate an average of 4 ppm in sandstone, 6 ppm in carbonate rocks, and 40 ppm in shale, clay, and mudstone (*20*). Metamorphic rocks also contain relatively

small concentrations. About 70 minerals contain cobalt as an important constituent, but only 20 commonly exist in economic concentrations.

Although cobalt may be recovered as either a coproduct or a byproduct in the mining of copper ore, it is usually recovered as a byproduct. When mined with nickel, it invariably is mined as a byproduct. Cobalt has also been mined as a byproduct of iron, zinc, lead, silver, and uranium.

The deposits of Zaire and Zambia are of many different geological ages, and typically one metal predominates (usually copper, lead, or zinc). The largest production of cobalt is from cupriferous strata-bound deposits. These deposits occur in an area with the shape of a large folded arc. The cobalt-bearing ores contain the minerals bornite, chalcocite, chalcopyrite, linnaeite, and carrolite, and the secondary minerals heterogenite, sphaerocobaltite, and erythrite. The ore beds and veins themselves in places may contain as high as 3 or 4 percent cobalt.

The Sudbury district of Ontario, Canada, considered by some to be an intrusive body, is rich in copper and nickel. It contains about 0.07 percent cobalt (*3*). Some of the ore minerals at this deposit are pyrrhotite, pentlandite, pyrite, marcasite, magnetite, and cobaltite. These minerals occur in pods, lenses, veins, and disseminated grains in a layered sequence of norite, quartz diorite, micropegmatite, and rocks of transitional composition (*20*). Other similar iron-nickel-copper sulfide deposits are found in the Duluth gabbro complex southeast of Ely, Minn.

Laterite deposits result from deep weathering of basic igneous rocks in tropical climates with abundant rainfall. By this weathering process, magnesia and silica are selectively leached out and deposited at a lower level or completely removed, whereas iron, manganese, and cobalt are residually enriched relatively close to the surface. Nickel is also leached out and often concentrated below and apart from the cobalt. Laterite deposits with relatively high concentrations of cobalt occur in Australia, Cuba, New Caledonia, Indonesia, and the Philippines. Cobalt-bearing material in the laterite (termed "asbolane") may take the form of nodules, lenses, or small veins in a matrix of red iron-rich clay, usually without a high concentration of nickel in direct association with cobalt. In Cuba, however, both nickel and cobalt are often found more closely associated.

One potential source of cobalt is small nodules on the ocean floor. Under oxidizing conditions, cobalt exhibits a strong tendency to follow nickel geochemically and to concentrate with manganese. This is probably the primary reason for the association of cobalt and nickel in manganese nodules on the ocean floor. According to Mero (*12*), MnO precipitates as submicroscopic colloidal particles that capture nickel, copper, cobalt, zinc, and molybdenum in the water. A typical assay of metal (*15*) in the nodules reveals the following percentages: 24 manganese, 1 nickel, 1 copper, and 0.35 cobalt. The nodules also contain 6 percent iron and are very porous, containing about 30 weight-percent water.

TECHNOLOGY

Exploration, Mining, and Processing

Since cobalt occurs in nature associated with nickel and copper, the metals of principal interest, exploration and development generally follows the pattern for these major commodities. Because of the diversity of cobalt deposits, various techniques are used to extract the ore. In the largest deposits, in Shaba Province of Zaire, open pit and underground methods are used. In Zambia, another location of a cobalt-bearing copper deposit, a modified, sublevel and cut-and-fill stoping method is employed. A variety of methods are used in the cobalt-bearing nickel-copper deposits of Canada. At Sudbury, for example, blasthole-stoping, block-caving, and open-pit methods have been found effective.

A variety of extractive metallurgical techniques are used in processing cobalt. Depending on the original state of the ore, whether arsenide, sulfide, or oxide, one or a combination of the following methods may be utilized: magnetic concentrating; roasting; leaching at normal or elevated temperature and pressure with an ammoniacal solution or sulfuric acid; and chemical precipitation by solids, liquids, or gases (*21*). Electrolytic reduction is common.

In general, the ores that were mined in 1978 contained 0.1 to 0.5 percent cobalt, which necessitates concentration prior to processing. After crushing, grinding, and flotation of the ore, a concentrate is obtained that has a cobalt content four to eight times higher than that of the ore.

The concentrate produced from the copper-cobalt ore in Zaire is leached in sulfuric acid. After extraction of the copper by electrolysis, part of the electrolyte is purified in the cobalt plant, where cobalt is precipitated in the form of a hydrate. The hydrate is then leached in acid, and the resulting solution is clarified. Cobalt is extracted by electrolysis.

Sherritt Gordon Mines, Ltd., uses a pressure ammonia treatment of nickel concentrate. A continuous leaching operation is employed in which compressed air and ammonia are applied to the concentrate. Nickel and cobalt are separated, and cobalt is reduced to powder form, washed, dried, and made into briquets or sold in its original form (*17*).

Research

One of the most important developments regarding the supply of cobalt has been that of mining and processing cobalt-bearing manganese nodules from the ocean floor. This relatively recent development is discussed in a later section, entitled "Possible Technological Progress."

Another recent development has been that of rare-earth-cobalt magnets. These magnets are particularly attractive because they can attain a much higher coercive force than conventional Alnico magnets, have been shown to be as much as 50 times more resistant to demagnetization, and are less subject to limitations of size and shape. Bureau of Mines research has been directed at determining properties of praseodymium-cobalt and mischmetal-cobalt magnets. Cost of high-energy magnetic materials may be reduced by replacing samarium in the samarium-cobalt alloy with mischmetal or other less expensive rare-earth elements, if performance characteristics can be equaled or exceeded.

The Bureau of Mines also is actively engaged in research aimed at reducing U.S. import dependence. Through research on processing laterites in Oregon and northern California, treatment of lead-zinc ores in Missouri, sulfide ores in Minnesota, and seabed manganese nodules, total U.S. independence in cobalt might someday be achieved.

In superalloy research, industry has made significant strides in directionally solidified alloys and single crystals for use in jet engine component parts, but has not emphasized the development of new alloys. The purpose of this research is the attainment of higher operating temperatures in jet engines, considered important because engine efficiency is increased at higher temperatures, while pollution is decreased. Moreover, superalloy producers are now putting more emphasis on improving techniques of fabrication, such as hot forging, extrusion, hot isostatic pressing, and precision metalworking. This is being done because it is becoming increasingly difficult to justify the expense of research into what may be only marginal improvements in superalloy properties. One exception to this trend is the development of ceramic composite materials, which may allow greatly increased engine-operating temperatures. In addition, the Department of Defense sponsors a great deal of research by private industry on the use of cobalt in superalloys. Ordnance laboratories of the U.S. Army, Navy, and Air Force also perform studies on the use of cobalt.

At least two companies in North America currently are engaged in development directed toward production of an ultrafine cobalt powder suitable as a binder for cemented carbides. A commercially acceptable product has been produced by one of these companies using tungsten carbide scrap as feedstock. The main source for this special powder (about 1.4-micron mean particle diameter) is Belgium. The Belgian producer, Metallurgie Hoboken Overpelt, announced in 1978 that it would construct a plant in Laurinburg, N.C., to produce about 1 million pounds per year of extra fine powder, to be fully operational by 1980.

On a worldwide basis, research is being conducted on heat-resistant alloys and composites, cobalt-alloy systems and phases, permanent and soft magnetic materials, cobalt-bearing nickel-zinc batteries (*9*), and films and coatings. Powder metallurgy, in which metal particles are heated to fusion and then cooled in a vacuum, has made significant inroads in tool steel fabrication. Other research is being conducted on wear-resistant steels, cemented carbides, and alloyed coatings and compounds (*4*).

SUPPLY-DEMAND RELATIONSHIPS

Domestic Distribution

U.S. industry relies almost entirely on imports of cobalt for its supply. Actual 1978 imports constituted 87 percent of domestic demand. The remainder came from drawdown of industry stocks (8 percent) and secondary production (5 percent). The only cobalt produced in the United States in 1978 was from imported nickel-copper-cobalt matte refined in Louisiana. Some of the cobalt metal produced by the refinery was shipped to European customers under contract arrangements. Total domestic shipments in 1978 were distributed as apparent consumption (92 percent) and exports (8 percent).

Production and Consumption

Since virtually all cobalt is recovered as a by-product of copper or nickel, the quantity pro-

duced is dependent on the rate of mining for these metals. In 1978, world mine production of cobalt was estimated at 70 million pounds, of which 29 million pounds was produced in Zaire. Australian production was estimated at 7.6 million pounds in 1978, the third largest in the world, and Zambian production at 4.0 million pounds, which placed it fourth. Most New Caledonian production was cobalt content of ore, from which relatively small quantities of cobalt were refined to metal. With rising prices for cobalt metal, New Caledonia, which ranks second in world production, may become a significant source. Société Le Nickel (SLN) plans to complete a cobalt circuit at its Le Havre refinery in France, which would produce approximately 1 million pounds of cobalt per year from New Caledonian ores. However, a serious fire in late 1978 delayed production startup until 1980. Despite transportation problems between Southern Rhodesia and Zambia and closure of the Benguela railway resulting from the civil strife in Angola, both Zambia and Zaire have been able to maintain shipments via alternative routes to destinations on the east coast of Africa and to Zaire's own port of Matadi at the mouth of the Zaire River on the Atlantic. The east coast ports include East London in South Africa and Dar es Salaam in Tanzania. The Tazara railway, linking Dar es Salaam with the interior, has been the principal route of export for copper and cobalt from Zambia. Most recently, it has been economical to transport all Zambian cobalt by air to foreign markets. Zaire has also made use of air transport for cobalt. The Benguela railway through Angola, closed in 1975, is a potential future transport route for use by Zaire and Zambia.

Estimated world production of refined metal in 1978 amounted to 50.7 million pounds. Zaire led with 57 percent of the total world production, followed by Japan (8 percent), Zambia (8 percent), the U.S.S.R. (8 percent), and France (5 percent). Not included in the total production figure is Belgium, which does not publish production data but is believed to process about 14 million pounds annually. The total metal produced falls short of the cobalt content of total mine production by about 19 million pounds. This may be accounted for by the following: (1) Recovery rates vary in different locations but never exceed 95 percent and in some cases are as low as 25 percent, and production of some countries, such as Australia and New Caledonia, is given as total cobalt content of ore, rather than recoverable content, (2) all metal-producing countries do not report, (3) there are inherent inaccuracies in both estimations and reported data of mine and metal production, and (4) unrefined matte and concentrates, accumulated in stock or in transit, are unaccounted for. Belgium processes to various forms and compounds roughly one-half of the refined cobalt exported by Zaire and, in turn, exports about one-half its production to the United States. Nearly all cobalt from Morocco is refined in France. In addition, about 1 million pounds per year of cobalt is expected to be refined in the United States by late 1979, principally from imported matte produced in Botswana, New Caledonia, and the Republic of South Africa. A labor strike at the Port Nickel, La. refinery in September may prevent attainment of this production level in 1979. Table 5 lists reported world production of refined cobalt metal.

Table 5.—Cobalt: World refined metal production by country

(Thousand pounds)

	Metal[1]	
	1977	1978[p]
Canada[2]	1,012	1,000
Finland	2,172	2,000
France	[e]1,880	2,400
Germany, Federal Republic of	[e]1,770	800
Japan	2,410	4,100
Norway	1,554	1,500
U.S.S.R.[e]	4,200	4,000
United Kingdom[e]	1,780	1,200
United States	508	650
Zaire	22,520	29,000
Zambia	3,754	4,000
Total[3]	42,560	50,700

[e] Estimated. [p] Preliminary.
[1] Czechoslovakia and the U.S.S.R. presumably recover cobalt from materials imported from Cuba. Belgium continued to process cobalt, but output is not reported and available general information is inadequate for formulation of reliable estimates of output levels.
[2] Data presented represent the output within Canada of metallic cobalt from ores of both Canadian and non-Canadian origin.
[3] Data may not add to totals shown because of independent rounding.

U.S. consumption traditionally has accounted for a significant portion (35 to 45 percent) of total world consumption, which was estimated to be about 50 million pounds in 1977 and 55 million pounds in 1978. Few data are available on consumption patterns in the rest of the world. Besides the United States, only Japan and Canada publish end-use data. Japan consumed about 5 million pounds of cobalt in 1978 and Canada about 300,000 pounds. Figure 3 illustrates the U.S. supply-demand pattern. Table 6 lists supply-demand statistics in the United States for 1969–78.

World Trade

U.S. imports of cobalt in 1978 totaled 19 million pounds, the highest level since 1973. Surging domestic demand in line with generally accelerated industrial activity, particularly in air-

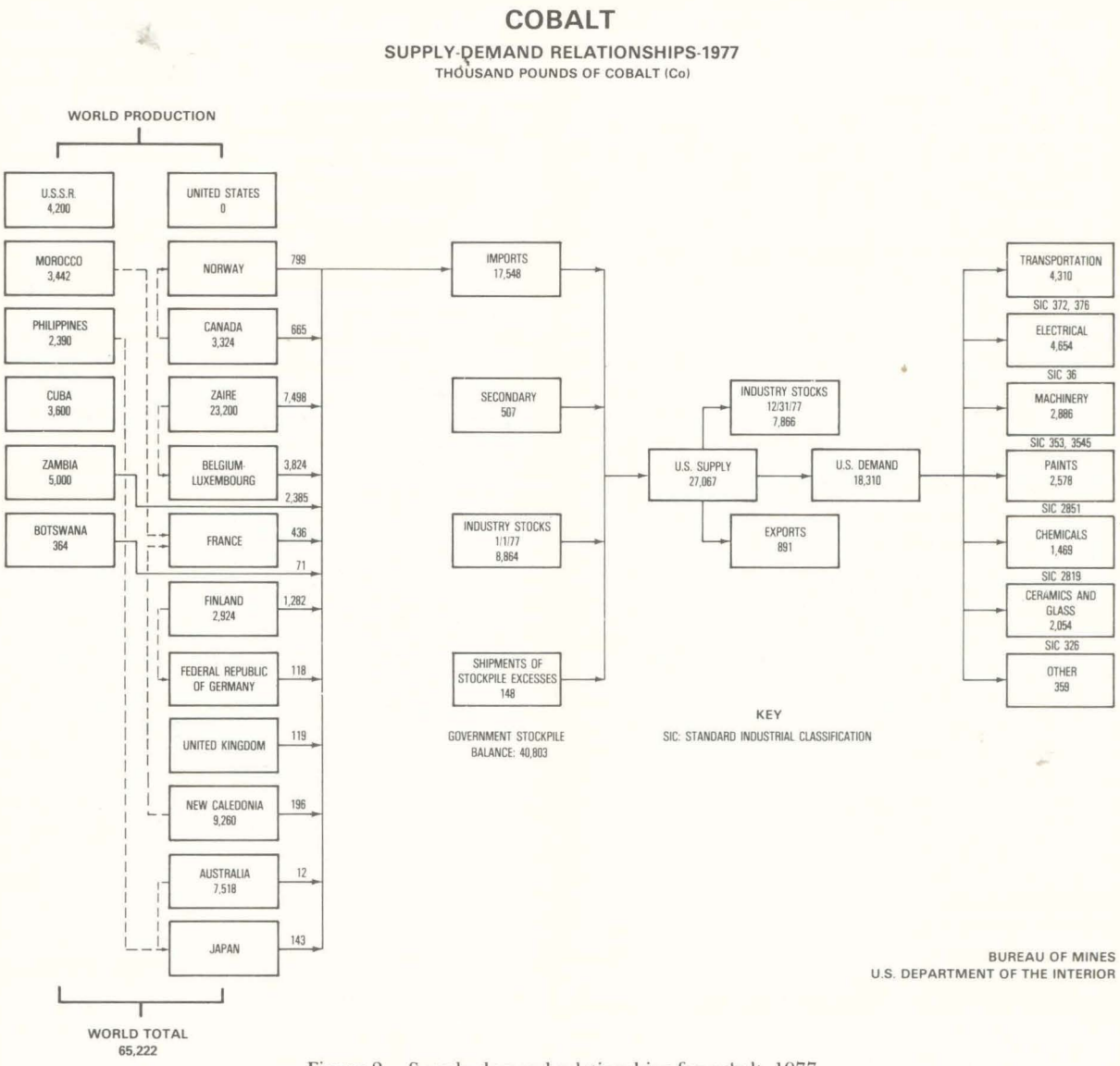

Figure 2.—Supply-demand relationships for cobalt, 1977.

craft construction and nonmetallic applications, was the principal reason for the increase. Main import sources (averaged over the years 1974–77) were Zaire, 42 percent; Belgium, 23 percent; Zambia, 7 percent; Finland, 6 percent; and others, 22 percent. In 1978, U.S. imports were 10 percent greater than in 1977 (fig. 2).

An export-reporting system for cobalt-bearing materials was announced by the Department of Commerce and published in the Federal Register, December 29, 1978. The new system went into effect January 10, 1979, and requires that exporters send to the Department of Commerce, Office of Export Administration, a copy of the export declaration covering a list of specified materials (both wrought and unwrought) containing 10 percent or more of cobalt. Included in the list are cobalt and cobalt alloys, cobalt oxides and hydroxides, and various tungsten and nickel alloys containing cobalt. The reporting requirement was implemented in an effort to obtain data that will aid the Department of Commerce in determining if exports of cobalt-bearing materials are having an adverse impact on supplies or prices of cobalt in the United States. Authority for this action is provided under Sections 3(2)(A) and 4(C)(1) of the Export Administration Act of 1969, as amended.

The United States exports cobalt, primarily to Japan, France, the United Kingdom, and

Table 6.—Cobalt supply-demand relationships, 1969–78

(Thousand pounds)

	1969	1970	1971	1972	1973	1974	1975	1976	1977[p]	1978[e]
World production:										
United States	1,003	697	690							
Rest of world	43,556	52,590	54,598	54,752	64,856	68,090	68,000	62,394	65,222	70,000
Total	44,559	53,287	55,288	54,752	64,856	68,090	68,000	62,394	65,222	70,000
Components of U.S. supply:										
Domestic mines	1,003	697	690							
Secondary	328	69	125	197	264	270	342	329	507	1,036
Shipments of Government stockpile excesses	6,007	5,162	1,683	5,945	8,569	8,936	6,346	6,698	148	
Imports	12,911	12,417	10,912	13,915	19,238	16,122	6,608	16,487	17,548	19,029
Industry stocks, Jan. 1	5,888	5,128	5,733	5,235	4,534	9,184	9,467	6,939	8,864	7,866
Total U.S. supply	26,137	23,473	19,143	25,292	32,605	34,512	22,763	30,453	27,067	27,931
Distribution of U.S. supply:										
Industry stocks, Dec. 31	5,128	5,733	5,235	4,534	9,184	9,467	6,939	8,864	7,866	6,019
Exports	1,497	1,478	365	1,293	1,398	1,348	1,768	1,751	891	1,548
Industrial demand	19,512	16,262	13,543	19,465	22,023	23,697	14,056	19,838	18,310	20,364
U.S. demand pattern:										
Nonmetal:										
Paints	1,285	2,152	2,042	2,323	2,646	2,807	1,965	2,943	2,578	2,273
Chemicals	1,382	538	707	1,270	1,635	1,959	1,523	1,783	1,469	1,681
Ceramics and glass	858	1,890	1,768	1,982	2,228	2,331	1,630	2,348	2,054	2,630
Total nonmetal	3,525	4,580	4,517	5,575	6,509	7,097	5,118	7,074	6,101	7,584
Metal:										
Transportation: Aircraft	4,046	3,297	2,518	4,294	4,023	5,290	2,636	3,458	4,310	4,580
Electrical	4,024	4,589	3,633	6,069	6,460	5,628	3,166	5,513	4,654	4,552
Machinery:										
Machine tools	1,911	1,882	1,376	1,717	2,449	2,853	1,356	1,537	1,567	1,605
Construction machinery	1,268	1,179	1,080	1,190	2,034	2,211	1,355	1,608	1,319	1,702
Total	3,179	3,061	2,456	2,907	4,483	5,064	2,711	3,145	2,886	3,307
Coating and plating	3,351									
Other	1,387	735	419	620	637	618	425	648	359	341
Total metal	15,987	11,682	9,026	13,890	15,603	16,600	8,938	12,727	12,209	12,780
Total U.S. primary demand (total demand less secondary supply)	19,184	16,193	13,418	19,268	21,759	23,427	13,714	19,509	17,803	19,328
Total U.S. demand for primary metal	15,659	11,613	8,901	13,693	15,339	16,330	8,596	12,398	11,702	11,744

[e] Estimated. [p] Preliminary.

West Germany. Exports of unwrought metal and cobalt-bearing scrap in 1978 totaled 1.55 million pounds, a substantial increase over that of 1977. This occurred partly because metal traders shipped significant quantities to Europe early in the year; the difference between the value of the U.S. dollar and the Belgian franc made this action profitable, since price adjustments based on currency realinement are not made until a certain minimum percentage change has occurred. Therefore, it was possible to buy cobalt for a lower price in the United States than in Europe.

Secondary Sources

Foreign processors recover significant quantities of cobalt contained in mixed scrap. However, with rising price of cobalt, domestic interest has heightened. There are only a few significant sources of cobalt-bearing scrap. One of the most important is waste material discarded in forming and fabricating cobalt-bearing superalloys. Moreover, high-temperature alloy components of jet engines are replaced periodically, providing a major source of recycled material.

A significant new source of secondary material could be spent petroleum catalysts, those used in desulfurization of crude oil. In 1978, one U.S. corporation attempted to recover cobalt from these catalysts but was unsuccessful, while another began producing significant quantities of cobalt oxide from other chemical wastes. Another significant development involving recycling was a 1977 announcement by GTE Sylvania of plans to produce ultrafine cobalt powder, by a proprietary method, for use in cemented carbide cutting tools, using secondary material as feedstock. The new operation is housed in a 25,000-square-foot addition to the company's existing plant in Towanda, Pa. Production began in 1978. Prior to this development, this special powder was mainly produced by Metallurgie Hoboken Overpelt in Belgium. Relatively small quantities were also produced in France and West Germany. Several companies are using a process developed and patented by the Bureau of Mines to recover the cobalt binder from cemented carbide scrap. In the process, molten zinc is used to dissolve the cobalt. The molten zinc is then recovered by distillation, and re-

fractory cobalt and the carbides remain in a mass that can be recycled. One company used this process to treat 750 pounds per day of scrap using a specially designed furnace. Several other companies are also using the process (U.S. Patent no. 3,595,484).

Other cobalt-bearing products are largely dissipated. Superalloy scrap is mostly reprocessed directly to make new alloys, rather than refined for its individual metallic content. Grinding swarf generated in the production of Alnico magnets is another source of material. The amount of cobalt in purchased scrap reported in 1978 was 1,036,000 pounds. However, taking into account plants that do not report scrap consumption, the actual amount is probably close to 1.5 million pounds per year.

Substitutes

Nickel can be substituted for cobalt in some uses with little change in end-product characteristics. Cobalt may also be substituted for nickel. For example, in 1969, during the nickel-mining strike in Canada, cobalt was effectively used in electroplating. However, the fact that nickel is not now generally used as a substitute for cobalt even though it is one eighth as expensive is evidence that use of nickel would result in a sacrifice of important properties or characteristics of end products made with cobalt. The current trend is toward greater use of alloys with less cobalt rather than elimination of cobalt altogether in the alloy. This is true in the fields of both hard-facing and wear-resistant applications, and superalloys. One example is the substitution of nickel-base Inconel 718 for the L605 cobalt-base alloy in a rocket propulsion system application. In certain cases, the alloys Udimet-500 or Udimet-700, containing about 18 percent cobalt each, might be replaced with nickel-base alloys with similar properties, but containing less cobalt. Certain iron base, heat-resistant alloys, such as N–155, can be substituted for cobalt-base materials, such as X–40 (HS–31), FSX–414, etc., in turbine applications. In aircraft turbine applications, Alloy 713C (no cobalt) might replace Waspalloy with 13 percent cobalt. A problem to be noted in any substitution is the replacement of one expensive and scarce material with another that may become equally or more scarce and expensive. Replacing colbalt-bearing alloys with alloys relatively high in columbium and tantalum is a good example in the superalloy field.

Substitutes for cobalt as a catalyst or as a dryer in paints are usually not effective. In dryer applications, manganese and lead each can act either as a complement or as a substitute. In catalytic applications, molybdenum and aluminum are complements, and nickel and tungsten together are substitutes for cobalt (7). However use of nickel in petroleum catalysts reduces octane and requires higher pressures to be effective.

The properties that cobalt imparts to permanent magnets exceed those that can be contributed by other elements. Any substitution for cobalt results in loss of weight advantage or other physical properties. Nevertheless, ceramic magnets, which are made from barium and strontium ferrite, are potentially strong competitors. Because ferrite magnets are relatively inexpensive, they are used in a number of automotive applications, and because of price and availability considerations, they will replace Alnico magnets in auto loudspeakers in 1980 models. However, because of the drive to conserve space and weight in automobiles, cobalt-rare-earth magnets may eventually replace ceramic ferrite magnets. In comparison with Alnico magnets, a net cobalt savings of 30 percent can be achieved by use of Cobalt-rare earth magnets. In magnetic usage, demand is more sensitive to price than in other uses such as superalloys, thus rendering replacement more likely.

Although ceramic magnets have made other inroads, particularly in other loudspeaker applications, they have not been used more extensively because of lower resistance to demagnetization (higher "magnetic leakage") and the high tooling cost required to produce intricate shapes. The lead time and cost of necessary redesign because of size and weight differences also prevent substitution in magnets, as well as other materials.

An alloy of manganese-aluminum-carbon, known as Almax in Japan, is considered to have great potential for replacement of some Alnico applications, and one Japanese company planned to begin marketing these magnets in the United States in 1980. Another relatively new magnet called Chromindur (chromium-cobalt-iron) may replace Remalloy currently being used in telephone receivers, resulting in an overall cobalt savings of about 15 percent.

No satisfactory substitutes have been developed for use of cobalt as a binder for carbides. However, with further experimentation, molybdenum-based tool steels, which can contain up to 12 percent cobalt, may achieve the performance characteristics of cemented carbides, or an equally suitable binder may be found. Recent

advances in powder metallurgy have afforded an opportunity to attain competitive performance levels. With research, it may also be found that alloys with a base of nickel, vanadium, chromium, or tungsten can be developed that have properties equal or superior to those of alloys containing cobalt.

Two principal alloy manufacturers announced in 1978 the development of effective substitutes for high-cobalt alloys in wear-resistant and hard-facing applications. One alloy reportedly achieves the same hardness (Rockwell C 40 to 42) and nearly equal performance characteristics as the major cobalt-base hard-facing alloys while using only about half as much cobalt; excellent wear and corrosion resistance were reported. Another alloy is said to maximize wear resistance with minimum utilization of cobalt. Typical applications are hard-facing augers, extrusion screws, valves, chain saws, and bearing areas.

Ceramic materials may eventually replace cobalt superalloys in gas-turbine engines. Recent research has resulted in the development of a silicon carbide composite material that combines the temperature and corrosion resistance of ceramics with the ductility and high fracture toughness of metals. However, this material has yet to be used commercially. Although ceramics reportedly have been flight-tested in jet engines, first use will probably be in land-based gas turbines in powerplants or possibly in automobiles. Because of the relatively low shock resistance of ceramics, many years will probably be required to develop a satisfactory ceramic material for aircraft use.

STRATEGIC CONSIDERATIONS

Cobalt is considered critical because of its uses (1) as a high-temperature-resistant alloy in jet engines, (2) as a binder in cemented carbides for metal cutting, (3) in mining tools (drill bits, valves, and pipes), hard facing (valves, chain saws, shovel teeth, scrapers, etc.) and applications in the steel industry, such as drawing and forming dies and rolls for the manufacture of steel, and (4) as a desulfurizing catalyst in refining and processing crude oil.

Cobalt is held in the National Defense Stockpile under authority of the Strategic and Critical Materials Stock Piling Act of 1946, as amended. Since that time, there have been several changes affecting the quantity of cobalt in storage. In 1973, the stockpile objective was lowered from 38.2 million pounds to 11.95 million pounds. Sales of cobalt from stockpile excesses were discontinued as of September 1976, and on October 1, 1976, the stockpile goal was raised to 85.4 million pounds. About 5,131,000 pounds of cobalt were sold from the stockpile in 1976, the last year of sale from excess stocks, leaving about 40.8 million pounds of cobalt in storage at year-end 1978. Any new purchases of cobalt from the stockpile will require congressional authorization.

Table 7. Time—price relationship for cobalt[1]

Year	Average annual price, dollars per pound	
	Actual prices	Based on constant 1978 dollars
1957	2.03	4.75
1958	2.00	4.60
1959	1.77	3.99
1960	1.54	3.41
1961	1.50	3.29
1962	1.50	3.23
1963	1.50	3.19
1964	1.50	3.14
1965	1.62	3.32
1966	1.65	3.27
1967	1.85	3.56
1968	1.85	3.41
1969	1.89	3.31
1970	2.20	3.66
1971	2.20	3.48
1972	2.45	3.73
1973	3.00	4.31
1974	3.46	4.54
1975	3.98	4.76
1976	4.44	5.05
1977	5.58	5.99
1978	11.53	11.53
1979 (September)	25.00	----

[1] Prices are weighted averages for each year (based on African Metals Corp. price list).

ECONOMIC FACTORS AND PROBLEMS

Pricing

Quoted cobalt prices have risen from $1.50 per pound in 1964 to $25 per pound in February 1979. During the last 5 years, prices have risen nearly exponentially. Tables 7 and 8 show changes in price that have taken place during this period. From 1973 to 1978, the price of cobalt rose an average of 30 percent per year. In terms of constant 1978 dollars, the price rose 22 percent per year over the same period. Total increases since 1973 have been 733 percent for actual price and 480 percent for price adjusted for inflation. Table 9 lists the prices quoted by African Metals Corp. for various grades and forms of cobalt.

One of the most important relationships in the cobalt market is cobalt's byproduct relationship with either copper or nickel and the consequent price inelasticity of supply. Selective mining of ores with higher than normal cobalt-to-copper ratios has taken place in Zaire, but the capability to mine selectively in this manner is limited. Increasing extraction efficiency can also

Table 8.—Cobalt-metal price changes

Year	Date of change	Dollars per pound[1]	
	Date	Actual	Based on constant 1978 dollars
1957	Annual	2.03	4.75
1971	Dec. 27	2.20–2.45	5.13–5.72
1973	Feb. 12	2.70	5.19
	Aug. 13	3.30	6.34
	Sept. 1	3.20	6.14
	Nov. 28	3.10	5.95
1974	May 16	3.45	5.51
	July 8	3.75	5.99
1975	Jan. 23	4.00	5.31
1976	Apr. 15	4.40	5.28
	Sept. 13	4.90	5.57
	Dec. 24	5.40	6.48
1977	Jan. 21	5.20	5.58
	July 13	6.00	6.44
	Dec. 21	6.40	6.87
1978	Feb. 24	6.85	6.85
	May 24	8.50	8.50
	July 20	12.50	12.50
	Sept. 14	18.00	18.00
	Oct. 27	20.00	20.00
1979	Feb. 1	25.00	25.00
	Annual growth rate:		
	5 years (1973–78) ..... percent	31	22
	20 years (1958–78) ..... do	9	5

[1] Dollars per pound of cobalt granules in 550-pound drums.

increase production, but the byproduct nature of cobalt places an upper limit on the amount of cobalt that can be produced. The price range at which supply would be greatly sensitive to price changes is unknown. Although the current cobalt price may have elevated cobalt to the status of coproduct in some copper mining operations, this situation is not expected to greatly alter the basic production capacity for cobalt at these locations.

The price structure plays a critical role for two reasons. First, an increase in price may result in reduced demand in nonstrategic areas, where demand is more responsive to price fluctuations. Secondly, greatly higher prices will both promote research into effective substitutes and encourage new production of primary and secondary material. The latter is already a distinct possibility in Idaho, where renewed mining reportedly is being considered. The cobalt price leader is generally SOZACOM, the marketing arm of Gécamines in Zaire.

In part, price rises have reflected the cost of mining and refining in foreign countries. Aside from inflation, other factors that have affected the price of cobalt since 1964 are (1) the removal of cobalt from a list of commodities prohibited from being shipped to Sino-Soviet nations (1964), (2) a 39-percent increase in U.S. industrial cobalt consumption as a result of the Vietnam war (1963–69, during the period of escalation), (3) the announced nationalization of mines in Zaire (1967), (4) increased demand because of the nickel shortage caused by the Canadian nickel strike (1969), (5) currency realinement between the United States and Belgium (1972–78), (6) halt in sales of excess cobalt from the national stockpile (1976), and (7) reduced copper and nickel production and poor extractive efficiency combined with unusually high demand, creating a tight supply situation. The price increases in May, July, and September 1978 reflected unusually high demand and lack of readily available effective substitutes.

The export revenue generated by cobalt in Zaire and Zambia increased from less than 10 percent of all mineral export earnings early in 1978 to more than 30 percent early in 1979. Moreover, total value of cobalt production in 1979 was expected to approach if not equal that of copper production in Zaire and Zambia. The free market or "gray" market is the cobalt offered for sale by dealers who do not represent producers. Virtually no such market existed prior to the 1977–78 currency realinements and allocation. When the allocation first became effective in May, 1978, free market prices were quoted at $10 to $15 per pound. By yearend, this range had risen to $40 to $50 per pound. The quantities of cobalt estimated to be sold at the free market price comprised probably no more than 15 percent of the total cobalt sold in the United States during the year.

The opening of new sources should have a dampening effect on price in the long run. For example, a breakthrough in the mining of cobalt-bearing manganese nodules could moderate the price considerably because of the potentially large supply, as could the expected mining of nickeliferous laterite deposits in New Caledonia, Indonesia, and the Philippines, and of sulfide and oxide deposits in Zambia.

The cost of all facets of mining and refining is also an important factor in setting the price of cobalt. In addition, the market price of cobalt is inseparably linked to the production of the primary product, whether it be copper or nickel. For example, increased demand for the primary product might result in an increase in the supply of byproduct cobalt, thus lowering its price.

Taxes and Tariffs

No tariffs are levied on cobalt ore and concentrates or unwrought metal, waste, and scrap. As of January 1979, there was a 9-percent-ad valorem duty on unwrought cobalt alloys and a 1.2-cent-per-pound duty on cobalt oxides. A duty of 3.6 cents per pound is imposed on cobalt linoleates. For alloys, oxides, and linoleates, statutory duties are 45 percent ad valorem, 20 cents per pound, and 14.5 cents per pound, respectively. Domestic mining operations are permit-

Table 9.—Cobalt price schedule February 1979 [1]

Product	Size	Unit[2]	Dollars per pound
Metal:			
Granules (shot), about 99.5 percent cobalt	⅛ to ¾ inch	250-kilogram (551.15 pound) drums	25.00
		50-kilogram (110.23-pound) drums	25.10
		Less than 50 kilograms	25.20
Electrolytic cathodes (broken) about 99.9 percent cobalt	¼ to 2 inches	250-kilogram (551.15-pound) drums	25.00
		50-kilogram (110.23-pound) drums	25.10
		Less than 50 kilograms	25.20
Powders:			
Coarse	300 mesh, 400 mesh (regular), and 100 mesh.	50-kilogram (11.23-pound) drums	27.92
		10-kilogram (22.046-pound) tins	28.20
Extra fine	Less than 1.6 microns	125-kilogram (275.58-pound) drums	31.00
		50-kilogram (110.23-pound) drums	31.09
		50-kilogram (110.23-pound) cases	31.37
		5-kilogram (11.023-pound) tins	31.77
Oxide:			
70 to 71 percent cobalt (black)			[3] 19.91
72 to 73 percent cobalt (black)			[3] 20.45
Metallurgical-grade (gray) about 76 percent cobalt			[3] 21.11
Fines	300 mesh	100-kilogram (220.46-pound) drums	27.49

[1] African Metals Corp.
[2] F.o.b. Port of New York, N.Y. or Chicago, Ill.
[3] Less 3 cents per pound of quantities over 5,000 pounds.

ted a depletion allowance of 22 percent (14 percent for operations in a foreign country).

TOXICITY

The National Institute for Occupational Safety and Health (NIOSH) recommended in September 1977 that, when the cobalt content of cemented carbide dusts exceeds 2 percent, the maximum exposure level standard for such dusts should be 0.1 milligram per cubic meter of air, determined as a time-weighted average concentration for up to a 10-hour workshift in a 40-hour workweek. This is the current Federal standard for occupational exposure to cobalt (*14*).

Toxic effects have been observed from therapeutic administration of cobalt for certain types of anemia. A single oral dose of up to 500 milligrams of cobalt chloride has resulted in severe toxic effects, while a number of more mild symptoms have been caused by intravenous injection of 5 to 10 milligrams daily. In some patients, cobalt has been shown to have a toxic effect on thyroid function. In addition, various forms of dermatitis may be caused by cobalt and cobalt salts (*6*).

Continuous inhalation of fine cobalt dust may, after long exposure, cause chronic bronchitis with shortness of breath; however, the condition is reversible if exposure to the dust is terminated (*19*). No standard for maximum concentration of cobalt in water has been established.

OUTLOOK

Demand

The primary demand for cobalt in the United States for the year 2000 is forecast at between 30 and 49.7 million pounds, corresponding to growth rates of 1.8 and 4 percent annually from 1977. Based on an analysis of contingencies assumed for end-use components of the two growth rates, the probable primary domestic demand in 1985 and 2000 is forecast at 26.3 and 42.6 million pounds, respectively, representing a growth rate of 3.3 percent.

The rest-of-the-world demand for primary cobalt is forecast at between 48 and 77.7 million pounds in 2000. The probable rest-of-the-world primary demand in 1985 and 2000 is forecast at 39.2 and 65 million pounds, respectively, corresponding to a growth rate of 3.4 percent. The primary demand growth rate for cobalt in the rest of the world is expected to about equal that of the United States as a result of accelerating expansion of high technology in Japanese industries, the rising use of superalloys in Western Europe, and the use of cobalt alloys required for efficient industrial development in other countries.

Evidence that this trend has already begun is the expansion of European aircraft and electrical industries and the relatively large apparent consumption of cobalt in the rest of the world in recent years.

The cumulative domestic requirement for primary cobalt from 1977 to 2000 is estimated at 699 million pounds. The cumulative demand for primary cobalt in the rest of the world is forecast at 1.06 billion pounds, bringing the world probable cumulative demand throughout the forecast period to 1.76 billion pounds. The potential supply of primary cobalt in the world, from reserves only, is 3.2 billion pounds at 1978 prices, easily adequate to meet this potential demand.

Reserve figures are not static, but constantly

Table 10.—Summary of forecasts of U.S. and rest-of-world cobalt demand, 1985 and 2000

(Thousand pounds)

		2000 Forecast range		Probable		Probable average growth rate 1977–2000 (percent)
	1977	Low	High	1985	2000	
United States						
Primary	17,803	30,000	49,700	26,300	42,600	[1] 3.3
Secondary	507	2,500	3,500	900	3,000	8
Total	18,310	32,500	53,200	27,200	45,600	3.5
Cumulative (primary)[1]	----	577,000	766,000	187,000	699,000	----
Rest of World						
Primary	[r] 30,000	48,000	77,700	39,200	65,000	3.4
Secondary	1,000	3,800	5,500	2,800	5,000	7.2
Total	31,000	51,800	83,200	42,000	70,000	3.6
Cumulative (primary)	----	894,000	1,170,000	280,000	1,060,000	----
World:						
Primary	47,803	78,000	127,400	65,500	107,600	3.4
Secondary	1,507	6,300	9,000	3,700	8,000	7.5
Total	49,310	84,300	136,400	69,200	115,600	3.6
Cumulative (primary)	----	1,471,000	1,936,000	467,000	1,759,000	----

[r] Revised.
[1] Calculated on basis of the 20-year primary demand trend value of 20,118 for 1977 for the United States.

Table 11.—U.S. cobalt demand by end use in 1977, and projections and forecasts for 2000

(Thousand pounds)

		2000			
			Contingency forecasts for United States		
			Forecast range		Probable
End use	1977	Statistical projection[1]	Low	High	
Metal:					
Transportation: Aircraft	4,310	7,544*	6,800	9,600	8,100
Electrical	4,654	11,973*	8,700	14,000	13,500
Machinery:					
Machine tools	1,567	4,618*	2,700	5,800	5,500
Construction machinery	1,319	2,556*	1,900	4,100	3,600
Other	359	2,805*	600	800	800
Total	12,209		20,700	34,300	31,500
Nonmetal:					
Paints	2,578	10,395	4,100	7,200	5,000
Chemicals	1,469	6,935	4,100	6,500	4,100
Ceramics and glass	2,054	4,390*	3,600	5,200	5,000
Total	6,101	----	11,800	18,900	14,100
Grand total	18,310	----	32,500	53,200	45,600

[1] Statistical projections are derived from regression analyses based on historical time series data, and forecasts of economic indicators, such as GNP, FRB index, etc. Projection equations with a coefficient of determination (R-squared) less than 0.70 are indicated by an asterisk (*).

change with price and exploration. Also, for various reasons mining companies often do not attempt to prove reserves beyond a 10 to 20-year expected production period. Therefore, any projection of reserve exhaustion 20 or 30 years in the future would be invalid. However, because of cobalt's byproduct character, supply of refined metal will be available only at a rate of 40 to 60 million pounds per year unless major new copper and nickel mining projects are undertaken. This supply will cover the high of the range of forecast total world demand for about 8 years from 1978. Therefore, the rate of annual production must be increased by about 1985. Periodic shortages may take place in the intervening years because of reduced copper and nickel production, surges in demand, and political or military interruptions of production.

Forecasts of U.S. and rest-of-world demand for cobalt in 1985 and 2000 are summarized in table 10. Forecast ranges for U.S. demand by end use in 2000 are presented in table 11. A comparison of domestic cobalt production from 1957 to 1978 and projected production and demand for 1985 and 2000 is given in table 12. Figure 3 shows plots of historical data and projections of supply and demand to the year 2000.

The forecast of each end use in the year 2000 was derived from statistical regression analysis, based on the historical end-use data from 1960 through 1977. The statistical results indicate that the best-fit explanatory variables for each end use are transportation (Federal Reserve Board (FRB) production index of iron and

steel); electrical (gross private domestic investments (GPDI)); machine tools (FRB production index of special and general industrial machinery); construction machinery (FRB production index of construction and allied equipment); other (FRB production index of special and general industrial machinery); paints and chemicals (FRB production index of chemicals and products); and ceramics and glass (FRB production index of stone, clay, and glass products). Judgment factors were applied to arrive at a probable value for the year 2000.

The probable 2000 forecast of 8.1 million pounds in table 11 for aircraft demand was based on historical growth patterns of iron and steel. (Historically, there appears to be a correlation in growth patterns between steel and superalloys, but the reason for this relationship is not clear.) The probable forecast of 13.5 million pounds for electrical equipment and supplies was based on historical growth patterns in private domestic investment. For cobalt use in machine tools (cutting tools), the probable forecast of 5.5 million pounds was also derived by relating the trend to the historical growth pattern in special and general industrial machinery. Demand for cobalt in construction and related machinery, based on historical patterns of the FRB index of construction and allied equipment, plus estimates of unusually high demand because of energy-related applications, indicates a probable forecast of 3.6 million pounds of cobalt in 2000. In paints and allied products, and chemical products, the demand for cobalt, when related to construction, is expected to reach 5 and 4.1 million pounds, respectively. The demand for cobalt in ceramics is expected to grow with the total population at 0.9 percent and reach 5 million pounds by 2000.

Supply

Currently, no cobalt is mined in the United States. However, continuation of a relatively high price for cobalt may result in mining of one or several of the higher grade domestic deposits. Nevertheless, no production is expected before 1980. The supply of cobalt is relatively inelastic with respect to its price. However, depressed prices and resultant decreased production of copper and/or nickel can result in a decline in cobalt supply.

Future supply depends largely on foreign political conditions and circumstances, so forecasting is difficult. However, current and historical data provide a general indication of market behavior for several years in the future. At present rates of production, total world re-

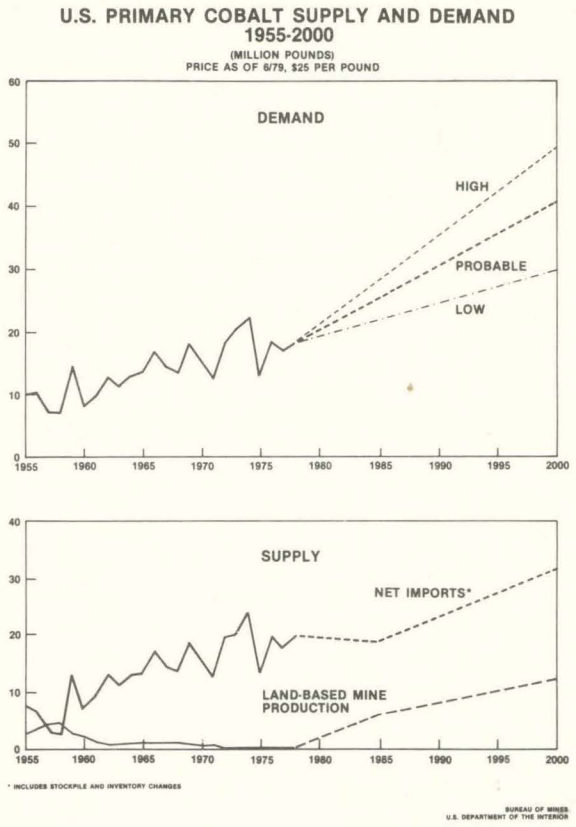

Figure 3.—U.S. primary cobalt supply and demand, 1955–2000.

Table 12.—Comparison of domestic cobalt demand and production 1957–78, 1985, and 2000

(Thousand pounds)

	U.S. primary demand	Domestic primary production
1957	11,914	4,144
1958	7,542	4,844
1959	15,595	2,994
1960	8,930	2,247
1961	11,563	1,045
1962	13,622	721
1963	12,002	948
1964	13,831	1,084
1965	14,297	1,186
1966	17,900	1,215
1967	15,220	1,168
1968	14,441	1,176
1969	19,184	1,003
1970	16,193	697
1971	13,418	690
1972	19,268	0
1973	21,759	0
1974	23,427	0
1975	13,714	0
1976	19,509	0
1977	17,803	0
1978	19,335	0
1985	[1]26,300	[2]0 [3]6,000
2000	[1]42,600	[2]0 [3]10,000

[1] Probable forecast from table 10.
[2] 20-year trend.
[3] Estimated possible production from land-based sources.

serves would last 50 years. If all known resources are included (excluding cobalt in deep sea manganese nodules), world supplies would last an additional 85 years. Therefore, world reserves may be considered at least adequate for the near future. Moreover, over the next 3 years, capacity of current operations is sufficient to meet world demand, which is projected to grow about 3.5 percent annually. Even so, without a commensurate growth rate in copper and nickel, supply of refined metal has not been and may continue not to be adequate to meet demand. It is also necessary that refining facilities be operated at maximum efficiency at capacity levels to meet demand.

New projects may include very large laterite developments in Indonesia, New Caledonia, and the Philippines, where cobalt may be mined as a byproduct of nickel. Grade of ore, suitability for processing, and price are the factors that determine whether byproduct cobalt is extracted. Another potentially significant source is the Tenke-Fungurume project in Zaire (stated capacity, 14 million pounds per year of cobalt), which was postponed indefinitely in early 1976 because of financial difficulties. There were indications in mid-1979 that the project would be revived. With a higher price for cobalt, additional metal should be produced by Zambia, Finland, Canada, and Australia, making it less necessary, strictly from the standpoint of total world supply, to develop alternative sources. The Republic of South Africa is also a potentially significant new source. However, by far the most important potential source is cobalt-bearing manganese nodules from the Pacific Ocean. The potential exists to mine enough cobalt from the ocean floor to meet the total U.S. demand in the year 2000. If all of the various environmental, legal, and technical obstacles can be overcome, the nodules could dramatically alter the supply situation.

Possible Supply-Demand Changes

Although no domestic primary cobalt was produced in 1978, it is possible that with high cobalt prices during the next 5 years, domestic mining may again become profitable. The figure of 6 million pounds given for U.S. production in 1985 (table 12) includes a potential estimated 3 million pounds from the Blackbird sulfide district in Idaho, 2 million pounds per year from the lead belt of southeastern Missouri (including the former Madison mine), and 1 million pounds per year from the Duluth gabbro in Minnesota.

Because of energy shortages and the consequent increased rate of oil and gas drilling, as well as expanded coal mining, the demand for metals, including cobalt, that are used in these operations has increased. In sour-gas applications, an alloy with 35 percent cobalt is being used. Other related uses for cobalt are the hardfacing of processing equipment, and power shovel teeth, as a binder in tungsten carbide rock-bit cutting inserts and nozzles for drilling, and in diamond drill bits. In addition, gradual depletion of low-sulfur crude oil reserves and more stringent environmental standards have increased usage of cobalt molybdate catalysts used to remove sulfur from light petroleum distillates targeted for the gasoline pool.

As a result of the drive toward increasingly lighter weight vehicles, the auto industry reportedly is considering replacing ferrite magnets with the more powerful but more expensive cobalt-rare-earth magnets. The cobalt magnets exhibit a combination of properties that are required for automotive use, including high strength, high resistance to demagnetization, and good temperature stability. Moreover, because the magnets have a pulling strength several times that of conventional magnets, both space and weight may be conserved through their use. In the 1980's, new uses for magnets may include fan and alternator motors, tire pressure gages, remote trunk releases, and seat actuators. However, problems of availability and price may prevent use of cobalt in this field. For example, because of these considerations, the major auto manufacturers decided early in 1979 to replace cobalt-bearing Alnico magnets in loudspeakers with ferrite magnets. A relatively new manganese-aluminum-carbon alloy may also be used as a substitute for some Alnico applications in the future.

Commercial aircraft manufacturing is expected to increase significantly over the next decade, as aging commercial airline fleets are replaced. The average age of commercial aircraft presently is about 8 years. Obsolescence and stricter antinoise regulations, which must be met by 1985, provide incentives to build new aircraft and engines. This growth has already been reflected in reported cobalt consumption in superalloys, which exceeded the 1977 figure by 16 percent. There was also a significant gain in deliveries of aircraft in 1978 over that of 1977, as compared with a decline from 1976 to 1977. In addition to military aircraft, orders for commercial aircraft reportedly have surged and are a major part of the current boom. Another intriguing development in transportation that

could significantly affect the demand picture is the possibility of use of cobalt in nickel-zinc batteries in electric vehicles (*9*). In addition, use of a cobalt catalyst in the production of synthetic fuel could significantly increase demand.

One possible major change in the geographical distribution of the cobalt supply is development of the laterite deposits in Oceania, which are expected to become productive over the next 10 years. Included are deposits in Australia, New Caledonia, Indonesia, and the Philippines. Expansion of facilities at Chambishi in Zambia should significantly increase supply available from that country. There is also a significant potential for production of cobalt in South Africa, as a byproduct of platinum and gold mining operations.

Supply and demand of cobalt are expected to remain in balance over the long term to 2000. However, in the interim, pending commercial exploitation of cobalt-bearing manganese nodules and greater development of laterite deposits, demand could exceed supply. A weak copper and nickel market has resulted in curtailment of some cobalt expansion projects, in spite of a rise in cobalt demand. This has contributed to relatively high prices. Such a situation can develop because the major end uses of copper, nickel, and cobalt are somewhat independent of each other. The basic by product relationship is expected to continue to affect supply.

Cobalt-bearing manganese nodules are not expected to be a major source for some years. Three factors could hasten development: (1) An increase in land-based mining and manpower costs, (2) producer nations using their supply as a lever in negotiating political or financial concessions from consumer nations, and (3) unilateral price increases not strictly based on increased costs. The latter situation occurred in mid-1978, as the smaller dealers and all of the producer nations took advantage of strained market conditions, aggravated by the invasion of Shaba Province in Zaire, to raise prices dramatically. The fact that cobalt demand was relatively high at that time, and not greatly sensitive to price, encouraged an escalation of the price. However, as a result, considerably more interest in recycling, substitution, and new production was generated. For example, the amount of cobalt recycled in 1978 approximately doubled compared with that of 1977, and the likelihood of domestic mine development was increased considerably. Recently, there has been increased interest in renewed domestic mining of cobalt-bearing deposits in Idaho, southeastern Missouri, northern California, and Minnesota. In some locations, concrete steps toward development have been taken.

Secondary cobalt could become a significant source of supply since the demand has exceeded the primary supply presently obtained from traditional sources. However, recycling of scrap probably cannot supply more than about 10 percent of demand because of dissipation, which is a problem in magnetic and hard-facing applications, as well as in the nonmetallic applications. Also, because of the relatively long life of jet engines and other products containing cobalt, and because demand continues to expand, only a small portion of current demand will be met by recycling the relatively small quantity of used products made 5 or more years ago.

Besides secondary material, large percentages of cobalt metal in low-grade cobalt ore are retained in slag, which is frequently stockpiled. For example, slag dumps in Zaire and Zambia, containing up to 1 percent cobalt, have not yet been utilized. Under prevailing favorable economic conditions, more cobalt in slag may be recovered, and this could represent a more important source. In addition, in 1978, Zambia's largest cobalt producer recovered only about 30 percent of the cobalt from the ore. With a higher price for cobalt, metallurgical processes can be adjusted to recover a greater percentage of cobalt from the concentrate.

Possible Technological Progress

Perhaps the most significant cobalt research in progress is in the development of techniques for mining and processing cobalt-bearing manganese nodules from the ocean floor. Many patents have been issued on all phases of the new technology. There is a debate as to whether hydrometallurgical or pyrometallurgical techniques would be more effective and economical. The principal factors to be weighed are energy requirements and reagent costs. Pyrometallurgical processes are basically energy intensive. Moreover, the smelting of nodules requires the production of manganese, and because wet, impure 30-percent-manganese nodules are inferior as an ore to 45- to 50-percent-manganese terrestrial material, the smelting method is considered by many not to be economically feasible. In hydrometallurgy, three main processes are being studied. These are sulfuric acid (H_2SO_4), hydrochloric acid (HCl), and ammoniacal leaching. For the ammoniacal leach procedure, investment and reagent costs reportedly are relatively low, while energy requirements are high, compared with those of the sulfuric acid process.

The sulfuric acid process would be more attractive if reagent costs could be reduced. The HCl process would be more attractive with a higher manganese price. In addition, material-investment costs in this process are high because of the need for materials with high corrosion resistance. (For complete review of these techniques, see reference *2*.) In one leaching process, an ammonium carbonate solution containing cuprous ions is contacted with the nodules in order to leach the metal content. Cobalt is recovered along with molybdenum by chemical means after nickel and copper are recovered by liquid ion exchange and electrowinning. Manganese is rejected to the tailings, or incremental amounts are recovered depending on market conditions (*1*).

The largest single cost item in lateritic nickel extraction is energy. A U.S. company developed a process that uses sulfur as a source of primary energy, with sulfuric acid byproduct credits. Using this process, the garnieritic fraction of lateritic ore bodies may be economically treated by a combination of high pressure and atmospheric pressure leaching. Residual acid in the solution is neutralized by soluble magnesium from the garnierite fraction. Part of the solution is treated to precipitate nickel in the nickelous state. The nickelous precipitate is oxidized, and the oxidized nickelous precipitate is mixed with the rest of the starting solution to oxidize cobalt to the cobaltic state. The resultant precipitate is separated for recovery of cobalt, while the residual solution having a nickel-to-cobalt ratio greater than 2,000:1 is processed for nickel. This technical advance may enable development of laterite deposits that were previously considered uneconomic because of excessive energy costs. AMAX, in a joint venture with the French Bureau de Rechérches Géologiques et Minières (BRGM), plans to use the new process on New Caledonian ores.

Deep-seabed nodules containing manganese, nickel, copper, and cobalt are present over wide areas of the ocean floor of both hemispheres. Deposits of probable economic significance have been delineated by sampling, photography, and traverses of favorable areas with television cameras. The composition of the nodules is variable both locally and regionally.

Current estimates of total resources, 76 billion tons of nodules, must be considered highly speculative until the data base for this emerging resource is improved. However, available data indicate that the resource is widespread and large by any standard. The continuing collection of data on nodule occurrences and their metallic grades in all oceans and the more detailed resource assessment work underway in potential mine site areas will eventually lead to a better understanding of the magnitude of this resource.

Many analyses of nodules from the Pacific Ocean have been published. Representative grades for the four metals appear to be approximately as follows: 24 percent manganese, 1.0 percent nickel, 1.0 percent copper, and 0.35 percent cobalt. A mining operation with a capacity to produce 1 million dry tons of nodules per year could supply a processing plant with 240,000 tons of manganese, 10,000 tons each of nickel and copper, and 3,500 tons of cobalt. At this rate, at 100-percent recovery, each deep-seabed mining operation of similar size could supply the following percentages of 1978 primary U.S. demand: Manganese, 17; nickel, 5; copper, 0.5; and cobalt, 36.

Discussions were continued on international cooperation and jurisdiction over these nodules at the Seventh and Eighth Sessions of the Third United Nations Conference on the Law-of-the-Sea, held during 1978 and 1979. The United States will continue in the effort to construct a framework of international cooperation in the oceans. Domestic legislation on deep-seabed mining that would permit U.S. companies to proceed, was considered by committees having jurisdiction in the 96th Congress.

A distinct possibility is the development of better methods to separate cobalt from nickel when the two are in chemical solution, as is the case when processing laterite ores. Also, improvement in beneficiating complex cobalt-copper-arsenic sulfide ores may lead to economical treatment of cobalt ores of the Blackbird district of Idaho. The Bureau of Mines is conducting research on extractive processes for treatment of laterites in Oregon, sulfides in Minnesota, and lead ores in Missouri. These processes may be found economically feasible in the 1980's, and therefore some domestic production may result. In addition, greater use of refractory metal scrap in the United States is anticipated because of rising price, the increased use of cobalt in superalloys, and the necessity of recycling the scrap generated during forming and machining.

Over the long term, it is conceivable that ceramics and composites will replace cobalt in many applications, including superalloys, cutting tools, and magnets. While the raw materials cost of ceramics is much lower than that of cobalt, fabrication can be more expensive. More-

over, equal performance characteristics may prove impossible to achieve.

Possibly significant is the development of a cobalt-bearing automotive-emission control catalyst that works with leaded gas. About 80 percent of the B cations in the catalyst, having the general formula ABO_3, are cobalt (*13*). Possibly even more significant is research being carried out on a cobalt-bearing, nickel-zinc battery. If electric vehicles become popular and this battery is adopted, significant expansion of cobalt demand could result (*9*). In addition, with greater attention being focused on the development of synthetic fuels, use of a cobalt catalyst in the Fischer-Tropsch synthesis of hydrocarbons could lead to much greater use of cobalt. This catalyst was used by Germany during World War II to produce gasoline and oil products. Although similar iron catalysts are cheaper and more readily available, cobalt catalysts have the advantage of a longer life with virtually no change in composition during synthesis (*21*).

The development of the maraging family of steels, with their high strength and toughness and the development of rare-earth-cobalt magnets are expected to be two additional significant areas of research and development in the future. The demand for cobalt should increase as new products based on this and other research are developed and marketed.

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U.S. Bureau of Mines publications:

Mineral Commodity Summaries (annual).
Mineral Industry Surveys (monthly and annual).
Minerals Yearbook (annual).
Mineral Trade Notes (monthly).
Minerals and Materials/A Monthly Survey.

Other sources:

American Metal Market.
Engineering and Mining Journal.
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World Mining.

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As the Nation's principal conservation agency, the Department of the Interior has responsibility for most of our nationally owned public lands and natural resources. This includes fostering the wisest use of our land and water resources, protecting our fish and wildlife, preserving the environmental and cultural values of our national parks and historical places, and providing for the enjoyment of life through outdoor recreation. The Department assesses our energy and mineral resources and works to assure that their development is in the best interests of all our people. The Department also has a major responsibility for American Indian reservation communities and for people who live in island territories under U.S. administration.



SOUTH AFRICA

Vaal Reefs to mine Afrikander Lease U_3O_8 property

The Afrikander Lease Ltd.'s uranium and gold mine will be put into production by the end of this year. The project has been under study for several years but was shelved in early 1978 when cost inflation and changes in tax laws made earlier, favorable economic projections invalid. The decision to go ahead again is based on an arrangement whereby **Vaal Reefs Exploration and Mining Co. Ltd.** will mine the Afrikander Lease property in return for a royalty based on revenue.

Both companies are in the **Anglo American Corporation of South Africa Ltd.** group. Anglo American says that over a wide range of uranium prices, production levels, and other operating parameters, the royalty arrangement should give Afrikander Lease a better return than if the company were to raise the capital to finance an independent mine.

Afrikander Lease is in the Klerksdorp area of the Western Transvaal as is Vaal Reefs. Uranium and gold ore was mined on the property for a short period around 1960. There is shaft access and underground development.

Underground mining and extensive diamond drilling have shown that there are two uranium bearing conglomerates, the Upper and Lower Reefs, at the base of the Dominion Reef group. The reefs crop out in the north of the area and are folded into a syncline plunging south. The reefs are separated by quartzite, which varies in thickness from virtually nil to 30 meters (100 feet) and reach about 1,200 meters in depth (4,000 feet) at the south limit of the area drilled.

The Upper Reef is the main uranium ore body and has two relatively high grade shoots in addition to large tonnages of lower grade material. High uranium values for the Lower Reef are present only in a limited region but some of the gold values are high.

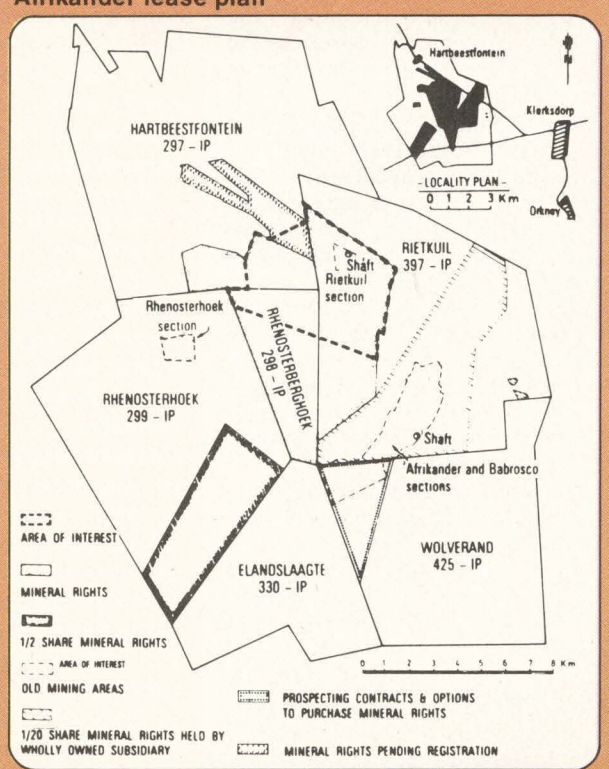
In order to get into production as soon as possible, mining will start in December 1979 and the ore will be transported to Vaal Reefs for treatment. Production initially will be at the rate of 15,000 metric tons per month. Vaal Reefs will build a new treatment plant for uranium and gold recovery at the Afrikander Lease mine. The new plant, with a capacity of 50,000 tons milled per month, will be commissioned in the first half of 1981 and achieve rated production about a year later. The Afrikander Lease reserves within the main block leased to Vaal Reefs could support a much larger operation so the plant is designed in a modular manner for easy expansion.

The 50,000-ton-per-month mill would have an average annual production of 385 metric tons of uranium oxide and 460 kilograms (about 15,000 ounces) of gold. This represents yields of 0.64 kilogram (1.4 pounds) uranium and 0.77 gram (0.02 ounce) gold per ton. This production level is more modest than that originally envisaged for the independent Afrikander Lease mine which was planned to mill 90,000 tons per month with a yield of about 0.7 kilogram uranium per ton to produce about 750 tons of uranium annually. However, by sinking additional shafts and expanding the treatment plant, production could be raised to 120,000 tons per month.

The value of the lease arrangement is that Afrikander Lease would only pay tax on the royalty it received at the normal company rate—much lower than the gold and uranium mining tax rate. Vaal Reefs, on the other hand, would not have to pay state's share of profits on the new mine, which profit would be taxed at the rate applicable to a post-1973 gold mine in Vaal Reefs hands, after taking the royalty into account. Very importantly, Vaal Reefs will be able to obtain immediate tax relief on capital expenditure made on the project. It can deduct this expenditure from its profit on existing mining operations and thus obtain a much better return on capital investment than could Afrikander Lease, which has no profit to offset.

It is reported that Vaal Reefs will spend an initial $70,000,000 on underground development and treatment plant construction to achieve the first phase of the project. Before the project was suspended last year Anglo American had been seeking sales contracts for the uranium production.

Afrikander lease plan



Where it is found, who produces the ore, how reserves are distributed, why it is in short supply, where will future supplies come from, what is it used for

Cobalt

Cobalt is today's most sought after metal. In 1978 the producer price of cobalt rose from $6.40 to $20.00 per pound, a 213 percent increase. Now, in 1979, some major producers have set a price of $25.00 per pound and the free market price is about $42.00 per pound.

The supply-demand balance of cobalt has always been in very fine equilibrium because of two factors. On the supply side there are very few producers, the production is highly concentrated geographically, and production is virtually entirely a byproduct of mining copper and copper-nickel ores. On the demand side the principal uses of cobalt are ones for which no adequate substitute can be found in the short or medium

by **Robert J. M. Wyllie**
Editor

term. Thus any disruption of the equilibrium tending to decrease the supply will send the price through the roof. This happened in May 1978 when a rebel army invaded Zaire's Shaba province.

Zaire produces about 60 percent of the western world's cobalt and all this production comes from the Shaba copper mines. The invasion did not last long, and the invaders did not do much damage to the mines and plants. What they did was to murder many of the expatriates Gécamines employed to fill key technical posts on the mines. Before the invasion there were about 650 expatriates plus their families employed in the Western Group copper-cobalt mines. After the invasion and the subsequent flight of the survivors there were none. Today there are reportedly about 100 expatriates at Kolwezi. Gécamines considers it needs about 250 to assure operations.

It appears that the expatriates are needed not so much for the daily production tasks as for the continuing maintenance of the complicated mining and processing facilities. Thus immediately after the invaders were driven out Gécamines was able to announce that production had been restarted at practically pre-invasion levels. This claim was received with some incredulity, but it was probably near the mark. What is more likely is that production is now in a steady decline as vital maintenance procedures are neglected through lack of qualified staff.

Zairian sources claimed that the country's cobalt production in 1978 was 13,000 metric tons (against 10,200 tons in 1977). Most independent estimates give Zaire a 1978 production of about 11,000 to 12,000 tons. The Zambians, who as number three producers have a vested interest in minimizing Zaire's capability, have estimated that their northern neighbor's 1978 production could only be about 8,000 tons.

What is very significant is that Gécamines has admitted that in January and February of 1979, cobalt production was 200 and 430 tons below target respectively. These shortfalls were attributed to neglected maintenance. Gécamines has a target production of 13,000 tons in 1979 but outside observers consider that actual production will only be 10,000 tons or even less.

No one believes that the cobalt market crisis will ease in the course of 1979, nor for some time beyond. Although other producers are scrambling to raise production, their short term contribution cannot be very great. In addition, Zaire's share of supply is so great, and its economic and political position so parlous, that the imponderables about the level of Zairian production must outweigh all other considerations.

Looking at the global picture, world cobalt mine production in 1977

Cobalt Report Published

What is the connection between Ming porcelain and jet turbines? Full marks if you answered cobalt. The same element which imparts the fragile blue glaze to the one gives the strength to withstand searing operating temperatures to the other.

Several hundred years ago Saxon metallurgists in the Harz mountains cursed the goblin Kobold whom they held responsible for making certain copper ores refractory. Today metallurgists eagerly seek ways to recover cobalt from mixed ores.

Political disturbances in Africa have disrupted supplies of this essential element and sent prices soaring to record peaks. Never before has attention been so firmly focussed on all aspects of cobalt from geology through end use.

An extraordinarily detailed study of the world of cobalt has just been published. This is *Cobalt. Cobalt* is an English language version of a report prepared by scientists, statisticians, and researchers at the West German government's Raw Materials and Economic Research Institutes. WORLD MINING has secured an exclusive publishing arrangement with the German organizations to publish an English language edition of the original report.

The article on the next few pages dips into the report to give the WORLD MINING reader a general picture of the cobalt industry from the point of view of distribution of reserves, production, mine output, metal output, and uses.

For more details on the contents and how to get further information about the report, see the announcement on page 29 of this issue of WORLD MINING.

amounted to 26,360 tons. The countries of the western world produced 90 percent of this; more than 50 percent came from Zaire, 13 percent from Australia, and barely 10 percent from Zambia. In the eastern block, mined cobalt production came about equally from the Soviet Union and Cuba. The average yearly growth of cobalt production over the past decade has been 4 percent.

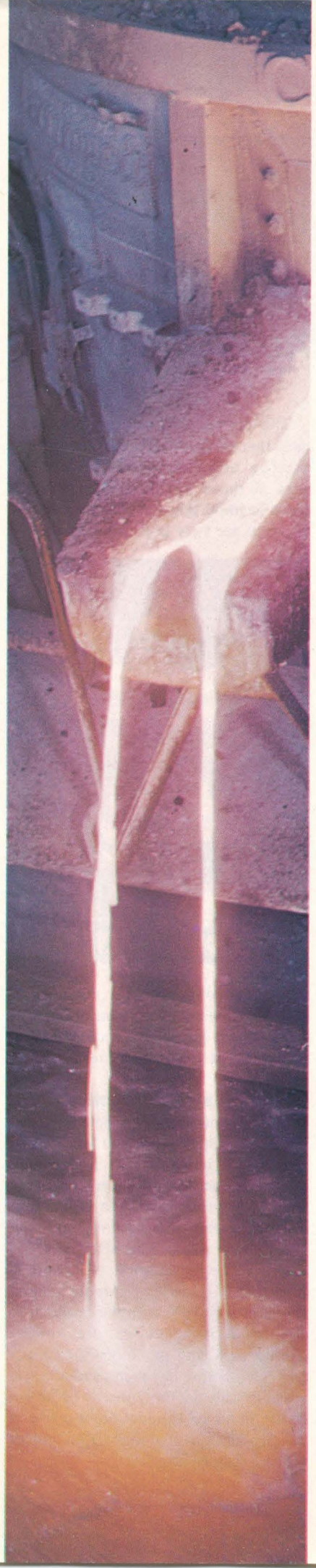
The world's proved and probable reserves of cobalt are 3,670,000 tons of contained metal. Potential reserves total 4,450,000 tons. The western countries share in these categories of reserves is 73 percent and 89 percent respectively. The greatest part of the proved and probable reserves lie in Cuba (just over 22 percent), Indonesia has about 15 percent, and Zaire and the Philippines have about 12 percent each. The rest is concentrated mainly in New Caledonia, Zambia, Canada, the Soviet Union, and Australia. About one-third of the reserves within the categories mentioned are contained in sulphide or sulphide/oxide ore deposits, two-thirds are contained in nickel laterite ores, as shown in Table 1.

Linked to Cu and Ni

The most important deposit types for the production of cobalt today are the strata-bound copper cobalt deposits of Zaire and Zambia. During recent years these deposits have accounted for between 55 and 65 percent of world production.

TABLE 1.
Share of various deposit types in world production (1977) and reserves (1978) of cobalt, in percent

Deposit type	Production	Reserves, proved and probable	Potential
Nickel-copper (Sudbury type)	21	14	14
Copper-cobalt (Katanga type)	55	21	7
Volcanogenic-sedimentary	7	1	1
Nickel laterite	17	64	78



In Zaire Gécamines produces cobalt at its Western group mines (around Kolwezi) and at its Central group (around Likasi). About 80 percent of the cobalt comes from the Western group. Approximately two-thirds of the cobalt is mined from open pits.

Gécamines is currently involved in an ambitious plan to increase copper and cobalt production. This project involves developing new open pits at the Mashamba-Dikuluwe deposit near Kolwezi, a new flotation concentrator, a flash smelter, and a new refinery. These facilities would include a plant for the production of an additional 6,000 annual tons of cobalt. Although some parts of the expansion project have been completed or are under construction, it seems likely that Gécamines will be hard pressed to maintain production much less increase it for some time yet.

South of Zaire on the Zambian Copperbelt, both state companies, Roan Consolidated Mines Ltd. and Nchanga Consolidated Copper Mines Ltd., are cobalt producers.

Cobalt containing copper ore is mined at NCCM's Rokana Division and at RCM's Chibuluma and Baluba mines.

Both groups are involved in crash programs to increase their cobalt production. The NCCM plant at Rokana has a capacity of about 2,200 tons of metal per year and in 1978 produced about 1,700 tons. Planned production this year is 2,000 tons. NCCM has large reserves of cobalt (estimated 150,000 tons metal equivalent) in the western boundary of the Nchanga open pit at Chingola. Plans are to build a concentrator at Chingola to treat this ore. The

TABLE 2.
World mine production of cobalt in metric tons

Country	1974	1975	1976	1977
Canada	1,565	1,355	1,375	1,540
Botswana	35	80	150	180
Morocco	1,625	1,960	860	910
Zambia	2,380	2,385	2,200	2,450
Zaire	17,630	14,000	11,000	13,610
Finland	1,270	1,400	1,280	1,270
Philippines	30	105	465	635
Australia	1,080	2,710	3,480	3,450
New Caledonia	1,920	2,055	1,800	1,815
Soviet Union	1,725	1,770	1,770	1,900
Cuba	1,635	1,635	1,635	1,600
World total	30,895	29,445	26,015	29,360

Note: Very small producers such as Poland, South Africa, Spain, etc. have been omitted from this tabulation.

TABLE 3.
World production of cobalt metal in tons

Country	1975	1976	1977
Canada	580	795	800
United States	30	165	250
Botswana	10	70	150
Zambia	1,860	1,700	1,700
Zaire	13,640	10,685	10,200
West Germany	340	350	350
France	770	850	850
Finland	820	890	980
Norway	775	575	600
Japan	50	515	1,150
Soviet Union	1,800	1,800	1,900
World total	20,675	18,395	18,930

concentrate would be sent to the expanded cobalt plant at Rokana, or to the new RCM cobalt plant at Chambishi.

The RCM Chambishi plant was under construction when the cobalt crisis erupted. Work was accelerated but Zambia too suffers from supply delays, cash shortages, and importantly, skilled expatriate staff shortages. The plant was scheduled for production early this year but it is now hoped to have it in operation by June 1979.

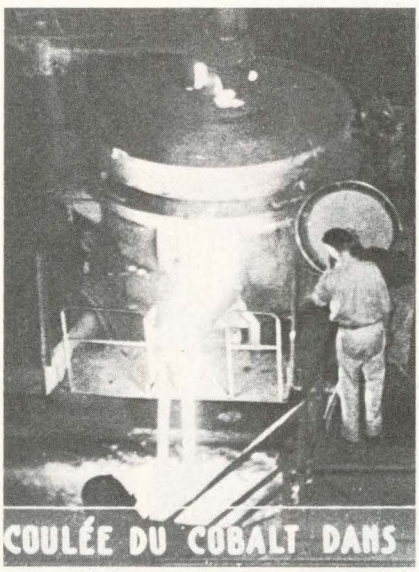
Zambia has visionary plans to build up its cobalt production eventually to about 10,000 annual tons. It has the ore reserves; what it is short on is money and skilled manpower. Recently Zambia signed contracts with a group of British cobalt consumers guaranteeing a three-year supply of cobalt at a rate of 685 tons per year. The contract was associated with an immediate loan advance against future deliveries.

After the African copper-cobalt ores, the next great source of cobalt production today are the nickel-copper sulphides of such areas as Sudbury, Ontario, Canada, and Norilsk in the Soviet Union. Similar, but smaller deposits of this type are found in Africa (e.g. Botswana), Australia, Finland, and elsewhere.

There are also less important volcanogenic-sedimentary type base metal deposits which contain recoverable cobalt. Examples of this type are the Cu-Co-Zn ores of the Keretti deposit in Finland, and most notably Bou Azzer in Morocco. This latter mine is the only one in the world where the primary ore mineral is cobalt. In all other cases the cobalt is mined as a byproduct of copper, nickel, etc.

Finally there are the lateritic nickel deposits which contain cobalt. The large and economically important cobalt containing lateritic nickel deposits are located in New Caledonia, Australia, Indonesia, Philippines, Cuba, and other such tropical areas.

The authors of *Cobalt* (see box) consider that the prospects for the discovery of unknown ore deposits of the same types as those which are at present of decisive importance in the production of cobalt are very promising. It should



be pointed out, however, that in all cases the low cobalt content of the ores means that it can only be recovered as a byproduct of some other metal.

Constraints on Production

World production of cobalt peaked at 30,895 tons in 1974. It then fell back to 26,015 tons in 1976. This was mainly caused by the drop in Zaire's output—from 17,630 tons in 1974 to 11,000 tons in 1976—which was partially compensated for by expansions elsewhere.

Since the production of cobalt is more than 95 percent a byproduct linked almost exclusively with the production of sulphide nickel-copper ores, nickel lateritic ores, and cobalt bearing copper ores, an increase in cobalt production is possible only in the context of an increase in nickel production or a rise in the production of cobalt from cobalt bearing copper ores.

For the extraction of cobalt from nickel laterites, however, the type of metallurgical processing of this ore is of fundamental importance. Cobalt can only be extracted by the use of hydrometallurgical processes. If nickel matte is produced the extraction of cobalt is possible only to a limited extent. If ferro-nickel is produced the separation of cobalt cannot be carried out.

At present about 55 percent of nickel laterite is processed to ferronickel. The production of ferronickel is also being given primary importance in a large number of newly planned nickel mining projects. Under these circumstances even a considerable increase in nickel ore production would not give rise to a corresponding increase in the production of byproduct cobalt.

Table 2 shows cobalt mine production by countries.

Considering the actual production of cobalt metal in 1977, the world total was 18,930 tons. Again only 11 countries were involved, although they are different to some extent from the primary ore producers. Zaire, however, was by far the biggest metal producer and its world share was 54 percent followed at a long distance by the Soviet Union (10 percent) and Zambia (9 percent). These three countries accounted for almost three-quarters of the total cobalt metal production of the world.

Five countries, Japan, Finland, France, Canada, and Norway, had production shares between 3 and 5 percent of world total each. The other countries, including the United States, were below 2 percent each (Table 3).

Recycled cobalt also enters into the supply picture. Of the large number of cobalt containing products, only a few represent a practicable source of recuperation of the metal or its compound—although the recent sharp price rises have stimulated such recycling. Used and waste material of cobalt containing alloys from the manufacture or mainte-

TABLE 4.
Share of countries in world mine cobalt production in 1967 and 1977, percent

Country	1967 Percent	Cumulative	Country	1977 Percent	Cumulative
Zaire	50.9	50.9	Zaire	46.4	46.4
Morocco	10.1	61.0	Australia	11.7	58.1
Finland	9.4	70.4	Zambia	8.3	66.4
Canada	8.6	79.0	Soviet Union	6.5	72.9
Zambia	7.6	86.6	New Caledonia	6.2	79.1
Soviet Union	7.1	93.7	Cuba	5.4	84.5
Cuba	5.5	99.2	Canada	5.3	89.8
Australia	0.8	100.0	Finland	4.3	94.1
			Morocco	3.1	97.2
			Philippines	2.2	99.4
			Botswana	0.6	100.0

nance of gas turbines is of great importance for recycling. In extreme cases the manufacture of gas turbine components from forged pieces by machine cutting processes can result in 90 percent waste, for example in the manufacture of turbine rings. Even milling out the root of the blade from cast blanks gives a sizeable yield of metal chips. Overall, in the construction of jet turbines about 50 percent waste can be expected.

Producer Concentration

The concentration of cobalt mine production is remarkable. In 1967 total world production came from only 8 countries. In 1977 this total had risen to 11. In 1967 the five biggest producing countries had just over 87 percent of the production, in 1977 about 79 percent. Only two countries, Zaire and Zambia, retained their positions among the five leaders in both years under comparison. Morocco, Finland, and Canada (1967) were replaced by Australia, the Soviet Union, and New Caledonia (1977) which, however, were already producing cobalt in 1967.

Apart from Zaire, which retained first place throughout, great changes occurred in the order of the other producing countries. Although the concentration of production has declined slightly, today almost half the cobalt in the world is produced in one country and about two-thirds in only three countries (Table 4).

In the western world the degree of concentration of production by companies is also extraordinarily high. Again more than half the production came from one company and two-thirds from only three (Table 5).

Substitutes Hard to Find

Cobalt is in great demand because of certain of its properties, viz. temperature resistance, resistance to wear at elevated temperatures, corrosion resistance, fatigue strength, bonding properties, magnetic properties, catalytic properties, and others.

The most important consumption sectors in metallurgy are the special steel industry, the non-ferrous metals industry, the hard metals industry, and the permanent magnets industry.

In the special steel sector, cobalt is above all alloyed with the high speed steels and the refractory steels.

TABLE 5.
Share of largest companies in western world mine production of cobalt in 1977, percent

Company	Percent	Cumulative
Gécamines (Zaire)	53	53
Western Mining (Australia)	9	62
Soc. Met. le Nickel (New Caledonia)	7	69
Nchanga Consolidated Copper Mines (Zambia)	6	75
Outokumpu Oy (Finland)	5	80
Inco Ltd. (Canada)	4	84
Compagnie de Tifnout Tiranimine (Morocco)	4	88
Metals Exploration Ltd. (Australia)	4	92
Roan Consolidated Mines Ltd. (Zambia)	3	95

For example, in turbines the use of cobalt alloys is concentrated in the hot area of the turbine with impeller and guide blades and their relative discs. The first stage contains the highest cobalt content while all further stages contain progressively lower cobalt contents. No substitution by other metals is to be found in this application.

High temperature alloy producers are conducting joint research programs with big jet engine makers aimed at developing nickel based alloys that could substitute for cobalt in gas turbines. However, nickel alloys cannot yet endure the extreme temperatures that jet engines must withstand. Research efforts are costly, time consuming, and show little indication of bearing fruit in the short term. Stationary or marine gas turbines could possibly accept some lower grade cobalt content, but the necessary process to validate a substitute or an alloy containing a reduced amount of cobalt in the aircraft industry could take several years. Thus jet engine makers will probably be bound to cobalt alloys for a long time.

Another huge consumer is the magnet industry. Here substitution is possible where competition is increasing from ferrite or iron based permanent magnets. Ferrites are not yet the answer in all cases. For example in the big color television industry ferrite magnets create unwanted magnetic fields that cause distortions.

In percentage terms, the super alloys and magnetic alloys each account for about 30 percent of cobalt consumption, chemical compounds, for 25 percent.

Slower Consumption Growth

World cobalt consumption rose from 22,000 tons in 1966 to 28,200 tons in 1976. The biggest consumer of cobalt was the United States with a 32 percent share of world consumption in 1976. It was followed by the Soviet Union with 16 percent, Japan with 13 percent, and West Germany in fourth place with 8 percent in 1976.

By 1990 it is estimated that world cobalt consumption will be over 44,000 tons. World trade in cobalt takes place predominantly in cobalt metal and cobalt oxide. Here Belgium acts as central distributor for cobalt material from Zaire.

Future consumption of cobalt is likely to rise less steeply than in the past, owing to the diminished economic growth generally expected in the world.

The cobalt reserves in the copper deposits of Zaire and Zambia and the nickel deposits of the world will not be exhausted in the foreseeable future.

Cobalt reserves in the polymetallic ocean nodules have not been taken into account in the survey because it is felt that they will not be mined in any significant quantity before the end of the 1980's. When they are mined, the cobalt could be won in such quantities that the cobalt supply would be considerably increased.

Although no shortage of cobalt is to be expected in terms of the available deposits, even in the long term, restrictions of supply may well occur for economic and political reasons—this is happening now and is likely to continue for a considerable time.

The root cause for this lies in the close connection between cobalt extraction and copper ore mining in Zaire and Zambia and the equally highly concentrated nickel production of the world. ■

Interest in sponge iron will center on its high quality, production will continue to be restricted by high costs

Future prospects for the direct reduction of iron

The growth in the production of sponge iron by the direct reduction process continues. In 1978 output reached nearly 14,000,000 metric tons (see top figure). The number of direct reduction plants under construction at present assures an increase to over 20,000,000 tons by 1980 and this figure could be as high as 30,000,000 tons. The majority of plants use natural gas as the reductant and two processes, HyL and Midrex, dominate the scene. At present these two processes are employed in about 60 percent of the plants around the world, but by 1980 this figure is likely to rise to 70 percent.

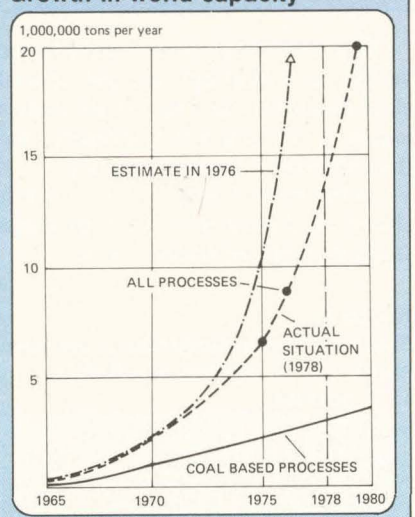
Jacques Astier, research director of the French iron and steel research center, made these predictions at a meeting in Brussels, Belgium, of the Société Française des Minerais Préréduits—an organization for those interested in prereduced ores.

Astier told the meeting that direct reduction plants fall primarily into two categories, the first being as part of an integrated iron and steel works based on direct reduction. In the second category are the plants producing sponge iron as a commercial product for sale around the world. A further discrimination of the integrated plants can be made using capacity as a reference. There are the mini-works with capacities between 150,000 and 600,000 tons per year; the medium-works have a capacity of about 1,000,000 tons, and there are the large integrated works with annual capacities of over 2,000,000 tons.

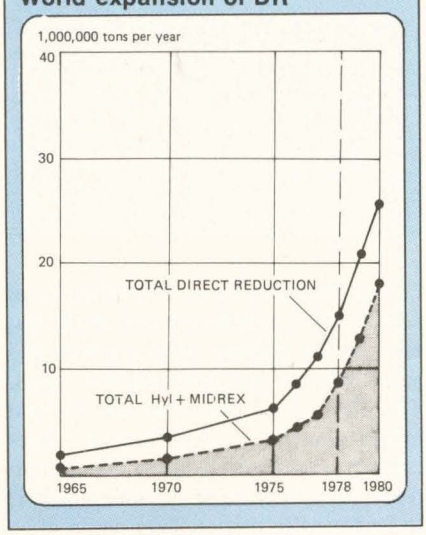
A study of the list of world integrated reduction plants shows that most are in the developing countries. The growth of the technique has been much slower in the developed industrial nations, one reason being that many of these developing countries have their own supply of cheap natural gas.

Mini-works have recently been constructed or are under construction in about a dozen countries. Medium-sized plants associated with the production of billets and rolled products are in operation in Iraq and Mexico, and in Canada and Mexico associated with flat products. Further medium-works are under construction in Nigeria, Iran, and Indonesia among other places. Large integrated mills based on direct reduction are soon to come into production for SIDOR in Venezuela, for Nisic at Ahvaz, and Bandar Abbas on the Persian Gulf in Iran. Another plant under construction in the Soviet Union will have an initial capacity of 2,000,000 tons per year increasing to 3,000,000 tons.

Growth in world capacity



World expansion of DR



Marketing sponge iron

Direct reduction plants that are marketing their product fall into two categories. The first are those that are part of an integrated works but have a surplus of production to sell. One such plant is the Georgetown Ferreduction plant in South Carolina, United States, that utilizes a Midrex series 400 shaft furnace.

Truly non-integrated plants generally have long-term delivery contracts with steel works. Two such plants are the Fior plant in Venezuela and the NFW plant under construction at Emden, West Germany.

Sponge iron is a primary metal, comparable to scrap, the differences being that sponge iron has a higher content of residual oxygen, carbon, and gangue and a lower percentage of Fe and residual metalloids. However, considering the processing cost, sponge iron becomes comparable to pig iron as shown below:

Production costs for 1 ton of Fe in prereduced ore

Cost	Dollars per ton
Iron ore	$ 50 (25 to 60)
Energy	$ 30 (10 to 40)
Operating costs	$ 10
Total	$ 90 (45 to 110)
Amortization	$ 20
Total	$110 (65 to 130)

When viewing the economics, one must note that the price of iron ore has not increased, due to the worldwide steel recession. The same applies for energy costs, because price increases have generally been offset by savings generated by improved processes. Finally, processing and investment costs are likely to grow. The capital cost of establishing a direct reduction plant can be taken as being in the range of $100 to $200 per ton of annual capacity. Thus, a plant producing 500,000 tons of sponge iron per year would cost between $50,000,000 and $100,000,000 to put into operation.

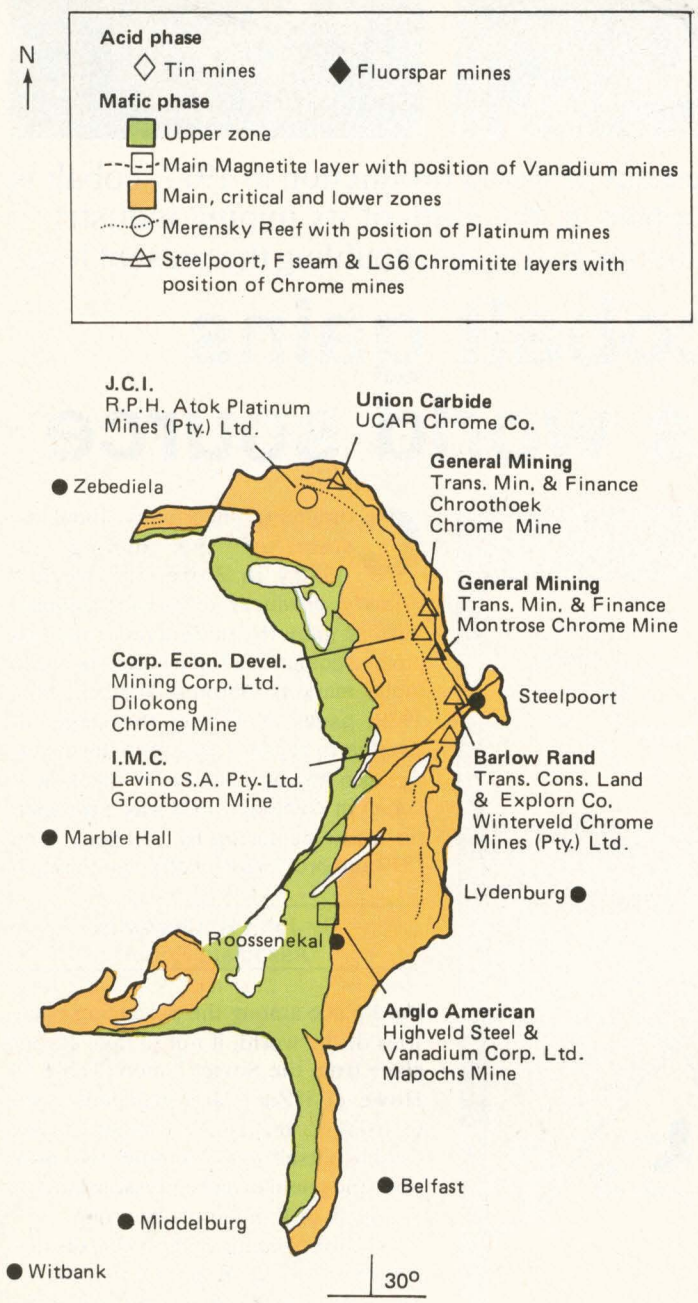
The share of sponge iron that will be consumed over the next few years by the industrialized countries will only be complementary to scrap. The main point in its favor will be its quality; quantity will remain restricted due to high capital and production costs. ■

patched to Feralloys. Total output from the Ntuane mine is exported to Europe and Japan and that from Marico Mineral Company is used in the local manufacture of refractories as well as being sold to ferroalloy producers.

Production from Mining Corporation's Dilokong Mine started in September 1977 with a maximum output of 2,200 tons (2,000 metric) for the year. Production for the first six months of 1978 was 33,000 tons (30,000 metric) with full production of 80,300 tons (73,000 metric) projected for the remainder of the year. The mine has an annual design capacity of 203,500 tons (185,000 metric).

It is estimated that 1,450,000 tons of chromite ore and 280,000 tons of ferrochromium were exported in 1977. Sales of ferrochromium and, indirectly, chromite ore are closely dependent on the production of stainless steel and a slump in the demand from the main export areas (United States, Europe, and Japan) has forced South African ferrochromium producers to reduce output to between 60 and 70 percent of capacity.

Chromium-bearing stainless steel is vital in many industrial processes and in machines operating at high temperatures and pressures such as jet engines. In view of the strategic importance of chromium there may be a degree of reluctance by western governments to become over reliant on South African ferrochromium and chromite ore. This factor may impose an artificial ceiling on the export potential of these products. ■

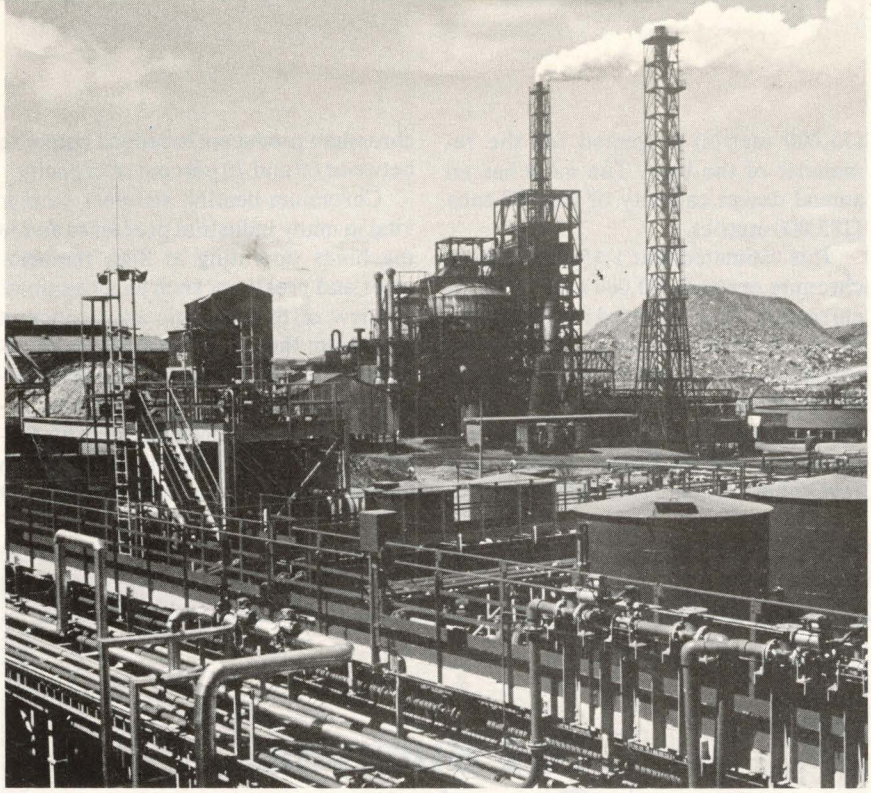


The Bushveld Complex

As Southern African gold production declines (in tonnage, not revenue), the industry must turn to alternative mineral deposits and the huge resources of the Bushveld complex must assume an ever increasing importance. In 1977 sales of platinum group metals, chromite ore, nickel, and tin from the Bushveld made up 9.3 percent of the country's total mineral sales.

The chromite deposits associated with four of the five separate bodies making up the mafic phase of the Bushveld Complex all occur in the Transvaal and extend over a distance of 250 miles (400 kilometers) from west to east. Chromite is confined to the Critical Zone present towards the base of the mafic phase and occurs as layers of varying thickness. Table 1 shows a list of the producing chromite mines; their locations are shown on the accompanying map. The 1977 production of 3,630,000 tons (3,300,000 metric) represented 36 percent of world production.

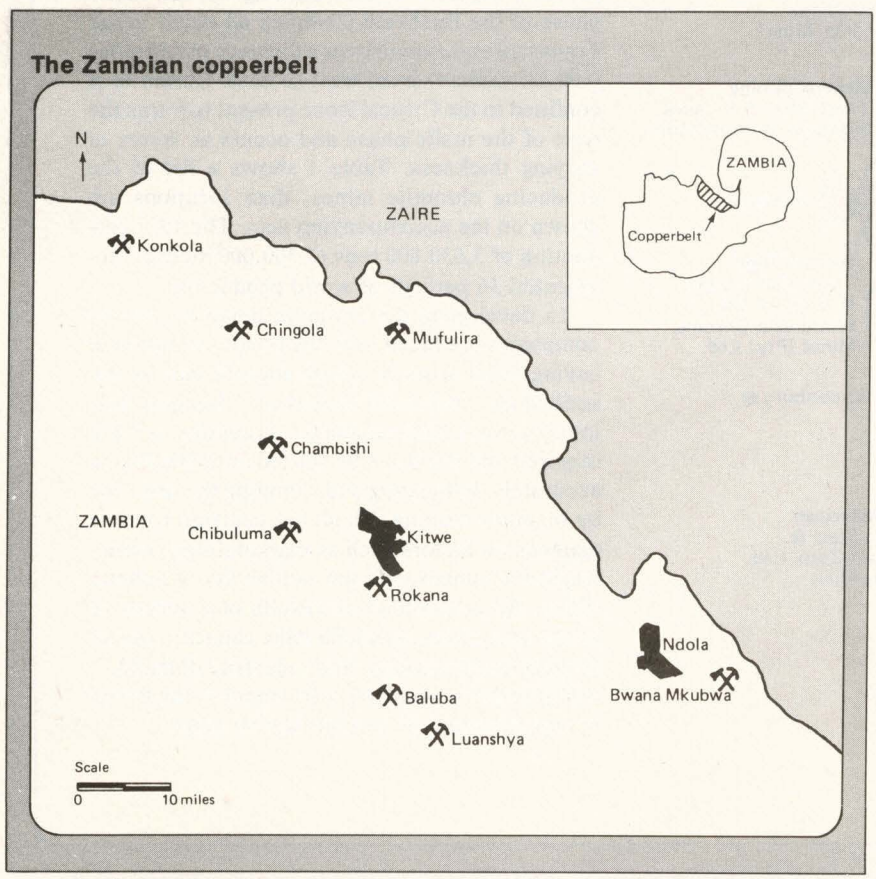
In developing the chromite deposits, mining companies are faced with the unique situation of having large areas in which the ore can be described as "indicated resources." Lengthy regional surveys and expensive programs are not required and work can be started immediately on accurately delineating and sampling the ore zone by diamond drilling. Thus, the decision to mine depends on factors such as marketability, metallurgical advances, and the availability of mineral rights. By presenting the results of a survey of active operations, available mine capacity, metallurgical developments, and industrial demand, it is hoped that a balanced assessment of the future mining potential of chrome can be aided.





To the country that accounted for 9.0 percent of world production in 1977, cobalt is a great prospect for a much needed upturn in the state of its mining industry. The coming years should see major production increases, doubling the present level

Zambian cobalt gains importance as world source



Continuing uncertainty about the state of the mining industry in Zaire, the world's primary producer of cobalt, coupled with a growth in demand that has pushed cobalt to a top position in world metal markets (WORLD MINING, May 1979, pages 40-43) set the stage for Zambia to regain some of its lost prestige as a mining country. Production of cobalt in Zambia in 1978 was 2,063 metric tons, increasing by 359 tons from 1977 and presumably sufficient to retain

By **John R. Chadwick**
Associate Editor

third place among the producing countries of the world, if not to take second place from the Soviet Union (Table 1). However, if Zambian cobalt plans come to fruition, the country should firmly establish itself in the number two position, and could even be pressing hard on the heels of Zairean production.

At the beginning of this year, the new cobalt plant of Roan Consolidated Mines Ltd. came on stream and at the Chingola Division of Nchanga Consolidated Copper Mines Ltd., major re-

NEW COBALT CIRCUIT at Chambishi (far left). The new roaster stack is on the right. Near left: cobalt is stripped from the cathodes by hand at Rokana.

serves of cobalt are to be developed. These two state mining companies are the sole producers of cobalt in the country. Prior to the new discovery at Chingola, proved and probable reserves amounted to 300,000 tons of metal (8.2 percent of the world total) and potential reserves totalled a further 50,000 (1.1 percent of world total).

The Zambian Copperbelt has long been one of the world's most important copper producing areas and since 1933 cobalt has been produced at the mining complex in Kitwe that is now the Rokana Division of NCCM. The area comprises two distinct belts and, to date, it is the southwesternmost of these two belts, comprising Konkola, Chingola, Chambishi, Chibuluma, Rokana, Baluba, and Luanshya that has exhibited cobalt mineralization.

Table 1.
World production of cobalt metal in tons in 1976, 1977 and 1978

Country	1976	1977	1978
Zaire	10,685	10,200	11,500
Soviet Union	1,800	1,900	N.A.
Zambia	1,700	1,704	2,063
Japan	515	1,150	2,050
Finland	890	980	985
France	850	850	850[1]
Canada	795	800	625
Norway	575	600	520
West Germany	350	350	350[1]
United States	165	250	N.A.
Botswana	70	150	261

1. Estimated.

The metallic ore minerals of the Copperbelt are found in the Katanga system of rocks which are mostly of sedimentary origin. This system comprises mainly schists, quartzites, and marbles that have been altered and metamorphosed. The rocks have been folded by earth movements and concurrent granite intrusions occurred.

The cobalt occurs almost entirely as carrollite ($CuCO_2S_4$) and linnaeite (CO_3S_4). There is an oxidized fraction, which appears as asbolite and sphaerocobaltite ($CaCO_3$), that forms a small and varying feed at Rokana. In chalcocite or bornite ores, the cobalt mineral is chiefly carrollite, while in chalcopyrite/pyrite, as in Rokana's south ore body, only linnaeite occurs, finely disseminated in the pyrite.

Developing Cobalt Potential

Cobalt concentrate is currently produced at NCCM's Rokana Division and at the RCM mines of Chibuluma and Baluba. In 1977, NCCM ranked fourth in the world with 6.0 percent of production, while RCM ranked ninth with 3.0 percent. Currently, the most important is Rokana where 42,693 tons of cobalt concentrate were produced in 1978 containing 3.90 percent Co (Table 2). It is indicative of the high value of this metal that the respective cobalt contents of the ores hoisted at Rokana's three shafts in 1977 were: Mindola, 0.094 percent; South ore body, 0.09 percent; and Central, 0.127 percent.

The Baluba section of Luanshya Division is a relatively recent addition to the cobalt producers, having been commissioned in early 1973. The reserves of this section for the financial year ending June 30, 1978 were 70,000,000 tons of ore assaying 2.46 percent copper and 0.15 percent cobalt. It is estimated that about 44,000 tons of cobalt metal can be removed from this ore body. At the same time, Chambishi Division's Chibuluma reserves were quoted as 7,000,000 tons assaying 4.69 percent copper and 0.21 percent cobalt.

Table 2.
Concentrate production in metric tons

		Copper concentrate			Cobalt concentrate		
Year	Mine	Tonnage	Copper, percent	Cobalt, percent	Tonnage	Copper, percent	Cobalt, percent
1976	Chibuluma	55,344	32.11	0.42	20,233	4.31	2.96
1977		52,146	31.70	0.51	28,844	2.74	2.58
1978		36,331	32.59	0.35	29,978	7.15	2.28
1976	Baluba	65,082	29.82	0.40	28,931	9.17	4.12
1977		92,187	29.08	0.44	35,156	7.60	3.85
1978		97,090	29.04	0.44	37,677	6.88	3.90
1976	Rokana	282,708	27.68	*	41,897	6.92	3.51
1977		251,424	28.26	*	41,447	7.94	3.81
1978		241,067	27.80	*	42,693	6.23	3.90

Source: Zambia Mining Year Book. *Cobalt not quoted as there is no recovery from this source.

Until recently, the production of cobalt metal was restricted to Rokana where the cobalt plant has been in operation for many years and has been used to produce cobalt for RCM on a toll basis as well as from its own concentrate. The plant has a rated capacity of 2,200 tons annually.

The flowsheet of this plant (Figure 1) is as follows: concentrate is fed via agitated tanks to obtain a homogeneous slurry, into the fluidized bed roaster. The roaster bed ensures that all the concentrate comes into and remains in contact with the air that is blown through it. Heat generated by the exothermic reaction is spread evenly. The roaster converts cobalt sulphide into water soluble cobalt sulphate, which is amenable to leaching.

A two stage leaching operation is used to ensure maximum recovery. Pregnant liquor from leaching passes through a copper tankhouse, where copper cathodes are produced, to secondary purification. There cobalt hydroxide is precipitated and cobalt is redissolved in acid.

Purified electrolyte which feeds the cobalt tankhouse will typically contain about 40 grams of cobalt per liter. A four-day pulling cycle is used, after which time the cathodes are washed and the cobalt metal is stripped from them manually. A typical cathode will contain about 99.66 percent cobalt.

The final stage of the operation is one of vacuum degassing, in which hydrogen is removed. The cobalt is heated in an electric furnace under vacuum. Some hydrogen remains on the metal surface. This is removed by burnishing. In all, the hydrogen content is reduced to less than 1 part per million. The finished cobalt is then packaged in 200-kilogram drums for shipment.

2,400-ton Annual Increase

The latest plant to be placed in operation is the Chambishi leach plant extension with a cobalt circuit capable of producing 200 tons of metal per month. This plant is now treating the cobalt concentrates from Chambishi and Baluba. Its production this year has steadily increased, from 65 tons in January to 180 tons in April.

Concurrent with the Chambishi cobalt plant coming on stream, the Luanshya, where Baluba ore is treated, and Chambishi concentrators have been geared to maximize the output of cobalt concentrate. Prior to modification of the flotation circuit in January, Luanshya's copper flotation mill recovered about 87 percent of the copper in the ore and 19 percent of the cobalt. The cobalt concentrate assays were 6 percent copper and 55 percent cobalt. The cobalt content of the copper concentrate was lost to slag. After the modification, the copper concentrate yielded a 51 percent copper recovery and 3 percent cobalt



COBALT CHIPS are packed in 200-kilogram drums for airfreighting. From January through May of this year, almost 1,000 tons were delivered this way.

recovery and the cobalt concentrate yielded a 43 percent copper recovery and 70 percent cobalt recovery. At Chambishi, cobalt recovery in the copper concentrate had been 24 percent and 64 percent in the cobalt concentrate; these two figures have now been improved to 10 percent and 85 percent, respectively.

A great deal of attention in the Zambian mining industry has been focused on increasing cobalt reserves and production over recent years. The reasons are obvious, not least of which is the opportunity to put the cobalt 'boom' to work in improving the country's poor economic situation. It is indicative of the emphasis placed on this metal, that in the first half of 1978, NCCM appointed a cobalt manager at Rokana Division. This position, the first of its kind, carries considerable power. The cobalt manager oversees mining, metallurgical, engineering, and cobalt recovery and is directly responsible to the general manager of the Division.

Major Production Boosts

As previously mentioned, large reserves have recently been delineated in the western boundary of the Nchanga open pit at Chingola and in the underground workings there. These reserves are estimated to contain 150,000 tons of cobalt metal. Provisions for the recovery of this cobalt will probably include a new flotation concentrator to be established at the Chingola Division at a cost of about $20,000,000. The concentrate will be sent to Rokana for the production of cobalt metal.

At Rokana, plans to build a new cobalt plant have been approved (see page 131 of this issue). A $19,000,000 program is to be instituted, the money to be spent on three specific projects: modifications to the existing concentrator to produce copper/cobalt concentrates, a new cobalt plant, and a new acid plant. At present, no cobalt is recovered from the Rokana copper concentrates; thus, the modifications to the concentrator will provide an important additional source of metal for the existing cobalt plant to recover.

The bulk of the money set aside for the cobalt expansion at Rokana is to be spent on the new cobalt plant which will have a capacity of 2,500 tons of finished metal per year. It is hoped that this will be in production by the end of 1985. This expansion will require a substantial increase in the plant area, which will in turn require demolition of a large area of mine housing adjacent to the mine. Concentrate sources for this new plant will include the new production from Chingola.

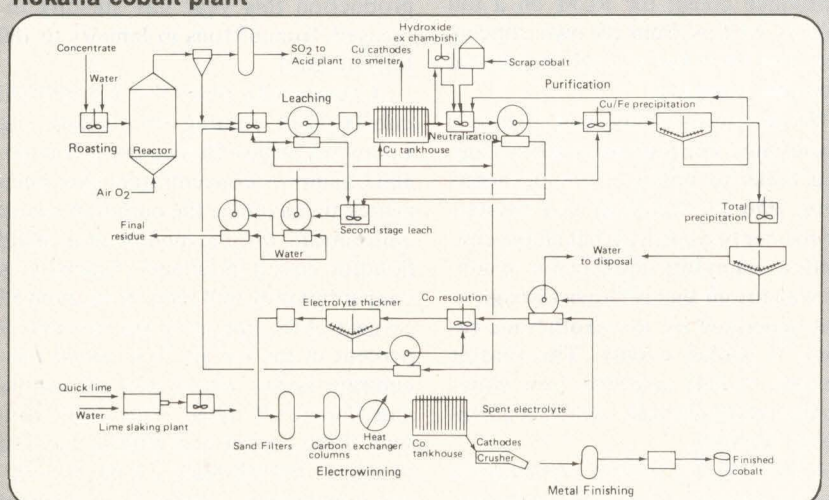
A further source of cobalt, particularly at Rokana, is the smelter slags. Part of the work being carried out on the Copperbelt is an investigation of the technical and economic feasibility of recovering this cobalt. One figure for the Rokana slag dump estimated that its cobalt content will amount to 8,000 tons of contained metal by 1983. The price of the metal probably justifies the development of this source, but the initial investment required would be large and take a number of years to amortize.

Zambia expects to produce a total of 3,300 tons of cobalt this year and a rapid rise in production is expected over the next few years. One may wonder how, with the country's continuing transport difficulties, the mining companies are going to deliver the product. The answer is the use of air freight. This method of transport is gaining increasing importance for the more valuable metallic products of various countries, particularly those in Africa. For example, Rössing Uranium Ltd. has been flying its product to western Europe since production began and the same has been true for Zambian cobalt in recent years.

This form of transport, under the right conditions, is not as expensive as one might imagine, while the advantages of getting a valuable product to the marketplace quickly are obvious. From January through May of this year, almost 1,000 tons of metal were delivered to Zambian customers in this way.

Various reports have come from Zambia recently on the anticipated cobalt production into the 1980's, the highest being an increase to as much as 10,000 tons annually. However, it seems certain that the plans that have been announced recently will increase production to over 7,000 annual tons by the middle of the decade to firmly establish Zambia as the world's second largest producer and who knows what the production from Zaire will be by then. ■

Rokana cobalt plant



6-983

BUREAU OF MINES
MINERAL EXPLORATION PROGRAM

RECORD OF DIAMOND DRILL HOLE VALUES

Hole No.: 1
Location: 23 ft. N. 13° W. Auger Hole #9, Line 57
Elevation of collar: 2396
Depth: 230 ft.
Dip: -55°
Bearing: N. 60°W.
Date Begun: Dec. 9, 1953
Date Finished: Jan. 23, 1954

Project No.: 772.1
Name Takilma - Waldo
Location: Copper District, Oregon
Queen of ~~Brazil~~ Mine
Bronze

Footage			Sample No.		Analyses							
					Core				Sludge			Adjusted Average
From:	To:	Feet:	Core	Sldg.	Cu	Au	Ag	Ph	cu	au	ag	
10	15	5	.03-1	.05-3	0.02	<.01	0.30	7.8	0.09	<.01	0.30	
15	18.6	3.6	.03-2		<.02	<.01	0.10	7.6				
18.6	24.2	5.6	.03-3		0.05	<.01	0.40	7.7				
24.2	28.6	4.4	.03-4		0.12	<.01	0.25	7.0				
28.6	37	8.4	.03-5		0.05	<.01	0.20	7.0				
37	40	3.0	.03-6		0.18	<.01	0.25	7.8				
40	45	5	.03-7	.05-9	0.80	<.01	0.24	7.5	0.69	<.01	0.20	
45	50	5		.05-10					0.96	<.01	0.25	
50	55		.03-8	.05-11	0.21	<.01	0.30	7.4	1.20	<.01	0.30	
check P.MACK 55	59.4	4.4	.03-9		0.64	<.01	0.20	7.4				
59.4	64.1	4.7	.03-10		0.61	<.01	0.25	7.5				
64.1	71	6.9	.03-11		0.73	<.01	0.10	7.2				
71	76	5	.03-12		0.45	<.01	0.10	7.4				
76	80	4	.03-13		0.55	<.01	0.10	7.5				
AH 80	85	5	.03-14	.05-17	0.70	<.01	0.10	7.4	0.71	<.01	0.20	OK; D3-14, 0 Au, 0 Ag, 0.72 Cu - HANKS
85	90	5	.03-15	.05-18	0.48	<.01	0.10	7.6	0.59	<.01	0.20	
90	95	5	.03-16	.05-19	0.59	<.01	0.10	7.8	0.69	<.01	0.20	
95	100.5	5.5	.03-17		0.37	<.01	0.15	7.3				
100.5	106	5.5	.03-18		2.50	<.01	0.30	7.0				
check P.MACK 106	111	5	.03-19		0.51	<.01	0.10	7.7				
111	116	5	.03-20		0.89	<.01	0.10	7.8				

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6-983

BUREAU OF MINES
MINERAL EXPLORATION PROGRAM

RECORD OF DIAMOND DRILL HOLE VALUES

Hole No.: 1
Location:
Elevation of collar:
Depth:
Dip:
Bearing:
Date Begun:
Date Finished:

Project No.: _____
Name _____
Location:

Footage			Sample No.		Analyses: Core				Analyses: Sludge			Adjusted Average
From:	To:	Feet:	Core	Sldg.	Cu	Au	Ag	Ph	Cu	Au	Ag	
116	118.5	2.5	.03-21		0.92	<.01	0.30	8.0				
118.5	129.5	11.0	.03-22		0.03	<.01	0.20	9.3				
129.5	140	10.5	.03-23		0.02	<.01	0.20	9.5				
140	150	10	.03-24	.06-2	0.02	<.01	0.20	9.4	0.52	<.01	0.10	
150	159	9	.03-25		<.02	<.01	0.20	9.3				
159	171	12	.04-1		<.02	<.01	0.20	9.2				
171	179.2	8.2	.04-2		0.02	<.01	0.20	9.4				
179.2	190	10.8	.04-3		<.02	<.01	0.20	9.1				
190	198	8	.04-4		<.02	<.01	0.30	9.0				
198	207	9	.04-5		<.02	<.01	0.20	9.3				
207	215	8	.04-6		<.02	<.01	0.26	9.7				
215	223	8	.04-7		<.02	<.01	0.20	9.3				
223	230	7	.04-8		<.02	<.01	0.20	9.5				
0	5	5		.05-1					0.09	<.01	0.20	
5	10	5		.05-2					0.06	<.01	0.20	
10	15	5		.05-3					0.09	<.01	0.30	
15	20	5		.05-4					0.07	<.01	0.20	
20	25	5		.05-5					0.15	<.01	0.30	
25	30	5		.05-6					0.40	<.01	0.15	
30	35	5		.05-7					0.75	<.01	0.25	
35	40	5		.05-8					0.75	<.01	0.30	

1498

6-983

BUREAU OF MINES
MINERAL EXPLORATION PROGRAM

RECORD OF DIAMOND DRILL HOLE VALUES

Hole No.: 1
Location:
Elevation of collar:
Depth:
Dip:
Bearing:
Date Begun:
Date Finished:

Project No.: _____
Name _____
Location:

Footage			Sample No.		Analyses							
					Core				Sludge			Adjusted Average
From:	To:	Feet:	Core	Sldg.	Cu	Au	Ag	Ph	Cu	Au	Ag	
40	45	5		.05-9					0.69	<.01	0.20	
45	50	5		.05-10					0.96	<.01	0.25	
50	55	5		.05-11					1.20	<.01	0.30	
55	60	5		.05-12					0.86	<.01	<.01	
60	65	5		.05-13					0.87	<.01	0.20	
65	70	5		.05-14					2.65	<.01	0.10	
70	75	5		.05-15					1.60	<.01	<.01	
75	80	5		.05-16					0.97	<.01	0.10	
80	85	5		.05-17					0.71	<.01	0.20	
85	90	5		.05-18					0.59	<.01	0.20	
90	95	5		.05-19					0.69	<.01	0.20	
95	100	5		.05-20					0.66	<.01	0.25	
100	105	5		.05-21					1.66	<.01	0.20	
105	110	5		.05-22					0.54	<.01	0.10	
110	115	5		.05-23					0.62	<.01	0.20	
115	120	5		.05-24					0.55	<.01	0.10	
120	130	10		.05-25					0.16	<.01	<.01	
130	140	10		.06-1					0.32	<.01	0.30	
140	150	10		.06-2					0.52	<.01	0.10	
150	160	10		.06-3					0.42	<.01	<.01	
160	170	10		.06-4					0.31	<.01	<.01	

1498

6-983

BUREAU OF MINES
MINERAL EXPLORATION PROGRAM

RECORD OF DIAMOND DRILL HOLE VALUES

Hole No.: 1
Location:
Elevation of collar:
Depth:
Dip:
Bearing:
Date Begun:
Date Finished:

Project No.: _____
Name _____
Location:

Footage			Sample No.		Analyses		
					Core	Sludge	Adjusted Average
From:	To:	Feet:	Core	Sldg.	Cu	Cu	
170	180	10		.06-5		0.07	
180	190	10		.06-6		0.10	
190	200	10		.06-7		0.05	
200	205	5		.06-8		0.05	
					For adjusted average		
10	40	30			0.07	0.37	0.25
40	45	5			0.80	0.64	0.70
45	50	5			none	0.96	0.96
50	55	5			0.21	1.20	0.57
55	59.4	4.4			0.64	0.86	.70
59.4	80	20.6			0.54	1.33	1.17
80	85	5			0.70	0.71	0.71
85	90	5			0.48	0.59	0.55
90	95	5			0.59	0.69	0.62
95	100.5	5.5			0.37	0.66	0.51
100.5	118.5	18			1.23	0.82	0.95
118.5	140				0.02	0.17	0.15
140	150	10			0.02	0.52	0.45
150	205	55			<.02	0.12	0.10

78.5 ft of .861

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BUREAU OF MINES
MINERAL EXPLORATION PROGRAM

RECORD OF DIAMOND DRILL HOLE VALUES

Hole No.: 2
Location: 27 ft. N. 7° W. Auger Hole # 16 Line 59
Elevation of collar: 2337
Depth: 202.2
Dip: -45°
Bearing: N. 60° E.
Date Begun: Jan. 30, 1954
Date Finished: March 5, 1954

Project No.: 772.1
Name Takilma - Waldo
Location: Copper Dist., Oregon
Queen of Brazil Mine

	Footage			Sample No.		Analyses						Adjusted Average			
						Core				Sludge					
	From:	To:	Feet:	Core	Sldg.	Cu	Au	Ag.	Ph	Cu		Cu	Au	Ag	
	0	5	5		.06-10					0.05		0.05			
	5	10	5		.06-11					0.07		0.07			
	10	15	5		.06-12					0.04		0.04			
	15	20	5		.06-13					0.03		0.03			
	20	25	5		.06-14					0.07		0.07			
	25	30	5	.04-9		<.02	<.01	0.20	9.2			<.02			
	30	35	5	.04-10		<.02	<.01	0.20	9.6			<.02			
	35	37	2	.04-11		<.02	<.01	0.20	9.6			<.02			
	37	46	9	.04-12		<.02	<.01	0.10	8.8			<.02			
	46	51	5	.04-13		0.04	<.01	0.20	9.3			0.04			
AW	51	55	4	.04-14		0.29	<.01	0.20	8.3			0.29			
	55	56	1												
	56	61	5	.04-15	.06-15	0.11	<.01	0.20	9.1	0.32					
	61	64.5	3.5	.04-16	.06-16	<.02	<.01	0.20	9.6	0.25					
	64.5	68.3	3.8	.04-17	.06-17	0.04	<.01	0.20	9.1	0.08					
check	68.3	73	4.7	.04-18	.06-18	<.02	<.01	0.20	9.0	0.08					
	73	77	4	.04-19	.06-19	0.45	<.01	0.15	8.7	0.17					
AW	77	81.5	4.5	.04-20	.06-20	0.62	<.01	0.20	8.9	0.41					
	81.5	86.3	4.8	.04-21	.06-21	0.02	<.01	0.16	9.5	0.11					
	86.3	87.8	1.5	.04-22		0.02	<.01	0.28	9.5						
	87.8	92.2	4.4	.04-23	.06-22	0.02	<.01	0.10	9.5	0.23					

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check AW 04-14, 0 Au, 0 Ag, 0.10 Cu
04-20, .02 Au, 0 Ag, 0.56 Cu

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BUREAU OF MINES
MINERAL EXPLORATION PROGRAM

RECORD OF DIAMOND DRILL HOLE VALUES

Hole No.: 2
Location:
Elevation of collar:
Depth:
Dip:
Bearing:
Date Begun:
Date Finished:

Project No.: _____
Name _____
Location:

Footage			Sample No.		Analyses						
					Core				Sludge		Adjusted Average
From:	To:	Feet:	Core	Sldg.	Cu	Au	Ag	Ph	Cu		
92.2	97.0	4.8	.04-24	.06-23	0.02	<.01	0.44	9.4	0.07		
97.0	101.5	4.5	.04-25	.06-24	0.02	<.01	0.38	9.3	0.03		
101.5	106	4.5	.07-1	.06-25	0.04	<.01	0.16	9.3	0.07		
106	111.7	5.7	.07-2	.08-1	0.03	<.01	0.16	9.6	0.07		
111.7	117	5.3	.07-3	.08-2	<.02	<.01	0.12	9.4	0.07		
117	121	4	.07-4	.08-3	<.02	<.01	0.16	9.4	0.05		
121	126.5	5.5	.07-5	.08-4	0.02	<.01	0.12	9.2	0.05		
126.5	132	5.5	.07-6	.08-5	<.02	<.01	0.08	9.3	0.05		
132	136.4	4.4	.07-7	.08-6	<.02	<.01	0.12	9.2	0.05		
AH 136.4	141.3	4.4	.07-8	.08-7	0.04	<.01	0.14	9.2	0.02		ex. DZ-8, 0 Au, 0 Ag, 2 trace Cu - Mahns
141.3	145	3.7	.07-9	.08-8	<.02	<.01	0.16	9.2	0.03		
145	150	5	.07-10	.08-9	0.04			9.3	0.04		
150	155	5	.07-11	.08-10	0.04			9.5	0.03		
155	160.2	5.2	.07-12	.08-11	0.05			9.2	<.02		
160.2	165.4	5.2	.07-13	.08-12	0.02			9.4	0.04		
165.4	171	5.6	.07-14	.08-13	0.02			9.4	0.02		
171	176	5	.07-15	.08-14	0.04			9.1	0.03		
176	181.5	5.5	.07-16	.08-15	<.02			9.2	0.10		
181.5	185.5	4	.07-17	.08-16	<.02			9.3	0.05		
185.5	191.5	6	.07-18	.08-17	<.02			9.3	0.04		
191.5	195.5	4	.07-19	.08-18	<.02			9.3	0.04		

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BUREAU OF MINES
MINERAL EXPLORATION PROGRAM

RECORD OF DIAMOND DRILL HOLE VALUES

Hole No.: 2
Location:
Elevation of collar:
Depth:
Dip:
Bearing:
Date Begun:
Date Finished:

Project No.: _____
Name _____
Location:

Footage			Sample No.		Analyses						
					Core				Sludge		Adjusted Average
From:	To:	Feet:	Core	Sldg.	Cu	Au	Ag	Ph	Cu		
199	200.9	1.9	.07-20		<.02			9.4			
195.5	200.9	5.4	.07-20	.08-19	<.02			9.4	0.04		
200.9	202.2	1.3	.07-21	.08-20	<.02			9.3	0.04		

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BUREAU OF MINES
MINERAL EXPLORATION PROGRAM

RECORD OF DIAMOND DRILL HOLE VALUES

Hole No.: 3
Location: 29' N. 7° W. Auger hole # 9, Line 54
Elevation of collar: 2300
Depth: 117 ft.
Dip: vertical
Bearing:
Date Begun: March 18, 1954
Date Finished: March 30, 1954

Project No.: 772.1
Name Takilma etc.
Location:

Footage			Sample No.		Analyses					
					Core			Sludge		Adjusted Average
From:	To:	Feet:	Core	Sldg.	Cu		Ph	Cu		
0	5	5		.08-22				0.16		
5	10	5								
10	15.5	5.5		.08-23				0.19		
13	15.5	2.5	.07-22		0.16		7.8			
15.5	20	4.5	.07-23	.08-24	0.10		7.8	0.13		
20	25	5		.08-25				0.11		
25	30	5		.010-1				0.13		
30	35	5		.010-2				0.07		
35	40	5		.010-3				0.06		
40	45	5	.07-24	.010-4	0.05		7.8	0.04		
45	50	5	.07-25	.010-5	<.02		8.5	0.07		
50	55	5		.010-6				0.03		
55	60	5	.09-1	.010-7	0.03		8.2	0.04		CK: D10-7, trace Au, 0 Ag, 0.02 Cu Hanks
60	65	5	.09-2	.010-8	0.06		8.4	0.08		
65	70	5	.09-3	.010-9	0.04		8.4	0.10		
70	75	5	.09-4	.010-10	<.02		8.3	0.03		
75	80	5	.09-5	.010-11	<.02		8.3	0.04		
80	85	5	.09-6	.010-12	<.02		8.5	0.07		
85	90	5	.09-7	.010-13	<.02		8.6	0.03		
90	95	5	.09-8	.010-14	<.02		7.9	0.04		
95	100	5	.09-9	.010-15	<.02		8.8	0.03		

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BUREAU OF MINES
MINERAL EXPLORATION PROGRAM

RECORD OF DIAMOND DRILL HOLE VALUES

Hole No.: 3
Location:
Elevation of collar:
Depth:
Dip:
Bearing:
Date Begun:
Date Finished:

Project No.: _____
Name _____
Location:

Footage			Sample No.			Analyses				
						Core			Sludge	Adjusted Average
From:	To:	Feet:	Core	Sldg.	Cu		Ph	Cu		
100	105.5	5.5	.09-10	.010-16	<.02		8.3	<.02		
105.5	110	4.5	.09-11	.010-17	<.02		8.2	<.02		
110	115	5	.09-12	.010-18	<.02		8.4	<.02		
115	117	2	.09-13	.010-19	<.02		9.0	<.02		

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BUREAU OF MINES
MINERAL EXPLORATION PROGRAM

RECORD OF DIAMOND DRILL HOLE VALUES

Hole No.: 4
Location: 6' South Augar hole # 15, Line 54
Elevation of collar: 2302
Depth: 78.5
Dip: vertical
Bearing:
Date Begun: April 2, 1954
Date Finished: April 9, 1954

Project No.: 772.1
Name Takilma - Waldo etc.
Location:

Footage			Sample No.		Analyses			
					Core		Sludge	Adjusted Average
From:	To:	Feet:	Core	Sldg.	Cu	PH	Cu	
0	5	5		.010-20			0.11	
5	10	5		.010-21			0.06	
10	15	5	.09-14	.010-22	<.02	7.70	0.05	
15	20	5	.09-15	.010-23	<.02	7.80	0.03	
check 20	25	5	.09-16	.010-24	<.02	7.80	0.02	
25	30	5	.09-17	.010-25	<.02	7.70	0.04	
30	35	5	.09-18	.012-1	<.02	7.60	0.02	
35	40	5	.09-19	.012-2	<.02	8.10	0.04	
40	45	5	.09-20	.012-3	<.02	8.10	0.02	
45	50	5	.09-21	.012-4	<.02	8.40	0.03	
50	55	5	.09-22	.012-5	<.02	8.20	0.03	
55	60	5	.09-23	.012-6	<.02	7.70	0.02	
check 60	65	5	.09-24	.012-7	<.02	8.20	0.04	
65	70	5	.09-25	.012-8	<.02	7.90	0.02	
70	75	5	.011-1	.012-9	<.02	8.40	0.02	
75	78.5	3.5	.011-2	.012-10	<.02	8.40	0.03	

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BUREAU OF MINES
MINERAL EXPLORATION PROGRAM

RECORD OF DIAMOND DRILL HOLE VALUES

Hole No.: 5
Location: 50ft. S6° E. Auger hole # 21, Line 54
Elevation of collar: 2263
Depth: 49.1
Dip: vertical
Bearing:
Date Begun: April 13, 1954
Date Finished: April 15, 1954

Project No.: 772.1
Name Takilma etc.
Location:

Footage			Sample No.		Analyses			
					Core		Sludge	Adjusted Average
From:	To:	Feet:	Core	Sldg.	Cu	pH	Cu	
0	5	5		.012-11			0.05	
5	10	5		.012-12			0.03	
10	15	5	.011-3	.012-13	0.03	8.70	<.02	
15	19.5	4.5	.011-4	.012-14	0.02	8.60	0.03	
19.5	25.7	6.2	.011-5	.012-15	<.02	8.70	<.02	
25.7	29.1	3.4	.011-6	.012-16	<.02	9.20	<.02	
29.1	33.7	4.6	.011-7	.012-17	0.02	9.30	0.04	
33.7	39.0	5.3	.011-8	.012-18	<.02	9.50	<002	
39.0	44.0	5.0	.011-9	.012-19	<.02	9.60	0.02	
44.0	49.1	5.1	.011-10	.012-20	<.02	9.60	<.02	

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BUREAU OF MINES
MINERAL EXPLORATION PROGRAM

RECORD OF DIAMOND DRILL HOLE VALUES

Hole No.: 6
Location: 17 ft. S. 18° E. Auger hole # 3, Line 53
Elevation of collar: 2281
Depth: 77 ft.
Dip: vertical
Bearing:
Date Begun: April 20, 1954
Date Finished: April 27, 1954

Project No.: _____
Name _____
Location:

Footage			Sample No.		Analyses			
					Core		Sludge	Adjusted Average
From:	To:	Feet:	Core	Sldg.	Cu	pH	Cu	
0	5	5						
5	10	5		012-21			0.03	
10	15	5	.011-11	.012-22	<.02	8.3	<.02	
15	20	5	.011-12	12-23	<.02	8.4	<.02	
20	25	5	.011-13	12-24	<.02	8.7	<.02	
25	30	5	.011-14	12-25	<.02	8.8	<.02	
30	35	5	.011-15	.013-1	<.02	8.7	0.03	
35	40.1	5.1	.011-16	.013-2	<.02	8.6	0.05	
40.1	45.2	5.1	.011-17	.013-3	<.02	8.4	0.05	
45.2	48.8	3.6	.011-18	.013-6	<.02	8.2	<.02	
48.8	55	6.2	.011-19	.013-7	<.02	8.5	<.02	
55	59.5	4.5	.011-20	.013-8	<.02	8.8	<.02	
59.5	65.5	6.0	.011-21	.013-9	<.02	8.5	<.02	
65.5	71.1	5.6	.011-22	.013-10	<.02	8.7	<.02	
71.1	75.8	4.7	.011-23	.013-11	<.02	8.7	<.02	
75.8	77	1.2	.011-24	.013-12	<.02	8.7	<.02	

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Form SM-18 (Revised April, 1945)
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BUREAU OF MINES
MINERALS EXPLORATION PROGRAM
DIAMOND DRILL HOLE LOG

HOLE NO: 1
Location: LINE 57, Auger Hole #9
Elevation of collar: 2396 ft.
Depth: 230 ft.
Dip: -55°
Bearing: N. 60° W.
Date begun: Dec. 9, 1953
Date finished: Jan. 23, 1954

PROJECT No. 12221.2 Name _____
Location: TAKILMA, OREGON

Theoretical wt. in grams of sludge per ft. of hole:
Bx-- 1176
Ax-- 918
Ex-- 353
NX -- 3506 (with plug bit)

Footage			Recoveries						Core Diam.	Sample Numbers		Description and Remarks
			Core		Sldg.	In percent						
From:	To:	Feet:	Feet:	Gms.	Gms.	Core	Sldg.	Wtr.	Ins.	Core	Sldg.	
0'	5.0	5.0	0		3166		18.0		NX		D5-1	NON CORING BIT.
5.0	10.0	5.0	0		7186		41.0		NX		D5-2	do
10.0	12.3	2.3	0.59	624		25.6			BX-1 5/8	D3-1		oxidized, altered meta volcanic. Fine grained, weathered, kaolinized, silicified. Light to moderate iron stain, light greenish color.
12.3	15.0	2.7	1.06	1121		39.2			BX	D3-1		SAME with light manganese stain (copper ?), less kaolin. Brecciated, fractured.
15.0	18.6	3.6	0.67	708		18.7			BX	D3-2		SAME. no visible copper stain.
18.6	23.4	4.8	1.69	1786		34.9			BX	D3-3		SAME.
23.4	24.2	0.8	0.55	581		66.2			BX	D3-3		SAME - moderate to heavy iron stain. Lower 0.25' of run is earthy limonite with some cellular box works.
24.2	25.8	1.6	0.93	985		58.8			BX	D3-4		SAME - Earthy limonite with some cellular box works near center and upper part of run.
25.8	28.6	2.8	2.2	2338		80.7			BX	D3-4		SAME but ~~light to moderate~~ less iron stain than previous Run.

Form SM-18 (Revised April, 1945)
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BUREAU OF MINES
MINERALS EXPLORATION PROGRAM
DIAMOND DRILL HOLE LOG

HOLE NO: 1
Location:
Elevation of collar:
Depth:
Dip:
Bearing:
Date begun:
Date finished:

PROJECT No. 12321.2 Name _____
Location:

Theoretical wt. in grams of sludge per ft. of hole:
Bx--
Ax--
Ex--

Footage			Recoveries						Core Diam.	Sample Numbers		Description and Remarks
			Core		Sldg.	In percent						
From:	To:	Feet:	Feet:	Gms.	Gms.	Core	Sldg.	Wtr.	Ins.	Core	Sldg.	
28.6	32.6	4.0	0.94	985		23.5			BX 1 5/8	D3-5		SAME
32.6	37.0	4.4	0.44	460		9.95			BX	D3-5		SAME
37.0	40.0	3.0	0.88	921		29.3			BX	D3-6		SAME but light gray color. no brecciation. Lost water at 37-40'
~~40.0~~	~~45.0~~	~~0~~	~~0.32~~	~~340~~	~~4527~~	~~6.4~~	~~79.7~~		BX	~~D3-7~~	~~D5-9~~	
40.0	45.0	5.0	0.32	340	4527	6.4	79.7		BX	D3-7	D5-9	SAME - Encountered old stope (caved or back filled) at 41 feet. No core from 41 to 45'. No water return 42.5 to 45.
45.0	50.0	5.0	0	0	2743	0	48.0		BX		D5-10	Hole caving
50.0	55.0	5.0	1.12	1176	683	22.4	12.0		BX	D3-8	D5-11	Driller reports hole still in old stope. Core may be from boulder in muck pile. Rock type same but again light ~~green~~ gray color. 0.2 sq. of upper core is pyrite, cellular box works and earthy limonite.
	58.0	3.0										
55.0	~~59.4~~	~~4.4~~	0.23	240		7.7			BX	D3-9		Same rock type with heavier iron staining. Light gray color. Lizzie box works & earthy limonite at upper end of core. No water from 57' to 58'
58.0	59.4	1.4	0.61	645		42.9			BX	D3-9		Same rock type. Less iron staining. no water recovery

Form SM-18 (Revised April, 1945)
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BUREAU OF MINES
MINERALS EXPLORATION PROGRAM
DIAMOND DRILL HOLE LOG

HOLE NO: /
Location:
Elevation of collar:
Depth:
Dip:
Bearing:
Date begun:
Date finished:

PROJECT No. 12931.2 Name _____
Location:

Theoretical wt. in grams of sludge per ft. of hole:
Bx--
Ax--
Ex--

Footage			Recoveries						Core Diam.	Sample Numbers		Description and Remarks
			Core		Sldg.	In percent						
From:	To:	Feet:	Feet:	Gms.	Gms.	Core	Sldg.	Wtr.	Ins.	Core	Sldg.	
59.4	60.4	1.0	0.54	567		54.0			Bx 1 5/8	D3-10		SAME type. Lower end somewhat
~~60.4~~	~~62.1~~	~~1.7~~	~~1.10~~	~~1162~~		~~66.3~~						brecciated, no water
60.4	62.1	1.7	1.10	1162		66.3			Bx	D3-10		SAME type - small amount ~~copper~~ MALACHITE
												no water return. STILL in
												caved slope.
62.1	64.1	2.0	0.59	624		29.5			Bx	D3-10		SAME type - small amount malachite.
												~~no~~ partial water return 62'-65'
64.1	71.0	6.9	0.31	325		0.4			Bx	D3-11		Same type - light green ~~ish~~ grey color.
65.0	70.0	5.0			1307		23.0		Bx	~~D3-14~~	D5-14	poor water return.
71.0	76.0	5.0	1.77	1857		35.4			Bx	D3-12		SAME type. ~~Some~~ Driller reports solid ground at 74' - Some brecciation
76.0	80.0	4.0	1.77	1864		44.2			Bx	D3-13		SAME type. Good water return
80.0	85.0	5.0	1.27	1332	8351	25.4	147.0		Bx	D3-14	D5-17	SAME type - small ami. malachite
												near lower part of core. light gray color.
85.0	87.0	2.0	0.83	872		41.5			Bx	D3-15		SAME type - small amount pyrite
87.0	90.0	3.0	1.92	2020	6467	64.0	(85'-90') 113.8		Bx	D3-15	D5-18	near middle of core.
90.0	93.0	3.0	3.1	3537		104.0			Bx	D3-16		same type - some quartz +
93.0	95.0	2.0	1.56	1772	2637	78.0	90'-95' 46.4		Bx	D3-16	D5-19	pyrite in structures in lower
~~95.0~~												part of core
95.0	97.3	2.3	2.1	2268		91.4			Bx	D3-17		SAME, but brecciated, light green ~~ish~~ grey color
97.3	99.0	1.7	0.93	992		55.7			Bx	D3-17		SAME, some malachite along fractures.

Driller reports ground caving badly. maybe another caved slope.
Cemented hole 80-99 ft.

Form SM-18 (Revised April, 1945)
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BUREAU OF MINES
MINERALS EXPLORATION PROGRAM
DIAMOND DRILL HOLE LOG

HOLE NO: /
Location:
Elevation of collar:
Depth:
Dip:
Bearing:
Date begun:
Date finished:

PROJECT No. 1221.2 Name _____
Location:

Theoretical wt. in grams of sludge per ft. of hole:
Bx--
Ax--
Ex--

Footage			Recoveries						Core Diam.	Sample Numbers		Description and Remarks
			Core		Sldg.	In percent						
From:	To:	Feet:	Feet:	Gms.	Gms.	Core	Sldg.	Wtr.	Ins.	Core	Sldg.	
99.0	100.5	1.5	1.0	1098		68.6			BX - 1 5/8	D3-17		SAME TYPE - some malachite in fractures
100.5	106.0	5.5	2.61	2778		47.4			BX	D3-18		Same TYPE - Increased amount malachite
106.0	111.0	5.0	3.39	3607		67.8			BX	D3-19		same type - no malachite noted. Some pyrite in fractures.
111.0	116.0	5.0	0.76	744		15.2			BX	D3-20		SAME Type - light gray color, SOME slickensides. Little malachite in fractures with secondary quartz.
116.0	118.5	2.5	1.85	1970		74.0			BX	D3-21		same type -
118.5	123.0	4.5	0.71	808		15.8			BX	D3-22		same type, but less altered + fresher in appearance. - Greenish gray color, some disseminated pyrite.
123.0	125.0	2.0	0.53	602		26.5			BX	D3-22		same type - light iron stain.
125.0	129.5	4.5	1.25	1424		27.8			BX	D3-22		same type - calcite and some cellular box works in fractures.
129.5	~~144~~ 132.0	2.5	1.61	1899		64.4			BX	D3-23		Same Type - No iron stain, Quartz along fractures Some disseminated pyrite.
132.0	133.0	1.0	0.84	559		84%			AX 1 7/16	D3-23		Same Type - light iron stain, Quartz along fractures, reduced to AX at 132 feet.
133.0	138.0	5.0	0.41	240		8.2%			AX	D3-23		Same Type - Quartz along fractures. Some disseminated Pyrite.
138.0	140.0	2.0	None	None					AX			No Core

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BUREAU OF MINES
MINERALS EXPLORATION PROGRAM
DIAMOND DRILL HOLE LOG

HOLE NO:
Location:
Elevation of collar:
Depth:
Dip:
Bearing:
Date begun:
Date finished:

PROJECT No. _____ Name _____
Location:

Theoretical wt. in grams of sludge per ft. of hole:
Bx--
Ax--
Ex--

Footage			Recoveries						Core Diam.	Sample Numbers		Description and Remarks
			Core		Sldg.	In percent						
From:	To:	Feet:	Feet:	Gms.	Gms.	Core	Sldg.	Wtr.	Ins.	Core	Sldg.	
140	145	5	2.1	1240		42.0			AX 1 7/6	D3-24	D6-2	Same Type-Quartz along fractures, Some disseminated Pyrite.
145	150	5	2.94	1736	20104	58.8	2/6		AX	D3-24	D6-2	Same Type- Quartz along fractures, Some disseminated Pyrite
150	153	3	1.73	1020		57.6			AX	D3-25		Same Type - Quartz along fractures, Some disseminated Pyrite
153	153.9	.9	.8	474		86.9			AX	D3-25		Same Type - Quartz along fractures, no visible Pyrite
153.9	156.5	2.6	1.24	730		48			AX	D3-25		Same Type - Quartz and Calcite along fractures some disseminated pyrite.
156.5	159	2.5	1.26	744		50.4			AX	D3-25		Same Type - Quartz along fractures, Some disseminated Pyrite.
159	164	5	.97	574		19.4			AX	D4-1		Same Type - Some disseminated Pyrite, light iron stain.
~~164~~	~~167~~	~~3~~	~~.68~~	~~403~~		~~22.6~~			~~AX~~	D4-1		
164	167	3	.68	403		22.6			AX	D4-1		Same Type - light iron stain, No visible Pyrite
167	171	4	.70	411		17.5			AX	D4-1		Same Type - Quartz and Calcite along fractures No visible iron stain or Pyrite.
171	175.5	4.5	.58	340		12.8			AX	D4-2		Same Type - Quartz and Calcite along fractures
175.5	179.2	3.7	1.98	1169		52.8			AX	D4-2		Same Type - Quartz along fractures, Some disseminated Pyrite.
179.2	182.7	3.5								D4-3		Same Type - Some Quartz along fractures, Some disseminated Pyrite.
182.7	185	2.3	1.44	850		64.0			AX	D4-3		Same Type - No visible disseminated pyrite

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BUREAU OF MINES
MINERALS EXPLORATION PROGRAM
DIAMOND DRILL HOLE LOG

HOLE NO:
Location:
Elevation of collar:
Depth:
Dip:
Bearing:
Date begun:
Date finished:

PROJECT No._____ Name _____
Location:

Theoretical wt. in grams of sludge per ft. of hole:
Bx--
Ax--
Ex--

Footage			Recoveries						Core Diam.	Sample Numbers		Description and Remarks
			Core		Sldg.	In percent						
From:	To:	Feet:	Feet:	Gms.	Gms.	Core	Sldg.	Wtr.	Ins.	Core	Sldg.	
185	186.7	1.7	.82	233		46.8			EX 7/8"	D4-3		Same Type - No visible Pyrite. Reduced to AX
186.7	190	3.3	2.27	1339		83.0			AX 1 3/16"	D4-3		Same Type - Some brecciation, Some disseminated pyrite. Changed to AX at 186.7'
190	195	5	3.23	1906		64.6			AX	D4-4		Same Type - Quartz along fractures, No visible Pyrite.
195	198	3	.79	467		26.3			AX	D4-4		Same Type - Quartz and Calcite along fractures, Some disseminated pyrite.
198	202	4	1.09	644		27.2			AX	D4-5		Same Type - Quartz and Calcite along fractures, Some disseminated pyrite.
202	207	5	.73	432		14.6			AX	D4-5		Same Type - Quartz and Calcite along fractures, Some disseminated pyrite.
207	212	5	1.31	772		26.2			AX	D4-6		Same Type - Quartz and Calcite along fractures, Some disseminated pyrite.
212	215	3	1.31	772		43.6			AX	D4-6		Same Type - Quartz and Calcite along fractures No visible pyrite.
215	220	5	.79	467		15.8			AX	D4-7		Same Type - Quartz and calcite along fractures.
220	221	1	.17	99		17.0			AX	D4-7		Same Type - Quartz and Calcite along fractures.
221	223	2	1.16	687		58.0			AX	D4-7		Same Type - Quartz and Calcite along fractures, Some disseminated pyrite.

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BUREAU OF MINES
MINERALS EXPLORATION PROGRAM
DIAMOND DRILL HOLE LOG

HOLE NO:
Location:
Elevation of collar:
Depth:
Dip:
Bearing:
Date begun:
Date finished:

PROJECT No._____ Name _____
Location:

Theoretical wt. in grams of sludge per ft. of hole:
Bx--
Ax--
Ex--

Footage			Recoveries						Core Diam.	Sample Numbers		Description and Remarks
			Core		Sldg.	In percent						
From:	To:	Feet:	Feet:	Gms.	Gms.	Core	Sldg.	Wtr.	Ins.	Core	Sldg.	
223	226.5	3.5	2.0	1233		59.7			AX 1 3/16	D4-8		Same Type - Quartz and Calcite along fractures Some disseminated Pyrite.
226.5	228	1.5	.6	354		40.0			AX 1 3/16	D4-8		Same Type - Quartz and Calcite along fractures, Some disseminated Pyrite.
228	229.5	1.5	.7	418		47.3			AX	D4-8		Same Type - Quartz and Calcite along fractures, Some disseminated pyrite.
229.5	230	.5	none	none		none			AX			No Core

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BUREAU OF MINES
MINERALS EXPLORATION PROGRAM
DIAMOND DRILL HOLE LOG

HOLE NO: 2
Location: 28 feet N. 6°W. of Auger hole #16, Line 59
Elevation of collar: 2330'
Depth: 202.2
Dip: 45°
Bearing: N. 60°E
Date begun: 1/30/54
Date finished: 3/5/54

PROJECT No. 12212 Name _____
Location: to Kilmaz, Oregon

Theoretical wt. in grams of sludge per ft. of hole:
Bx-- 1109
Ax-- 865
Ex--
NX-- 3531 (with plug bit)

Footage			Recoveries						Core Diam.	Sample Numbers		Description and Remarks
			Core		Sldg.	In percent						
From:	To:	Feet:	Feet:	Gms.	Gms.	Core	Sldg.	Wtr.	Ins.	Core	Sldg.	
0	5.0	5.0	[illegible]	[illegible]	2970	0.0	16.8	50.0	NX	[illegible]	D6-10	Non Coring Bit
5.0	10.0	5.0	0.0	0.0	4030	0.0	22.8	50.0	NX		D6-11	Non Coring Bit. Hit old mine workings at 7 feet.
10.0	15.0	5.0	0.0	0.0	3910	0.0	70.4	50.0	BX 1 5/8		D6-12	No Core
15.0	20.0	5.0	0.0	0.0	4202	0.0	75.7	50.0	BX		D6-13	No Core
20.0	25.0	5.0	0.0	0.0	1439	0.0	25.8	10.0	BX		D6-14	No Core
25.0	30.0	5.0	1.4	1486		29.7		10.0	BX	D4-9		Oxidized, Altered, light iron stain, fine grained, metavolcanic, light greenish gray color - brecciated, fractured, mottled. Considerable amount of calcite and small amounts of quartz occur along fractures
30.0	32.5	2.5	.30	302		12.0		0.0	BX	D4-10		Oxidized, altered, fine grained, metavolcanic, gray color, fractured. Calcite occurs along fractures. Some massive Calcite with sulfides (Pyrite?).
32.5	35	2.5	2.0	2002		80.4		0.0	BX	D4-10		Greenish gray to light gray and white, massive calcite, fractured with calcite of another generation occuring along fractures. disseminated sulfides throughout.
35	37	2	1.6	1652		83.0		0.0	BX	D4-11		Same Type - Calcite along fractures, disseminated sulfides (Pyrite?)

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BUREAU OF MINES
MINERALS EXPLORATION PROGRAM
DIAMOND DRILL HOLE LOG

HOLE NO:
Location:
Elevation of collar:
Depth:
Dip:
Bearing:
Date begun:
Date finished:

PROJECT No._____ Name _____
Location:

Theoretical wt. in grams of sludge per ft. of hole:
Bx--
Ax--
Ex--

Footage			Recoveries						Core Diam.	Sample Numbers		Description and Remarks
			Core		Sldg.	In percent						
From:	To:	Feet:	Feet:	Gms.	Gms.	Core	Sldg.	Wtr.	Ins.	Core	Sldg.	
37.0	46.0	9	.3	359		4.0		0.0	BX 1⅝	D4-12		Small pieces of serpentine and dark gray metavolcanic. Probably backfill or caved stope.
46.0	51.0	5	.18	182		3.6%		0.0	BX	D4-13		Small pieces of dark gray metavolcanic. Probably backfill or caved stope. Hit old mine timber at 49'. Blasted timber. Some wood.
51.0	52.4	1.4	.35	352		25.1%		0.0	BX	D4-14		Probably backfill or caved stope. Oxidized, altered, fine grained, metavolcanic, light gray color fractured, Copper stain along fractures. Some pyrite and earthy limonite. Few small pieces of massive Calcite with copper stain along fractures. Some wood.
52.4	54.4	2	1.20	1200		60%		0.0	BX	D4-14		Upper 6" of Core is serpentine with Calcite along serpentenized fractures, light iron and Copper (malachite) stain. Lower core is oxidized, altered, light iron stained, fine grained metavolcanic, light grey color, copper stain along fractures.
54.4	55	.6	.49	499		83%		0.0	BX	D4-14		Metavolcanic, same type as above, brecciated. Quartz, calcite, Pyrite, occur along brecciation. Some Copper stain along fractures

Form SM-18 (Revised April, 1945)
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BUREAU OF MINES
MINERALS EXPLORATION PROGRAM
DIAMOND DRILL HOLE LOG

HOLE NO:
Location:
Elevation of collar:
Depth:
Dip:
Bearing:
Date begun:
Date finished:

PROJECT No._____ Name _____
Location:

Theoretical wt. in grams of sludge per ft. of hole:
Bx--
Ax--
Ex--

Footage			Recoveries						Core Diam.	Sample Numbers		Description and Remarks
			Core		Sldg.	In percent						
From:	To:	Feet:	Feet:	Gms.	Gms.	Core	Sldg.	Wtr.	Ins.	Core	Sldg.	
55.0	56.0	1	None	None					BX			No Core
56.0	61.0	5	1.0	550	4817	20.0	116.2	100.0	AX 1 7/16	D4-15	D6-15	Oxidized, altered, light iron stain, fine grained, metavolcanic, fractured. Quartz & Calcite and copper stain along fractures.
61.0	64.5	3.5	2.49	1367	2197	71.1	72.2	100.0	AX	D4-16	D6-16	Oxidized, altered, peridotite, fractured. Calcite along fractures.
64.5	65.3	.8	.3	164	Total below	37.5	162.3	100.0	AX	D4-17	D6-17	Altered, light grey, fine grained, metavolcanic, fractured. Calcite along fractures.
65.3	67.0	1.7	.59	324	Total below	34.8	162.3	100.0	AX	D4-17	D6-17	Same as above
67.0	68.3	1.3	1.0	597	5342	72.6	162.3	100.0	AX	D4-17	D6-17	Same Type - Calcite and quartz along fractures
68.3	69.0	.5	.6	204	Total below	74.4	135.7	100.0	AX	D4-18	D6-18	Same Type - Calcite, quartz and pyrite? along fractures.
69.0	70.2	1.2	.64	365	Total below	54.0	135.7	100.0	AX	D4-18	D6-18	Same Type - Calcite and quartz along fractures No visible pyrite
70.2	73	2.8	.38	208	5526	13.5	135.7	100.0	AX	D4-18	D6-18	Same Type - Calcite and quartz along fractures.
73	75.5	2.5	.23	128	Total below	9.3	143.5	100.0	AX	D4-19	D6-19	Same Type - Calcite and quartz along fractures.
75.5	77.0	1.5	.47	374	4973	31.6	143.5	100.0	AX	D4-19	D6-19	Same Type - Calcite along fractures. Sulfides up to 1/8" in width along fractures.
77.0	78.0	1.0	.47	376	Total below	42.8	116.5	100.0	AX	D4-20	D6-20	Same Type - Calcite along fractures. Sulfides up to 1/4" in width along fractures.

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BUREAU OF MINES
MINERALS EXPLORATION PROGRAM
DIAMOND DRILL HOLE LOG

HOLE NO:
Location:
Elevation of collar:
Depth:
Dip:
Bearing:
Date begun:
Date finished:

PROJECT No._____ Name _____
Location:

Theoretical wt. in grams of sludge per ft. of hole:
Bx--
Ax--
Ex--

Footage			Recoveries						Core Diam.	Sample Numbers		Description and Remarks
			Core		Sldg.	In percent						
From:	To:	Feet:	Feet:	Gms.	Gms.	Core	Sldg.	Wtr.	Ins.	Core	Sldg.	
78.0	81.5	3.5	.81	446	4501	23.2	115.5	100.0	AX	D4-20	D6-20	Same Type - Calcite and small amount of sulfides along fractures.
81.5	82.1	0.6	.43	269	Total below	73.1	48.4	100.0	AX	D4-21	D6-21	Same Type - Calcite along fractures. Small amount of sulfides (Pyrite?) disseminated through core.
82.1	82.7	0.6	.43	263	"	71.5	48.4	100.0	AX	D4-21	D6-21	Same Type - Calcite along fractures. No visible sulfides.
82.7	83.0	0.3	.33	206	"	112.0	48.4	100.0	AX	D4-21	D6-21	Same Type - Calcite and Quartz along fractures. Some disseminated sulfides (Pyrite?).
83.0	83.7	0.7	.23	143	"	33.2	48.4	100.0	AX	D4-21	D6-21	Same Type - Calcite and Quartz along fractures. No visible disseminated Sulfides
83.7	86.3	2.6	1.26	776	2027	48.7	48.4	100.0	AX	D4-21	D6-21	Same Type - Calcite and Quartz along fractures. Small amount of disseminated sulfides (Pyrite?).
86.3	87.8	1.5	.24	149	None	16.2			AX	D4-22		Same Type - Calcite and Quartz along fractures. Some disseminated sulfides (Pyrite?) throughout. No Sludge.
87.8	91.0	3.2	.52	324	below	16.5	99.1	100.0	AX	D4-23	D6-22	Same Type - Calcite and Quartz along fractures. Small amounts of disseminated sulfides (Pyrite?) throughout.
												Cemented from 81 to 88 feet. Hole caved from 79.5' to 88' after drilling Cement.
91.0	92.2	1.2	.30	189	4026	25.6	99.1	100.0	AX	D4-23	D6-22	Same Type - Calcite and Quartz along fractures. Small amount of disseminated Sulfides (Pyrite) through out.
92.2	93.5	1.3	.79	484	below	60.7	62.5	100.0	AX	D4-24	D6-23	Same Type - Calcite along fractures. Small amount of iron stain along one fracture. Disseminated Sulfides through out.

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BUREAU OF MINES
MINERALS EXPLORATION PROGRAM
DIAMOND DRILL HOLE LOG

HOLE NO:
Location:
Elevation of collar:
Depth:
Dip:
Bearing:
Date begun:
Date finished:

PROJECT No. _____ Name _____
Location:

Theoretical wt. in grams of sludge per ft. of hole:
Bx--
Ax--
Ex--

Footage			Recoveries						Core Diam.	Sample Numbers		Description and Remarks
			Core		Sldg.	In percent						
From:	To:	Feet:	Feet:	Gms.	Gms.	Core	Sldg.	Wtr.	Ins.	Core	Sldg.	
93.5	94.5	1.0	.93	571	below	46.6	62.5	100.0	AX 1 3/16	D4-24	D6-23	Same Type - Calcite along fractures. No iron stain. Disseminated sulfides throughout.
94.5	97.0	2.5	.22	135	2767	8.8	62.5	100.0	AX	D4-24	D6-23	Altered, dark gray to light gray, fine grained, meta volcanic, fractured. Calcite along fractures. Upper end of core is dark gray and contains an increased amount of Sulfides (Pyrite?).
97.0	98.2	1.2	.60	368	below	50.0	63.9	100.0	AX	D4-25	D6-24	Altered, light greenish gray color, fine grained, meta volcanic. Quartz and Calcite along fractures. Small amount of disseminated sulfides (Pyrite). Noted change in color and grain of rock at 97 feet.
98.2	99.0	0.8	.58	361	below	73.6	63.9	100.0	AX	D4-25	D6-24	Same type as 97.0 to 98.2. Quartz and calcite along fractures, Small amount of disseminated Sulfides (Pyrite). light iron stain along one fracture.
99.0	101.5	2.5	.90	554	2653	36.2	63.9	100.0	AX	D4-25	D6-24	Same type. No visible Sulfides or iron stain.
101.5	102.9	1.4	1.25	769	below	89.7	29.6	100.0	AX	D7-1	D6-25	Same as above. Quartz and Calcite along fractures.
102.9	106.0	3.1	2.01	1231	1229	64.8	29.6	100.0	AX	D7-1	D6-25	Same Type - Quartz and calcite along fractures. Some disseminated sulfides (Pyrite?) throughout.
106.0	107.0	1.0	.88	539	below	88.0	31.3	100.0	AX	D7-2	D8-1	Same Type - Quartz and Calcite along fractures.

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BUREAU OF MINES
MINERALS EXPLORATION PROGRAM
DIAMOND DRILL HOLE LOG

HOLE NO:
Location:
Elevation of collar:
Depth:
Dip:
Bearing:
Date begun:
Date finished:

PROJECT No. _____ Name _____
Location:

Theoretical wt. in grams of sludge per ft. of hole:
Bx--
Ax--
Ex--

Footage			Recoveries						Core Diam.	Sample Numbers		Description and Remarks
			Core		Sldg.	In percent						
From:	To:	Feet:	Feet:	Gms.	Gms.	Core	Sldg.	Wtr.	Ins.	Core	Sldg.	
107.0	109.0	2.0	1.92	1178	below	96.2	31.3	100.0	AX / $\frac{3}{16}$	D7-2	D8-1	Same Type - Quartz and Calcite along fractures. Small amounts of disseminated Sulfides throughout
109.0	111.7	2.7	2.45	1504	1651	91.0	31.3	100.0	AX	D7-2	D8-1	Same Type - Quartz and calcite along fractures. No visible Sulfides.
111.7	114.0	2.3	2.07	1267	below	90.0	30.0	100.0	AX	D7-3	D8-2	Same Type - Quartz and calcite along fractures. Some disseminated Sulfides.
114.0	117.0	3.0	2.65	1623	1466	88.3	30.0	100.0	AX	D7-3	D8-2	Same as above.
117.0	121.0	4.0	3.71	2276	871	92.9	236		AX	D7-4	D8-3	Same Type - Quartz and Calcite along fractures. Small amount of disseminated (Pyrite).
121.0	122.6	1.6	1.15	707	below	72.2	15.4		AX	D7-5	D8-4	Same as above.
122.6	126.5	3.9	3.41	2088	784	87.4	15.4		AX	D7-5	D8-4	Same as above
126.5	129.0	2.5	2.03	1248	below	81.5	14.6		AX	D7-6	D8-5	Same as above. except some disseminated sulfides.
129.0	132.0	3.0	2.90	1779	744	96.8	14.6		AX	D7-6	D8-5	Same as above.
132.0	134.0	2.0	1.63	999	below	81.6	23.0	100.0	AX	D7-7	D8-6	Same as above. except small amount of sulfides.
134.0	136.4	2.4	2.32	1422	937	96.8	23.0	100.0	AX	D7-7	D8-6	Same as above. except no visible sulfides.
136.4	141.3	4.9	4.46	2734	878	91.1	19.4	100.0	AX	D7-8	D8-7	Same Type - Quartz and calcite along fractures, small amount of sulfides. Increased amount of Quartz.
141.3	142.9	1.6	1.54	943	below	96.2	31.9	100.0	AX	D7-9	D8-8	Same as above.
142.9	145.0	2.1	1.44	882	1091	68.6	31.9	100.0	AX	D7-9	D8-8	Same as above.

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BUREAU OF MINES
MINERALS EXPLORATION PROGRAM
DIAMOND DRILL HOLE LOG

HOLE NO: 2
Location:
Elevation of collar:
Depth:
Dip:
Bearing:
Date begun:
Date finished:

PROJECT No._____ Name _____
Location:

Theoretical wt. in grams of sludge per ft. of hole:
Bx--
Ax--
Ex--

Footage			Recoveries						Core Diam.	Sample Numbers		Description and Remarks
			Core		Sldg.	In percent						
From:	To:	Feet:	Feet:	Gms.	Gms.	Core	Sldg.	Wtr.	Ins.	Core	Sldg.	
145.0	150	5.0	4.74	2906	1244	94.8	26.9	100.0	AX	D7-10	D8-9	Same Type - Calcite ¼ inch wide along fractures. Sulfides in calcite and rock.
150	155	5.0	4.48	2744	1356	89.6	29.4	100.0	AX	D7-11	D8-10	Altered, light grey, fine grained, metavolcanic. Calcite and Quartz along fractures. Sulfides disemmated throughout. Lower end of core darker grey color.
155	160.2	5.2	4.12	2523	1345	79.2	28.0	100.0	AX	D7-12	D8-11	Same type - Calcite along fractures. Some disseminated sulfides (Pyrite?)
160.2	165.4	5.2	5.66	3469	1463	108.8	30.5	100.0	AX	D7-13	D8-12	Altered, light greenish grey color, fine grained, metavolcanic. Calcite and quartz along fractures. Increased amount of disseminated sulfides.
165.4	168.7	3.3	2.43	1488	below	73.6	24.8	100.0	AX	D7-14	D8-13	Same Type - Some disseminated sulfides (pyrite) and some brecciation.
168.7	171	2.3	2.26	1387	1282	98.2	24.8	100.0	AX	D7-14	D8-13	Same as above
171	173.5	2.5	2.03	1248	below	81.2	26.8	100.0	AX	D7-15	D8-14	Same Type - darker grey. No visible sulfides.
173.5	175	1.5	1.30	800	below	86.6	26.8	100.0	AX	D7-15	D8-14	Altered, grey color, Fine grained, metavolcanic. Considerable quartz and some calcite along fractures some disseminated sulfides.
175	176	1.0	.81	501	1239	81.0	26.8	100.0	AX	D7-15	D8-14	Same as above
176	178.6	2.6	2.15	1317	below	82.6	23.2	100.0	AX	D7-16	D8-15	upper end of core Same as above. Lower end, Altered, light greenish grey, fine grained, metavolcanic. Calcite and Quartz along fractures. Some disseminated sulfides.

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BUREAU OF MINES
MINERALS EXPLORATION PROGRAM
DIAMOND DRILL HOLE LOG

HOLE NO: 2
Location:
Elevation of collar:
Depth:
Dip:
Bearing:
Date begun:
Date finished:

PROJECT No. _____ Name _____
Location:

Theoretical wt. in grams of sludge per ft. of hole:
Bx--
Ax--
Ex--

Footage			Recoveries						Core Diam.	Sample Numbers		Description and Remarks
			Core		Sldg.	In percent						
From:	To:	Feet:	Feet:	Gms.	Gms.	Core	Sldg.	Wtr.	Ins.	Core	Sldg.	
178.6	181.5	2.9	2.62	1609	1181	90.3	23.3	100.0	AX	D7-16	D8-15	Altered, light grey, fine grained, metavolcanic. Calcite and Quartz along fractures. Some disseminated sulfides.
181.5	185.5	4.0	3.37	2062	864	84.2	15.2	100.0	AX	D7-17	D8-16	Same Type - Small amount of peridotite? at upper end. Intermediate rock type occurs along fractures in metavolcanics.
185.5	186.5	1.0	.90	554	below	90.0	29.0	100.0	AX	D7-18	D8-17	Altered, light grey, Fine grained, metavolcanic, fractured. Calcite along fractures. Small amount of disseminated sulfides.
186.5	191.5	5.0	4.64	2839	1606	92.6	29.0	100.0	AX	D7-18	D8-17	Same Type — irregular masses of calcite occur up to 5 inches in diameter.
191.5	195.5	~~3-5~~ 4.0	~~3.53~~ 3.19	~~1468~~ 1953	906	~~95.1~~ 79.7	24.5	100.0	AX	D7-19	D8-18	Same as above - except small amount of disseminated sulfides.
195.5	199.0	3.5	3.33	1168	below	95.1	38.1	100.0	AX	D7-20	D8-19	Same as above - except some disseminated sulfides
199.0	200.9	1.9	1.40	859	1902	73.6	38.1	100.0	AX	D7-20	D8-19	Same as above
200.9	202.2	1.3	.84	519	252	64.6	21.0	100.0	AX	D7-21	D8-20	Same as above.

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BUREAU OF MINES
MINERALS EXPLORATION PROGRAM
DIAMOND DRILL HOLE LOG

HOLE NO: 3
Location: 89' N. 7°W. Auger hole #9, Line 54
Elevation of collar: ~~2298~~ 2300
Depth: 117'
Dip: vertical
Bearing:
Date begun: 3/18/54
Date finished: 3/30/54

PROJECT No. 1222.2 Name _____
Location: Tekilma, Oregon

Theoretical wt. in grams of sludge per ft. of hole:
Bx--
Ax--
Ex--

Footage			Recoveries						Core Diam.	Sample Numbers		Description and Remarks
			Core		Sldg.	In percent						
From:	To:	Feet:	Feet:	Gms.	Gms.	Core	Sldg.	Wtr.	Ins.	Core	Sldg.	
0	5	5	0		58.3	[illegible]	[illegible]	Not much	NX		D8-22	No Core used NX Plug bit. No[t] much water return
5	10	5	0		0							" " " " " " " " "
10	13	3	0		[illegible] below	46.8	[illegible]	?			D8-23	No Core.
13	15.5	2.5	1.17	586	4033	46.8	142.4	100.0	BX 1⅝	D7-22	D8-23	Tan soil, light iron stain, light manganese stain, some round rock fragments. Composed predominately of hemitite.
15.5	20.0	4.5	2.15	1077	2635	47.7	113.7	100.0	BX	D7-23	D8-24	Same as above
20.0	25.0				3690		143.3	100.0	BX		D8-25	No Core
25.0	30.0				5784		224.6	100.0	BX		D10-1	" "
30.0	35.0				4662		181.0	100.0	BX		D10-2	" "
35.0	40.0				2486		96.5	100.0	BX		D10-3	" "
40.0	45.0	5	.059	[illegible]	5490	1.18	213.2	100.0	BX	D7-24	D10-4	Same as above (13-15.5) one small piece of metevolcanic.
~~45.0~~	~~50.0~~	~~5~~	~~.13~~	~~151~~		~~2.6~~		~~100.0~~	~~BX~~	~~D7~~		
45.0	50.0	5	.13	151	6960	2.6	117.3	100.0	BX	D7-25	D10-5	Same as above
50.0	55.0	5	0		5550		93.5		BX		D10-6	No Core
55.0	58	3	1.10	1277	below	36.6	45.2	100.0	BX	D9-1	D10-7	Upper end of core is decomposed metevolcanic has the appearance of soil mantle. lower end of core is altered, oxidized, heavy iron stained, moderately manganese stained, fine grained metevolcanic, fractured, brecciated Small amount of earthy limonite.

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BUREAU OF MINES
MINERALS EXPLORATION PROGRAM
DIAMOND DRILL HOLE LOG

HOLE NO:
Location:
Elevation of collar:
Depth:
Dip:
Bearing:
Date begun:
Date finished:

PROJECT No._____ Name _____
Location:

Theoretical wt. in grams of sludge per ft. of hole:
Bx--
Ax--
Ex--

Footage			Recoveries						Core Diam.	Sample Numbers		Description and Remarks
			Core		Sldg.	In percent						
From:	To:	Feet:	Feet:	Gms.	Gms.	Core	Sldg.	Wtr.	Ins.	Core	Sldg.	
58	60	2	.65	758	2681	32.5	~~45.2~~	100.0	BX 1 5/8	D9-1	D10-7	Same as lower end of core above
60	65	5	.169	195	4561	3.3	76.9	100.0	BX	D9-2	D10-8	Same Type- moderately iron stained. No earthy limonite.
65	65.5	.5	.18	209	below	36.0	60.0	100.0	BX	D9-3	D10-9	A few small pieces of metavolcanic same as above. Rest of core is decomposed metavolcanic, tan color, moderate iron stain, light manganese stain, moderately Kaolinized.
~~65.5~~	~~70.0~~	~~4.5~~	.52	321		~~11.5~~		100.0	AX 1 3/16	D9-3		
65.5	70.0	4.5	.52	321	2769	11.5	60.0	100.0	AX 1 3/16	D9-3	D10-9	Reduced to AX- upper end of core is altered, oxidized, heavy iron stained, moderate manganese stain fine grained, metavolcanic, greenish grey color, Brecciated, fractured. Lower end of core same as lower end of (65-65.5)
70.0	75.0	5	.294	107	3549	5.8	129.5	100.0	AX	D9-4	D10-10	Decomposed metavolcanic, light tan color, light iron stain, moderate manganese stained, heavy Kaolinized.
75.0	80.0	5	.484	176	6506	9.6	237.4	100.0	AX	D9-5	D10-11	Same as above
80.0	85.0	5	.228	83	4765	4.5	173.9	100.0	AX	D9-6	D10-12	Same as above
85.0	90.0	5	.32	117	3891	6.4	142.0	100.0	AX	D9-7	D10-13	Same as above
90.0	95.0	5.0	.82	299	1924	16.4	70.2	100.0	AX	D9-8	D10-14	Upper end of core is same as above. Lower end is decomposed metavolcanic, light iron stain, light manganese stained, moderately Kaolinized.

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BUREAU OF MINES
MINERALS EXPLORATION PROGRAM
DIAMOND DRILL HOLE LOG

HOLE NO: 3
Location:
Elevation of collar:
Depth:
Dip:
Bearing:
Date begun:
Date finished:

PROJECT No._____ Name _____
Location:

Theoretical wt. in grams of sludge per ft. of hole:
Bx--
Ax--
Ex--

Footage			Recoveries						Core Diam.	Sample Numbers		Description and Remarks
			Core		Sldg.	In percent						
From:	To:	Feet:	Feet:	Gms.	Gms.	Core	Sldg.	Wtr.	Ins.	Core	Sldg.	
95.0	~~100~~ 97.5	~~5~~ 2.5	.741	269	below	29.0	73.2	100.0	AX	D9-9	D10-15	Same as lower end of core above
97.5	100.0	2.5	.57	207	2006	22.8	73.2	100.0	AX	D9-9	D10-15	decomposed metavolcanic, tan color, light iron stained, light manganese stain, lightely Kaolinized
100.0	103.0	3.0	.269	98	below	8.9	156.9	100.0	AX	D9-10	D10-16	Same as above.
103.0	105.5	2.5	.34	210	5477	13.7	156.9	100.0	AX	D9-10	D10-16	Altered, oxidized, light iron stained, light manganese stained, metavolcanic, greenish grey color.
105.5	108.7	3.2	3.07	1116	below	95.9	51.9	100.0	AX	D9-11	D10-17	decomposed metavolcanic, light iron stained, moderate manganese stain, moderately Kaolinized.
108.7	110.0	1.3	.81	497	1379	62.4	51.9	100.0	AX	D9-11	D10-17	Upper end same as above. lower end is oxidized, altered, light iron stained, light manganese stained, lightly Kaolinized, greenish grey color, fine grained, metavolcanic, fractured, brecciated.
110.0	112.1	2.1	.614	376	below	29.2	55.3	100.0	AX	D9-12	D10-18	oxidized, altered, lightely iron stained, light manganese, fine grained, light grey metavolcanic, Quart occurs along fractures.
112.1	113	.9	.59	365	below	66.2	55.3	100.0	AX	D9-12	D10-18	Same type - some brecciation.
113	115	2.0	.41	253	2551	20.6	55.3	100.0	AX	D9-12	D10-18	Same type - less altered.
115	117	2.0	.681	417	1155	34.0	62.6	100.0	AX	D9-13	D10-19	Same as above.

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BUREAU OF MINES
MINERALS EXPLORATION PROGRAM
DIAMOND DRILL HOLE LOG

HOLE NO: 4
Location: 6' South Auger hole #15, Line 54
Elevation of collar: 2302
Depth: 78.5
Dip: Vertical
Bearing:
Date begun: 4/2/54
Date finished: 4/9/54

PROJECT No. 1222.2 Name _____
Location: Takilma, Oregon

Theoretical wt. in grams of sludge per ft. of hole:
Bx--
Ax--
Ex--

Footage			Recoveries						Core Diam.	Sample Numbers		Description and Remarks
			Core		Sldg.	In percent						
From:	To:	Feet:	Feet:	Gms.	Gms.	Core	Sldg.	Wtr.	Ins.	Core	Sldg.	
0	5	5			2311		12.2	100.0	NX		D10-20	
5	10	5			3936		20.9	100.0	NX		D10-21	
10	15	5	.146	169	2941	2.9	49.7	1000	BX 1⅝	D9-14	D10-22	Pieces of decomposed metavolcanics, tan color, light iron stain, light manganese stain, also pieces of oxidized, altered, light iron stain, light manganese stain, greenish gray, fine grained, metavolcanic, fractured, brecciated.
15	20	5	.237	162	6566	4.7	187.6	100.0	BX	D9-15	D10-23	decomposed metavolcanic. Same as above.
20	25	5	.44	303	6171	8.8	176.3	100.0	BX	D9-16	D10-24	decomposed metavolcanic, light iron stain, light manganese stain, highly Kaolinized, light tan color.
25	30	5	.38	441	8139	7.6	137.6	100.0	BX	D9-17	D10-25	Upper half is oxidized, altered, light iron stained, fine grained metavolcanic, light gray color, some quartz along fractures. Lower half is decomposed metavolcanic tan color, Moderately Kaolinized, light manganese stain.
30	35	5	.23	262	4.5			100.0	BX	D9-18		
							151.4					
30	35	5	.23	262	8959	4.5	~~895~~ 9	100.0	BX	D9-18	D12-1	decomposed metavolcanic. Same as run 20-25.
35	38	3	.43	494	below	14.2	136.5	100.0	BX	D9-19	D12-2	oxidized, altered, light iron stain, light manganese stained, greenish gray, fine grained, metavolcanic, moderately Kaolinized.

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MINERALS EXPLORATION PROGRAM
DIAMOND DRILL HOLE LOG

HOLE NO: 4
Location:
Elevation of collar:
Depth:
Dip:
Bearing:
Date begun:
Date finished:

PROJECT No._____ Name _____
Location:

Theoretical wt. in grams of sludge per ft. of hole:
Bx--
Ax--
Ex--

Footage			Recoveries						Core Diam.	Sample Numbers		Description and Remarks
			Core		Sldg.	In percent						
From:	To:	Feet:	Feet:	Gms.	Gms.	Core	Sldg.	Wtr.	Ins.	Core	Sldg.	
38	40	2	.52	602	8076	26.1	136.5	100.0	BX	D9-19	D12-2	Same type – moderate manganese stain. One piece of massive quartz, contains some sulfides.
40	45	5	1.11	1282	9897	22.2	167.3	100.0	BX	D9-20	D12-3	Oxidized, altered, light manganese stained, gray to tan, fine grained, metavolcanic, moderately Kaolinized. Fractured
45	45.5	.5	.16	186	below	32.2	116.4	100.0	BX	D9-21	D12-4	Same as above - except tan color.
45.5	50	4.5	.53	614	6887	11.8	116.4	100.0	BX	D9-21	D12-4	Same as run 40-45
50	53	3.0	.62	721	below	20.1	104.3	100.0	BX	D9-22	D12-5	Same as above
53	55	2.0	.211	244	6172	10.5	104.3	100.0	BX	D9-22	D12-5	Same as above - except lighter color.
55	60	5.0	.36	418	7617	7.2	138.8	100.0	BX	D9-23	D12-6	Oxidized, altered, light iron stain, light manganese stain, grey color, fine grained, metavolcanic. Quartz along fractures. Some sulfides (pyrite?). Some earthy hemitite.
60.0	65.0	5.0	1.17	1351	8277	23.4	139.9	100.0	BX	D9-24	D12-7	Upper end of core same type with some brecciation. Lower end is decomposed metavolcanic, highly Kaolinized, light gray color.
65.0	70.0	5.0	.53	619	18271	10.7	308.8	100.0	BX	D9-25	D12-8	Same as above.
70.0	75.0	5.0	.43	502	8198	8.7	138.6	100.0	BX	D11-1	D13-9	Oxidized, altered, light iron stain, light manganese stain, moderately Kaolinized, grey color, Some Quartz along fractures.

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BUREAU OF MINES
MINERALS EXPLORATION PROGRAM
DIAMOND DRILL HOLE LOG

HOLE NO: 4
Location:
Elevation of collar:
Depth:
Dip:
Bearing:
Date begun:
Date finished:

PROJECT No._____ Name _____
Location:

Theoretical wt. in grams of sludge per ft. of hole:
Bx--
Ax--
Ex--

Footage			Recoveries						Core Diam. Ins.	Sample Numbers		Description and Remarks
			Core		Sldg.	In percent						
From:	To:	Feet:	Feet:	Gms.	Gms.	Core	Sldg.	Wtr.		Core	Sldg.	
75.0	78	3	.35	406	below	11.7	~~100.0~~	100.0	BX	D11-2	D12-10	Same type - less Kaolinized. lower end of core is decomposed metevolcanic.
78	78.5	.5	.149	172	6269	29.8	151.4	100.0	BX	D11-2	D12-10	Same as above - except no decomposed metevolcanic

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BUREAU OF MINES
MINERALS EXPLORATION PROGRAM
DIAMOND DRILL HOLE LOG

HOLE NO: 5
Location: 50' S. 6° E. Auger hole # 21, line 54
Elevation of collar: 2263
Depth: 49.1
Dip: Vertical
Bearing:
Date begun: 4/13/54
Date finished: 4/15/54

PROJECT No. 1222.2 Name _____
Location: Takilma, Oregon

Theoretical wt. in grams of sludge per ft. of hole:
Bx--
Ax--
Ex--

Footage			Recoveries						Core Diam. Ins.	Sample Numbers		Description and Remarks
			Core		Sldg.	In percent						
From:	To:	Feet:	Feet:	Gms.	Gms.	Core	Sldg.	Wtr.		Core	Sldg.	
0	5	5			3005	[illegible]	15.9	100.0	NX Plug	~~Bit~~	D12-11	used Nx plug bit.
5	10	5			6983	[illegible]	37.1	100.0	NX Plug		D12-12	" " " "
10	15	6	.519	598	5429	10.3	91.8	100.0	BX 1⅝	D11-3	D12-13	oxidized, altered, light iron stain, light manganese stain, greenish grey color, fine grained meta volcanic. Small amounts of hemitite are disseminated in lower end of core (probably altered sulfides).
15	17	2.0	.77	896	below	38.8	77.6	100.0	BX	D11-4	D12-14	Altered, light iron stain, light manganese stain, greenish grey color, fine grained meta volcanic. Some brecciation. Quartz along fractures.
17	19.5	2.5	1.44	1668	4132	52.8	77.6	100.0	BX	D11-4	D12-14	Same as above.
19.5	20.7	1.2	.50	581	below	42.0	68.2	100.0	BX	D11-5	D12-15	Same as above.
20.7	25.7	5.0	3.22	3717	5008	64.5	68.2	100.0	BX	D11-5	D12-15	Same as above.
26.7	26.5	.8	.78	906	below	98.2	105.4	100.0	BX	D11-6	D12-16	Same as above - except some disseminated sulfides.
26.5	27.8	1.3	1.22	1409	below	94.0	105.4	100.0	BX	D11-6	D12-16	Same as above.
27.8	29.1	1.3	1.00	1162	4241	77.5	105.4	100.0	BX	D11-6	D12-16	Same as above - except increased amount of sulfides (pyrite).
29.1	30.5	1.4	1.33	1534	below	95.1	81.8	100.0	BX	D11-7	D12-17	Same as above.
30.5	31.6	1.1	1.06	1229	below	96.9	81.8	100.0	BX	D11-7	D12-17	Same as above - except calcite and quartz along fractures.

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BUREAU OF MINES
MINERALS EXPLORATION PROGRAM
DIAMOND DRILL HOLE LOG

HOLE NO: 5
Location:
Elevation of collar:
Depth:
Dip:
Bearing:
Date begun:
Date finished:

PROJECT No._____ Name _____
Location:

Theoretical wt. in grams of sludge per ft. of hole:
Bx--
Ax--
Ex--

Footage			Recoveries						Core Diam.	Sample Numbers		Description and Remarks
			Core		Sldg.	In percent						
From:	To:	Feet:	Feet:	Gms.	Gms.	Core	Sldg.	Wtr.	Ins.	Core	Sldg.	
31.6	32.6	1.0	.87	1005	below	87.2	81.8	100.0	BX 1⅝	D11-7	D12-17	Same Type - No visible disseminated sulfides. some disseminated hemitite.
32.6	33.7	1.1	.64	741	449	58.4	81.8	100.0	BX	D11-7	D12-17	Same Type - Some disseminated sulfides. No disseminated hemitite.
33.7	34.3	.6	.49	566	below	81.8	60.9	100.0	BX	D11-8	D12-18	Same as above - ex
34.3	35.8	1.5	1.37	1584	below	91.6	60.9	100.0	BX	D11-8	D12-18	Same as above - except no visible disseminated sulfides.
35.8	37.5	1.7	1.56	1802	below	92.0	60.9	100.0	BX	D11-8	D12-18	Same as above.
37.5	39.0	1.5	1.41	1626	38.6	94.0	60.9	100.0	BX	D11-8	D12-18	Same as above. - except some disseminated sulfides. Sulfides appear to occur close to black mineral (or rock) which is disseminated in core.
39.0	42.5	3.5	3.24	3738	below	92.6	59.1	100.0	BX	D11-9	D12-19	Upper end of core same as above - except no visible disseminated sulfides. Lower end of core is dark gray, Altered, light iron stain, light manganese stain, fine grained metavolcanic, Quartz and calcite along fractures.
42.5	43.0	.5	.32	370	below	64.2	59.1	100.0	BX	D11-9	D12-19	Same as lower end of core above - except some disseminated sulfides.

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BUREAU OF MINES
MINERALS EXPLORATION PROGRAM
DIAMOND DRILL HOLE LOG

HOLE NO: 5
Location:
Elevation of collar:
Depth:
Dip:
Bearing:
Date begun:
Date finished:

PROJECT No. _____ Name _____
Location:

Theoretical wt. in grams of sludge per ft. of hole:
Bx--
Ax--
Ex--

Footage			Recoveries						Core Diam.	Sample Numbers		Description and Remarks
			Core		Sldg.	In percent						
From:	To:	Feet:	Feet:	Gms.	Gms.	Core	Sldg.	Wtr.	Ins.	Core	Sldg.	
43.0	44.0	1.0	.98	1137	3498	98.6	59.1	100.0	BX1⅝	D11-9	D12-19	Same as above.
44.0	46.0	2.0	1.82	2103	below	91.2	51.5	100.0	BX	D11-10	D12-20	Same as above - except no visible disseminated sulfides.
46.0	49.1	3.1	2.39	2762	3109	77.3	51.5	100.0	BX	D11-10	D12-20	Same as above.

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BUREAU OF MINES
MINERALS EXPLORATION PROGRAM
DIAMOND DRILL HOLE LOG

HOLE NO: 6
Location: 17 ft. S. 18° E. Auger hole #3, Line 53
Elevation of collar: 2281
Depth: 77 ft.
Dip: Vertical
Bearing:
Date begun: April 20, 1954
Date finished: April 27, 1954

PROJECT No. _____ Name _____
Location:

Theoretical wt. in grams of sludge per ft. of hole:
Bx--
Ax--
Ex--

Footage			Recoveries						Core Diam.	Sample Numbers		Description and Remarks
			Core		Sldg.	In percent						
From:	To:	Feet:	Feet:	Gms.	Gms.	Core	Sldg.	Wtr.	Ins.	Core	Sldg.	
0	5	5							NX Plug			used NX Plughit. No water return.
5	10	5			5076		100.5		NX Plug		D12-21	" " " "
10	15	5	.46	459	4562	9.3	90.3	100.0	BX 1⅝	D11-11	D12-22	Decomposed metavolcanic, light iron stain, light manganese stain, moderately Kaolinized, tan color.
15	20	5	.60	599	5659	12.1	112.0	100.0	BX	D11-12	D12-23	Same as above
20	25	5	.19	188	14699	3.8	291.0	100.0	BX	D11-13	D12-24	Same as above
25	30	5	.11	113	7531	2.2	149.1	100.0	BX	D11-14	D12-25	Same as above - except highley Kaolinized.
30	35	5	.46	456	7209	9.6	142.7	100.0	BX	D11-15	D12-1	Pieces same as above. Some preces of oxidized, altered, light iron stain, fine grained, metavolcanic.
35	38	3	1.57	1561	below	52.5	185.9	100.0	BX	D11-16	D13-2	upper end of core same as above. Lower end of core is decomposed metavolcanic, light manganese stain, moderately Kaolinized, tan color.
38	40.1	2.1	.55	548	9387	26.4	185.9	100.0	BX	D11-16	D13-2	Decomposed metavolcanic, light manganese stain, moderately Kaolinized, tan color.
40.1	45.2	5.1	2.01	1982	6661	39.5	129.3	100.0	BX	D11-17	D13-3	Same as above.
45.2	48.8	3.6	1.16	1144	6793	32.2	186.8	100.0	BX	D11-18	D13-6	Same as above.

Form SM-18 (Revised April, 1945)
113803

BUREAU OF MINES
MINERALS EXPLORATION PROGRAM
DIAMOND DRILL HOLE LOG

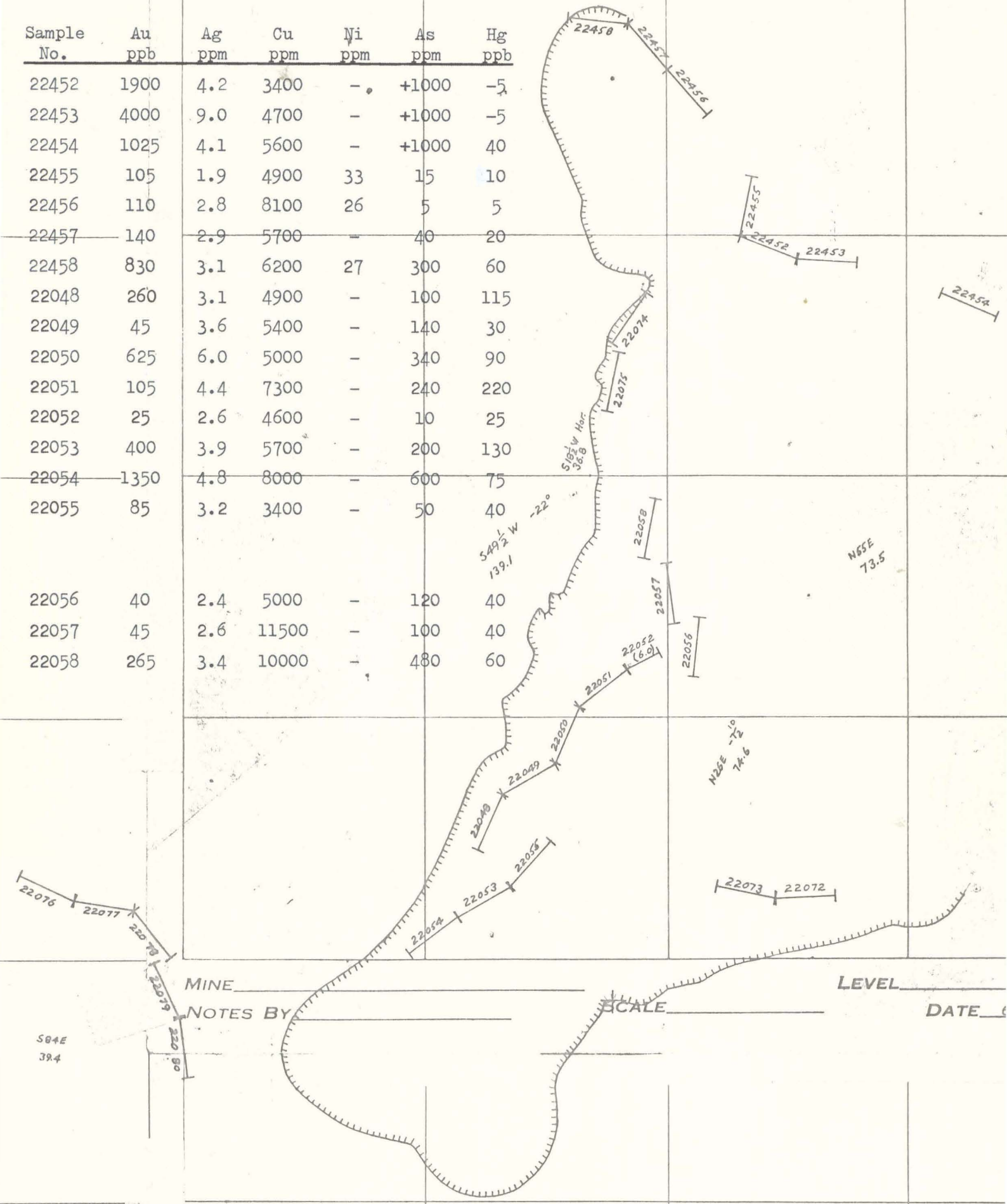
HOLE NO: 6
Location:
Elevation of collar:
Depth:
Dip:
Bearing:
Date begun:
Date finished:

PROJECT No._____ Name _____
Location:

Theoretical wt. in grams of sludge per ft. of hole:
Bx--
Ax--
Ex--

Footage			Recoveries						Core Diam.	Sample Numbers		Description and Remarks
			Core		Sldg.	In percent						
From:	To:	Feet:	Feet:	Gms.	Gms.	Core	Sldg.	Wtr.	Ins.	Core	Sldg.	
48.8	52	3.2	1.19	622	below	37.2	82.5	80.0	AX 1⅞	D11-19	D13-7	Reduced to AX. Same type as above.
52	55	3.0	1.45	757	4264	48.4	82.5	80.0	AX	D11-19	D13-7	Same as above.
55	57.7	2.7	1.70	889	below	63.1	76.9	80.0	AX	D11-20	D13-8	Same as above. Only lighter tan color.
57.7	59.5	1.8	.95	495	2722	52.7	76.9	80.0	AX	D11-20	D13-8	Same as above.
59.5	62.5	3.0	1.55	808	below	51.6	77.0	80.0	AX	D11-21	D13-9	Same as above-except darker tan color.
62.5	65.5	3.0	1.04	541	3632	34.6	77.0	80.0	AX	D11-21	D13-9	Same as above.
65.5	67.3	1.8	1.37	713	below	76.0	118.0	80.0	AX	D11-22	D13-10	Same as above
67.3	71.1	3.8	.34	178	5194	8.9	118.0	80.0	AX	D11-22	D13-10	Upper end of core same as above. Lower end pieces of altered, oxidized, light iron stain, light manganese stain, fine grained, metavolcanic, greenish gray color.
71.1	72.6	1.5	.39	142	below	26.3	85.1	80.0	AX	D11-23	D13-11	Same as lower end above.
72.6	74.6	2.0	.49	302	below	24.6	85.1	80.0	AX	D11-23	D13-11	Same as above-except tan to greenish gray color.
74.6	75.8	1.2	.62	378	3692	51.4	85.1	80.0	AX	D11-23	D13-11	Same as above-except greenish gray color.
75.8	77.0	1.2	.85	523	1369	71.1	123.6	80.0	AX	D11-24	D13-12	Same as above.

Sample No.	Au ppb	Ag ppm	Cu ppm	Ni ppm	As ppm	Hg ppb
22452	1900	4.2	3400	-	+1000	-5
22453	4000	9.0	4700	-	+1000	-5
22454	1025	4.1	5600	-	+1000	40
22455	105	1.9	4900	33	15	10
22456	110	2.8	8100	26	5	5
22457	140	2.9	5700	-	40	20
22458	830	3.1	6200	27	300	60
22048	260	3.1	4900	-	100	115
22049	45	3.6	5400	-	140	30
22050	625	6.0	5000	-	340	90
22051	105	4.4	7300	-	240	220
22052	25	2.6	4600	-	10	25
22053	400	3.9	5700	-	200	130
22054	1350	4.8	8000	-	600	75
22055	85	3.2	3400	-	50	40
22056	40	2.4	5000	-	120	40
22057	45	2.6	11500	-	100	40
22058	265	3.4	10000	-	480	60



MINE_____ LEVEL_____

NOTES BY_____ SCALE_____ DATE_____

Sample No.	Au ppb	Ag ppm	Cu ppm	As ppm	Hg ppb
22072	20	2.4	1350	20	600
22073	50	2.2	530	25	170
22074	425	3.0	5400	430	120

Σ Angles

17.5366	1st Sunshot
17.5185	
17.54209	
17.58775	2nd Sunshot
17.59281	
17.60089	
17.5254	4th Sunshot
17.52787	
17.52114	

157.9530499 ÷ 9 = 17.55033888

17° 33' 01"

Final Bearing of Line 1001 to 1002 is

S 17° 33' 01" E

April 4, 1977
Page 4

Largest E.O.C. = 19.2'

1st Sunshot

	Horz ∠	Vert ∠	
Dir	67° 52'	43° 00'	2:34
Dir	68° 14'	42° 49'	
Dir	68° 29'	42° 40'	
Rev	68° 20	41° 50'	
Rev	68° 48'	41° 33'	
Rev	69° 07'	41° 26'	2:42
	6√408° 170' =	6√250° 198'	2:38
	68° 28.3'	41.67° 33'	
		42° 13'	

Bearing
S 17° 32.2' E
✓ S 17° 32' 12" E

17.5366°

Dir	70° 56'	41° 21'	2:46
Dir	71° 13'	41° 11'	
Dir	71° 27'	41° 01'	
Rev	71° 07	40° 17'	
Rev	71° 31'	40° 02'	
Rev	71° 50'	39° 54'	2:52
	6√425° 184' =	6√242° 106'	2:49
	70.83° 30.67'	40.33° 17.67'	
	71° 20.67'	40° 37.67'	

S 17.5185E
✓ S 17° 31' 6.6" E

~~S 17° 32.4' E~~
~~S 17° 32' 24" E~~

17.54°

Dir	73° 37'	39° 47'	2:56
Dir	73° 50'	39° 37'	
Dir	74° 07'	39° 29'	
Rev	73° 46'	38° 44'	
Rev	74° 07	38° 32'	
Rev	74° 21	38° 22'	3:02
	6√441° 168' =	6√231° 211'	2:58
	73.5° 28'	38.5° 35.167'	
	73° 58'	39° 05.167'	

S17.54209
✓ S 17° 32' 32"

~~S 17° 32.78' E~~
~~S 17° 32' 46.8" E~~

17.5463

Ave. ~~Angle~~
S 17° 32' 27.6" E

2nd Sunshot

	Horz ∠	Vert ∠	Time	
✓✓ Dir	312° 39'	33° 04'	8:58	
✓✓ Dir	312° 49'30"	33° 12'		~~S 17.58846° E~~
✓✓ Dir	313° 03'	33° 23'		~~S 17° 35' 04.6" E~~
✓✓ Rev	312° 45'	34° 12'		S 17.587753 E
✓✓ Rev	313° 05'	34° 27'		S 17° 35' 16" E
✓✓ Rev	313° 17'	34° 35'	9:04	
	√1875 158.5	√201 113	9:01	
✓✓	312.5° 26.416	33.5° 18.833'		
	312° 56.416	33° 48.833'		
✓✓ Dir	315° 01'	34° 52'	9:08	
✓✓ Dir	315° 15'	35° 00'		~~S 17.58945° E~~
✓✓ Dir	315° 25'	35° 08'		~~S 17° 35' 22" E~~
✓✓ Rev	315° 00'	35° 51'		17. 592806
✓✓ Rev	315° 16'	36° 02'		S 17° 35' 34" E
✓✓ Rev	315° 26'	36° 09'	9:13	
	√1890 83	√211 122	9:10.5	
✓✓	315° 13.833'	35.167° 20.333'		
		35° 30.333'		
✓✓ Dir	317° 01'30"	36° 16'	9:17½	
✓✓ Dir	317° 13'	36° 24'		~~S 17.59718° E~~
✓✓ Dir	317° 30'	36° 35'		~~S 17° 35' 49.85" E~~
✓✓ Rev	317° 10'	37° 23'		S 17.600898 E
✓✓ Rev	317° 28'	37° 33'		S 17° 36' 3" E
✓✓ Rev	317° 44'	37 44'	9:23	
	√1902 126.5	√219 175	9:20.25	
✓✓	317° 21.08333'	36.5 29.167		
		36° 59.167'		

Ave ~~17.59041~~
~~17° 35' 25.5"~~

3rd Sunshot X No Good

Dir	321° 30'	39° 10'	9:36
Dir	321° 41'	39° 18'	
Dir	321° 50'	39° 24'	
Rev	321° 32	40° 08'	
Rev	321° 54'	40° 23'	
Rev	322° 16'	40° 34'	9:41
	√1927° 223'	√237° 117'	9:38.5
	321.167° 37.167'	39.5° 19.5	
✓	321° 47.167'	39° 49.5'	

S 18.063349° E
S 18° 03' 48" E

Dir	323° 21'30"	40° 18'	9:43
Dir	323° 53'	40° 35'	
Dir	324° 06'	40° 43'	
Rev	323° 44	41° 28'	
Rev	324° 06'	41° 38'	
Rev	324° 19'	41° 46'	9:49
	√1941° 149.5'	√243° 208'	9:46
	323.5° 24.9167'	40.5° 34.67'	
✓	323° 54.9167'	41° 04.67'	

S 17.610057° E
S 17° 36' 38" E

Dir	325° 37'	41° 24'	9:52
Dir	325° 55'	41° 43'	
Dir	326° 07'	41° 52'	
Rev	325° 30'	42° 41'	
Rev	326° 17'	42° 51'	
Rev	326° 29'	42° 59'	9:57
	√1953° 175'	√249° 280'	9:54.5
	325.5° 29.167'	41.5° 46.67'	
✓	325° 59.167'	42° 16.67'	

S 17.49891° E
S 17° 29' 56" E

April 7, 1977
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4th Sunshot

	Horz ∠	Vert ∠	Time	
✓✓ Dir	65° 05'	45°41'	2:22	
✓✓ Dir	65° 26'	45°31		
✓✓ Dir	65° 39'	45°26		~~S 17° 31' 52" E~~
✓✓ Rev	65° 19'	44°44		S 17° 31' 31" ✓
✓✓ Rev	65° 39' 30"	44°33		S 17.5254°
✓✓ Rev	66° 05'	44°22	2:28	
✓✓	65.166° 22.25	44.5° 32.83'	2:25	17.53121
	65° 32.25'	45° 02.83'		
	65° 32' 15"			
✓✓ Dir	68° 12'	44° 11'	2:33	
✓✓ Dir	68° 32'	44°01'		
✓✓ Dir	68° 43'	43°56'		~~S 17.53311° E~~
✓✓ Rev	68° 18' 30"	43°14'		~~S 17° 31' 59.2" E~~
✓✓ Rev	68°46'	43°00'		
✓✓ Rev	69° 04'	42° 50' 30"	2:38.5	S 17, 527877
	68.166° 25.9166'	43.166° 22.083'	2:35.5	S 17° 31' 40" E
	68° 35.9166'	43° 32.083'		
	68.59861			
✓✓ Dir	70°35'	42°57'	2:41	
✓✓ Dir	70°50'	42° 48'30"		~~S 17.526° E~~
✓✓ Dir	71° 08'	42°39'		~~S 17° 31' 33.7"~~
✓✓ Rev	70°47'	41°55'		
✓✓ Rev	71°01'	41°42'30"		S 17.52114 E
✓✓ Rev	71°24'	41°33'	2:47.5	S 17° 31' 16" E
✓✓	70.5 °27.5'	41.5° 45.833'	2:44.25	
	70° 57.5'	42° 15.833'		

Latitude = 42° 03' 01.4"

✓ 1st 2:38
✓ 2 12:
✓ 3 14:38
✓ 4 8:14.4
✓ 5 22:52.4 = GCT

22:52.4
22.873

✓ 6 +05° 33.8' Table 1
✓ 7 + 21.73' (22.873 × 0.95)
d ✓ 8 05° 55.53' Sans d ≈ 5.9255°
9 42° 13'
10 (1.06) +(0.11) Table 2
−(0.93)
+(0.95)
−(0.937) +(0.11)
−.827
→ −.8
h 42° 12.2' ≈ 42.2° True altitude

z [.2 [42.0 1.07] x] .07
42.2
44 1.00

$\frac{.2°}{2°} = \frac{x}{.07}$, x = .007

1.07
− .007
1.063

$\cos(z) = \frac{\sin d - \sin h \sin l}{\cos h \cos l}$

42° 03.02'

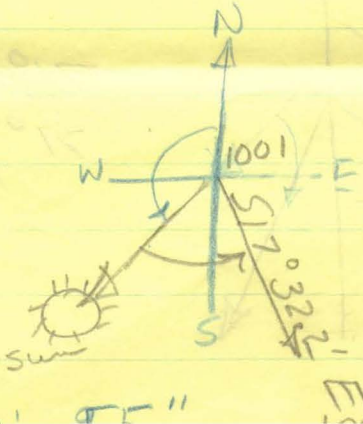
$\cos(z) = \frac{\sin(5°55.53') - [\sin(42°12.2')\sin(42°03.2')]}{\cos(42°12.2')\cos(42°03.2)}$

.6717

$\cos z = \frac{\sin(5.9255°) - [\sin(42.2°)\sin(42.0503)]}{\cos(42.2°)\cos(42.05°)}$

$\cos z = +.103235 - .449904 = -\frac{.346669}{.550092}$ = 89

$\cos z = -.630202$
$z = 129.065027$
$z = 129° 3.9'$



129
98
89 59 60
39 3 55
50 56 05
68 28 18
50 56 05
17 32 13

7 8
68 28.3
50 56.1
S17 32.2E

89 60
39 3.9
S50° 56.1'W

3) 14:49

4) 8:14.4

5) 23:03.4 = 6 CT

6) 5° 33.8 Table

7) 21.9 (23.0567 × 0.95)

8) 5° 55.7' ⟶ d 5.928°

9) 40° 37.67' = 40.628° $\frac{.628°}{2°} = \frac{x}{.08'}$ x = -0.025 ~~2~~ -1.15 / 1.125

10) [(1.12) + (.11)
(.93)
(.95)]

- (.99 + (.11)
-0.88

40° 36.79' ⟶ h 40.613

$$\cos z = \frac{\sin(5.928°) - [\sin(40.613°)\sin(42.053°)]}{\cos(40.613°)\cos(42.053°)}$$

d h l h l

.66978275 .742557109

$$\cos z = \frac{0.10327863 - [0.4360155]}{0.5636\cancel{6869}}$$

.926496

-.332,714

$\cos z = -.590\cancel{305752}$ 240257

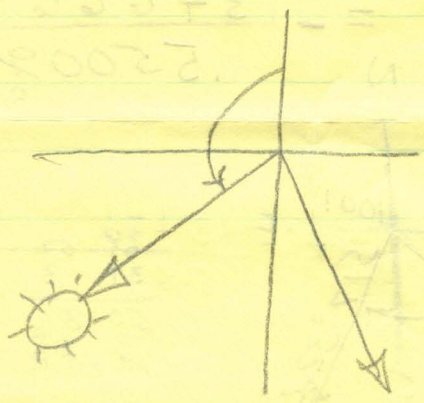
z = 126.179° = 126.174059 - 90 = 36.174 (90) / 53.825

z = 126° 10' 44.4" = 36° 10' 44.4" (89 59 60)

S 53 49 16 W

71.343
53.825
17.5185

$\boxed{\$17° 31' 6.6''}$



~~71° 21.67~~

71° 21.67'

~~71° 21' 40"~~ (70 81)

53 49 16

~~S 17 32 24 E~~

S 17° 31' 19."

✓1 2:59
✓2 12:
✓3 14:59
✓4 8:14.4
✓5 23:13.4 = GCT

✓6) 05° 33.8
✓7 + 22.06 (23.223 x 0.95)
(d) ✓8 05° 55.86 5.931°
✓9 39° 05.167'
✓10 (1.187) + (0.115) $\frac{1.086°}{2°} = \frac{x}{0.08}$ x = 0.043 (1.23)
✓ 0.93) ✓ 1.187
✓ 0.95)
✓ -1.049 + (0.115)
✓ - .934
(h) ✓11) 39° 04.233' 39.07~~055~~ 1

$$\cos z = \frac{\sin 5.931° - [\sin(39.071°)\sin(42.0503)]}{\cos(39.071°)\cos(42.0503)}$$

$$\cos z = \frac{(.10333071) - [(.6302829)(0.66981775)]}{(\quad)(\quad)}$$ ~ .669782755 ← .742557109

$$\cos z = \frac{(.10333071) - .422174695}{(.77636552)(.74252555)}$$

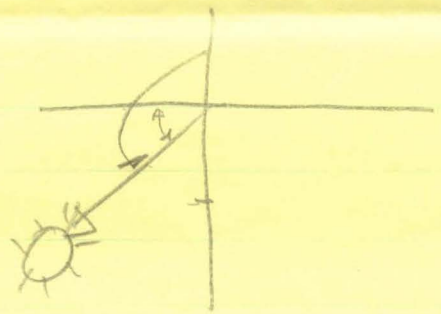
- .318821933
.5764~~71231~~ 95736

cos z = -.5530~~96092~~ 5431

cos z = 123.579678 89' 60.
123° 34.78' = 33° 34.78'
S 56 25.22

123.5754
-90
33.575
71
- 56.42456
17.5420

S 17° 32' 32



73 58
-56 25.22
S 17 32.78 E
S 17° 32' 46.8" E

4 8:14.4

✓5 17:24.9 GCT

✓6) 5° 56.7

✓7) 16.54' (17:24.9 × 0.95) = 17.415

(d) 8) 6° 13.24' ← (d)

9) 35° 30.333'

10) (1.345) + (0.12)

(0.93)

(0.95)

−1.1883 +

− 1.0683

✓(11) 35° 29.265' (h)

(6°.22067°)

✓cos z = sin(6° 13.24') − sin(35.48775) sin(l)

cos(35.48775) cos(l)

= (.108358) − .38884855

.604593748

✓cos z = − ~~.46393227~~ = .46387895 = 117.6376

−90

27.6376

270

S 297.6376

z = 117.6411

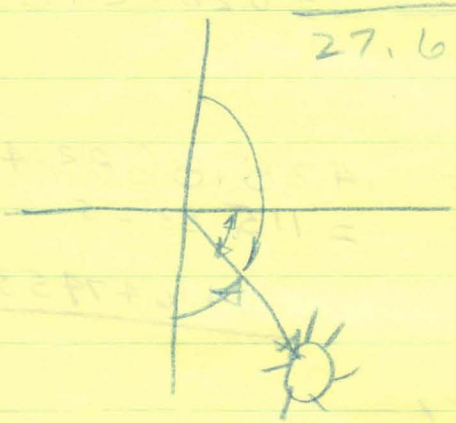
−90

27.6411

90.

27.6411

S 62.3589 E Sun



✓117.6411

180

297.6411

13.833

✓315.2305

297.6411

17.58945°

✓S 17° 35' 22" E

315.2305

297.63769

17.592806

17° 35' 34"

1 9.01

~~2~~

2 ____

3 9:01

4 8:14.4

5 17:15.4

6 05 56.7

d 7 16.394

8 6° 13.094'

9 33° 48.833'

10 -1.43

.93

.95

-1.26 + 0.12

1.14

h 33°47.69

$$\cos z = \left[\frac{\sin(6.2183^\circ) - (\sin(33.79483^\circ))(\sin 42.053^\circ)}{\cos(33.79483^\circ)\cos(42.053)}\right]$$

$$= \frac{(.10831688) - (.372566451)}{(.6170644695)} = 0.264249571$$

$\cos z = -.428\cancel{2365689} = .4281868314$

$= 115.3525$

-64.647453

17.587753

S17° 35' 16"

$z = 115.3557^\circ$

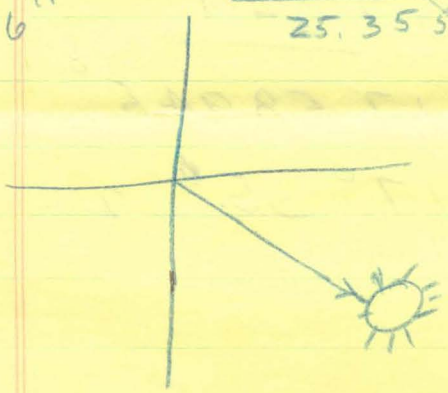
90

25.3557°

90.

25.3557

S 64.6443 E Sum



270

25.3557

295.3557

312.9403

295.3557

S 17.5846 E

S 17° 35.076 E

S 17° 35' 04.6" E

1) 9:20.25

2) _____

3) 9:20.25

4) 8:14.4

5) 17:34.65 ~~hr~~ × 0.95 dec/~~hr~~ = 'dec

6) 5° 56.7'

7) 16.698 (17.5775 × 0.95)

(d) → 8 6° 13.398'

9) 36° 59.167' 36.986°

10 (1.275) + (0.12)

(.93)

(.95)

− (1.1264) + 0.12

− 1.00646

(h) → (11) 36° 58.16054'

6013876 6698178

42.0503

$$\cos z = \frac{\sin(6.2233°) - (\sin(36.969342)\sin(l))}{\cos(36.969342)\cos l}$$

$$= \frac{(.1084036299) - .6013876(}{-.4028200896}$$

−.49622381

119.7505 − .2944164597

299.7505 .5932462953

cos z = −.4962803174

z = 119.7542°

180

299.7542

36' 3.2" 317.35138

299.7542

17.59718

S 17° 35' 49.85" E

April 5, 1977

(1) Page 7

3rd Sunshot

1) 9:38.5
2)
3) 9:38.5
4) 8:14.4
5) 17:52.9

6) 5° 56.7'
7 16.987 (17.881667 × .95) =
d → 8 6° 13.687
9) 39° 49.5' ⟷ (39.825°)
10) (1.157) + (0.11)
.93
.95
−1.0222 + 0.11
−0.912
(h) 11 39° 48.5877

$0.08\left(\frac{1.825}{2}\right) = x$

2 ⌈ 38
1.825
40

$\frac{1.825}{2^0} = \frac{x}{0.08}$

x = 0.073

$(0.08)\frac{1.825}{2} = \frac{x}{\cancel{0.08}}\cancel{(0.08)}$

$\cos z = \frac{\sin(\cancel{39.825}\ 6.22812^\circ) - \sin(39.809795)\sin l}{\cos(39.809795)\cos l}$

$= \frac{.108487 - (.428845)}{.5770388883}$

.32035 7548
.4288449

z − ~~.5616476029~~

.5551749724

z = ~~124.1698~~
= 123.722767
180
303.722767

321.7861167
303.722767
18.063349

18° 03' 48"

(2)

1) 9:46
2) ———
3) 9:46
4) 8:14.4
5) 1860.4 − 18:00.4

6) 5°56.7′
7) 17.1063 (18.00667 × .95)
8) 6°13.806 6.23301°

9) 41°04.67′ 41.07780
10) −1.11 +0.11
.93
.95
−0.9807 + 0.11
− 0.8707

H) 41° 3.8′ = 41.06333

$$\cos z = \frac{\sin(6.2301°) - \sin(41.06333)\sin l}{\cos(41.0633)(\cos l)}$$

180
126.3047
306.3047

323.9152
306.3047
17.6105 = .10 − .6568929(

.1085216 .43999852

= −.3314769 / .5598523 ✓

S 17.61057°E
17°36′38″ = −.59207919

= 126.3047

April 5
Page 7

(3)

1 9:54.5
2 _____
3 9:54.5
4 8:14.4
5 18:08.9

6) 5° 56.7'
7 + 17.24
d 8 6° 03.94

(18.14833 x .95)
6.2323333

9 42° 16.67'
10 (1.06) + (.11)
.93
.95
- .9365 + .11
= .8265

(42.277783°)

$\frac{.28}{2} = \frac{x}{0.07}$ $x = .01$

11 42° 15.8435' = 42.26406°

$\cos z = \frac{(\sin 6.232333°) - \sin(42.26406)\sin l}{\cos(42.26406)(\cos l)}$

$\frac{.45048485}{(.1085604) - (.6725484 \times .6698775)}$
$.7400532 \times .7425255$

128.4872

-.34198085
.5495084

17.49891
S 17 29 56 E

-.6223396
128.4872

April 7, 1977
Page 8

4th Sunshot

1 2:25
2 12
3 14:25
4 8:14.4
5 22:39.4

6) 6° 42.1'
7) ~~[illegible]~~9 (22.6566 × 0.94)
21.3
✓D 8 7° 03.4 = 7.05666°
9 45° 02.83'
10 (.965) + 0.105
(.93)
(.95)
− .8526 + 0.11
− .7526
✓H 11 45° 2.08' = 45°.03466

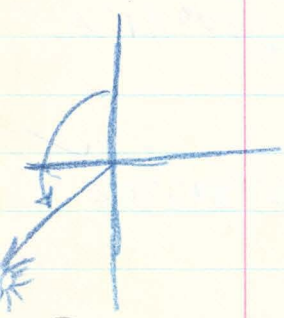
42.25
$\frac{.25}{2.0} = \frac{x}{.07}$

42.0503

$$\cos z = \frac{\sin 7.05666 - [\sin 45.03466 \sin 42.053°]}{\cos 45.03466 \cos 40.0503}$$

$$= \frac{.1228509 - [.7075345(\quad)]}{.7066788(\quad)}$$

$$\frac{.3510682}{.52472706}$$



$\cos^2 =$ ~~.66904916~~ .6689734073 = 131.987881

$z = 131.9937°$ = 131° 59' 37"

131° 59' 37"
90
41° 59' 37"
89 59 60
41 59 37
S 48 00 23 W

S 17° 31' 31"

65° 31' 15"
48 00 23
S 17 31 52 E

65.5375
−48.0121
S 17.5254 E

(2)

1 2:35.75
2 12
3 14:35.75
4 8:14.4
5 22:50.15

6) 6° 42.1
7 21.47 (.94 x 22.8358)
D. 8 7° 03.57′ = 7.0593°
9 43° 32.083′ $\frac{1.5}{2.0} = \frac{+}{.07}$ x = .05
10 1.02 + .11
.93
.95
.90 + .11
10 − .79
H. 11 43° 31.293′ 43.52155°

$\frac{7.0593 - (43.52155)(\quad)}{(43.52155(\quad)}$

0.62836054
= 128.929267
− 90
90
38.929
68.59861
51.070733

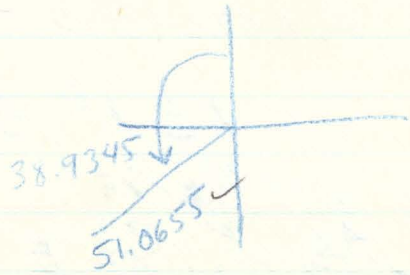
$\frac{\text{SIN D} - (\text{SIN H} \cdot \text{SIN L})}{\text{COS H} \cdot \text{COS L}}$

.46125485 | 17.52787

$\frac{.1228965 - (.6886274 \cdot .66981775)}{.7251155 \cdot .74252555}$ S 17° 31′ 40″

.33835835

128.9345 $\frac{\quad}{.53841678}$ = .62843202



68.59861
17.53311 17° 31′ 59.2″

APRIL 7

(3.)

1. 244.25
2. 12
3. 14:44.25
4. 8:14.4
5. 22:58.65 = GCT

✓ 6. 6° 42.1'

7. (.94 X 22.9775) = 21.59885

D ✓ 8. ~~7~~° 3.69885' ✓ 7.06° -

B 9. 42° 15.833'

10. -(1.06) + .11 = .95

.93
.95
-.93651 + .11
-.82651

h ✓ ~~42~~° 15' 42.25° -

$$\frac{7.06 - (42.25)(42.050}{42.25(42.0503)}$$

$$\cos Z \qquad \frac{\sin D - \sin H \cdot \sin L}{\cos H \cdot \cos L}$$

= -0.595703635
= 126.562

$$\cos Z = \frac{(\sin 7.06) - (\sin 42.25 \cdot .66981775)}{\cos 42.25 \cdot .74252555}$$

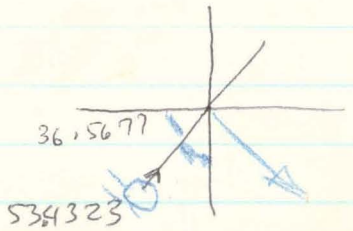
70.958
-53.43719
17.52114

S17° 31' 16"

$$\cos z = \frac{.1229087 - (.45036321)}{.54963092}$$

$$= \frac{-.32745451}{.54963092} \qquad \cos^{-1} .59577163 \quad 53.43234$$

126.5677



.9583333
17.526033 (17° 31' 33.7")

APRIL 4, 1977

(PAGE 4)

AV. TIME

	248 21	248 49	249 08	
	+ 179 59	+ 179 59	+ 179 59	
1ST	428 80	427 108	428 67	2:38
	- 360 00	- 360 00	- 360 00	
	REV. 68° 20'	REV. 68° 48'	REV. 69° 07'	
2ND	251 08	251 32	251 51	
	+ 179 59	+ 179 59	+ 179 59	
	430 67	430 91	430 110	2:49
	- 360 00	- 360 00	- 360 00	
	71° 07'	71° 31'	71° 50'	
3RD	253 47	254 08	254 22	
	+ 179 59	+ 179 59	+ 179 59	
	432 106	433 67	433 81	2:56
	- 360 00	- 360 00	- 360 00	
	72 46	73 07	73 21	
	73 46	74 07	74 21	

APRIL 5, 1977

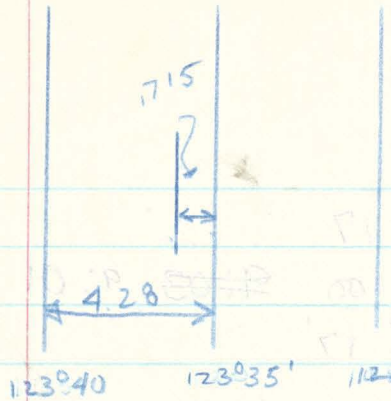
(PAGE 6)

1ST	132 45	133 05	133 17	~~9:03~~	9:01
+	180 00	180 00	180 00		
	312° 45'	313° 05'	313° 17'		
2ND	135 00	135 16	135 26		
	180 00	180 00	180 00	9:10.5	
	315° 00'	315° 16'	315° 26'		
3RD	137 10	137 28	137 44		
	180 00	180 00	180 00	9:20.25	
	317° 10'	317° 28'	317° 44'		

APRIL 5, 1977

(PAGE 7)

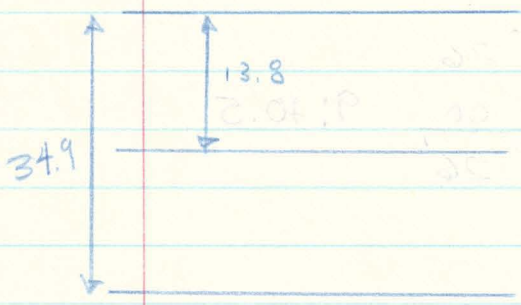
1ST	141 31	141 53	142 15		
	180 00	180 01	180 01	9:38.5	
	321° 32'	321° 54'	322° 16'		
2ND	143 43	144 05	144 18		
	180 01	180 01	180 01	9:46	
	323° 44'	324° 06'	324° 19'		
3RD	145 29	146 16	146 28		
	180 01	180 01	180 01	9:54.5	
	325° 30'	326° 17'	326° 29'		



$\frac{.715}{4.28} = \frac{x}{5}$ $x = .835$

$x = 50.1''$

Long = 123° 35' 50.1"



$\frac{13.8}{34.9} = \frac{x}{5}$ $x = 1.977$

$x = 1° 58.6''$

Lat. = 42° 03' 01.4"

Corr Long = .120 = 8 hrs

14 min or 8:14.4

3° 35' 50.1"

3° 35.835'

3.59725°

$\frac{3.59725}{15} = \frac{x}{1\text{ hr}}$

$x = .23982$ hr $\times 60$ min/hr

$x = 14.389$ mins

April 7 1977
Page 8

1ST

245° 19'	245° 39' 30"	246° 05'
+ 180° 00'	+ 180° 00' 00"	+ 180° 00'
425° 19'	425° 39' 30"	426° 05'
- 360° 00'	- 360° 00' 00"	- 360° 00'
65° 19'	65° 39' 30"	66° 05'

2ND

248° 18' 30"	248° 46'	249° 04'
+ 180° 00' 00"	+ 180° 00'	+ 180° 00'
428° 18' 30"	428° 46'	429° 04'
- 360° 00' 00"	- 360° 00'	- 360° 00'
68 18' 30"	68° 46'	69 04

3RD

250° 47'	251° 01'	251° ~~41'~~
+180° 00'	+180° 00'	180° 00'
430° 47'	431° 01'	431° 41'
-360° 00'	- 360° 00'	- 360° 00'
70° 47'	71° 01'	~~71° 41'~~
		71° 24'

SURVEY DATA - OFFICE COMPUTATIONS

SURVEY NO. _____

SHEET 1 OF 1

BRG. OF INITIAL BACKSIGHT FROM:
( ) PRECEDING PAGE. THIS SURVEY
( ) SURVEY NO. _____. SHEET _____

N 17°33'W

COÖRDINATES AND ELEV. OF INITIAL INST. STA. FROM:
( ) PRECEDING PAGE. THIS SURVEY
( ) SURVEY NO. _____. SHEET

1,770.11 N | 1,072.70 E

COMP. BY	COSINE VERT. ANGLE	HORIZ. DISTANCE	SINE VERT. ANGLE	VERT. DIST.	BEARING	COSINE BEARING	SINE BEARING	LATITUDE OF FS		DEPARTURE OF FS		ELEVATION OF FS
								+		+		+
								− 166	75	−		−
		167.90			S 6°43'10"E			1603	36	1092	35	
								+		+		+
								− 70	28	− 6	38	−
		70.57			S 5°11'10"W			1533	08	1085	97	
								+		+ 107	24	+
								− 162	71	−		−
		194.87			S 33°23'20"E			1370	37	1193	21	
								+		+		+
								− 184	49	− 44	18	−
		189.71			S 13°28'00"W			1185	88	1149	03	
								+		+ 18.39		+
					50"			− 140	68	−		−
		141.88			S 7°26'30" E			1045	20	1167	42	
								+		+		+
								− 128	92	− 21	78	−
		130.75			S 9°35'20"W			916	28	1145	64	
								+		+		+
								−		−		−
					BS =			+		+ 133	38	+
					N9°35'20"E			− 15	24	−		−
		134.25			S83°29'00"E			901	04	1279	02	
								+		+		+
								−		−		−
								+		+		+
								−		−		−
								+		+		+
								−		−		−
								+		+		+
								−		−		−
								+		+		+
								−		−		−
								+		+		+
								−		−		−
								+		+		+
								−		−		−
								+ 2000	00	+ 1000	00	+
					FROM SUNSHOTS			− 229	89	+ 72	70	−
		241.11			S17°33'E			1770	11	1072	70	

SURVEY DATA - OFFICE COMPUTATIONS

SURVEY NO. * SURFACE

SHEET 1 OF

BRG. OF INITIAL BACKSIGHT FROM:
( ) PRECEDING PAGE, THIS SURVEY
(X) SURVEY NO. 1, SHEET ~~2~~ 1

N 13°21'E

COORDINATES AND ELEV. OF INITIAL INST. STA. FROM:
( ) PRECEDING PAGE, THIS SURVEY
(X) SURVEY NO. 1, SHEET 1

1922.02 | 981.49

COMP. BY	COSINE VERT. ANGLE	HORIZ. DISTANCE	SINE VERT. ANGLE	VERT. DIST.	BEARING	COSINE BEARING	SINE BEARING	LATITUDE OF FS	DEPARTURE OF FS	ELEVATION OF FS
								+ 130 06	+ 81 35	+
								—	—	—
		153.41		+51.85	N 32°01'30"E			2052 08	1062 84	
								+	+	+
								—	—	—
								+ 41 77	+ 42 04	+
								—	—	—
		59.29		-1.03	N 45°11'15"E			2093 85	1104 88	
								+	+ 57 87	+
								— 6 43	—	—
		58.23		-4.26	S 83°39'45"E			2045 65	1120 71	
								+	+ 75 39	+
								— 4 45	—	—
		75.51		+15.60	S 86°37'15"E			2049 63	1183 23	
								+	+	+
								—	—	—
								+ 160 58	+ 111 68	+
								—	—	—
		195.60		+68.29	N 34°49'E			2208 21	1294 91	
								+	+ 41 35	+
								— 26 72	—	—
		49.23		-10.45	S 57°07'45"E			2020 91	1224 58	
								+	+	+
								—	—	—
								+ 68 58	+ 2 29	+
								—	—	—
		68.62		-8.81	N 01°54'45"E			2276 79	1297 20	
								+	+	+
								—	—	—
								+	+	+
								—	—	—
								+	+	+
								—	—	—
								+	+	+
								—	—	—
								+	+	+
								—	—	—
								+	+	+
								—	—	—
								+	+	+
								—	—	—

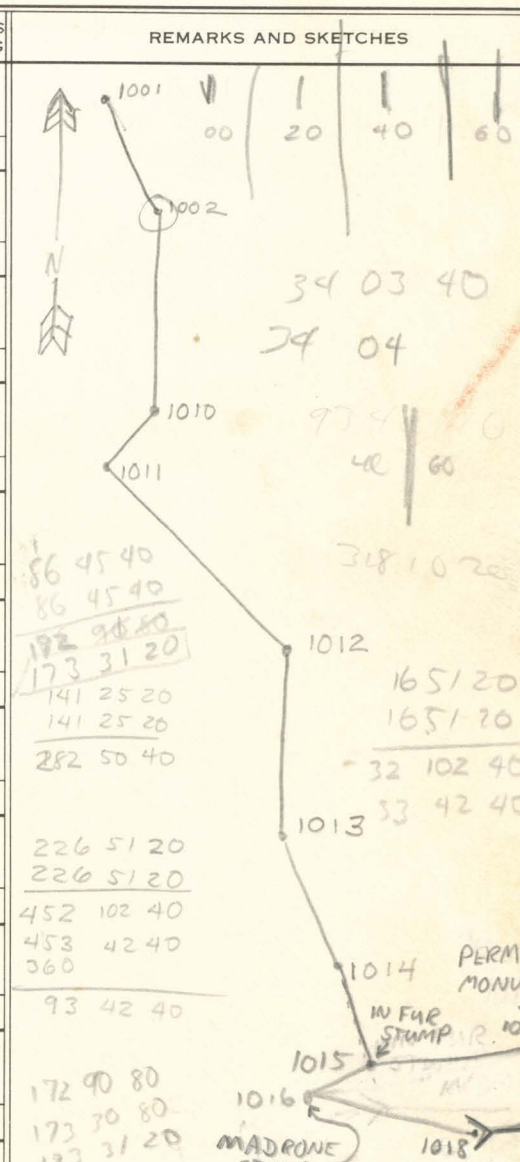
SURVEY DATA - FIELD SHEET

SURVEY NO. _____ SHEET 1 OF 1

MINE Queen of Bronze LEVEL Surface WORKING(S) Traverse from north workings to south workings

DATE	ENG. & HELPER	BS	INST	FS	HORIZONTAL ANGLE	VERTICAL ANGLE	SLOPE DISTANCE	HI	HP	PT. TO RAIL	COMPASS BEARING
11/11/80	JWW R(GA)E	1001			190 49 40	-03°38'30"					
			1002		21 39 40	-03°38'30		+4.62			
				1010	(190°49'50")	(-03°38'30")	168.24		-0-	—	—
"	"	1002			191°54'20"	-12°45'30"					
			1010		23°48'40"	-12°45'00"		+4.88			
				1011	(191°54'20")	(-12°45'15")	72.36		-0-	—	—
"	"	1010			141°25'20"	-14°04'30"					
			1011		282°51'00"	-14°05'00"		+4.61			
				1012	(141°25'30")	(-14°04'45")	200.91		-0-	—	—
"	"	1011			226°51'20"	-15°13'30					
			1012		93°42'40"	-15°13'30		+4.49			
				1013	(226°51'20")	(-15°13'30)	196.61		-0-	—	—
"	"	1012			159°05'20"	-04°43'30					
			1013		318°10'20"	-04°43'30		+4.73			
				1014	(159°05'10")	(-04°43'30")	142.36		-0-	—	—
4/23/81	"	1013			197°02'00"	03°30'00"					
			1014		34°04'20"	03°30'30"	130.99	+4.06			
				1015	(197°02'10")	(-03°30'15")			-0-	—	—
"	"	1014			268°25'20"	-22°50'00"					
			1015		176°50'40"	-22°50'00"		+3.83			
				1016	(268°25'20")	(-22°50'0")	27.04		-0-	—	—
"	"	1014			86°55'40"	-20°32'30					
			1015		173°51'20"	-20°32'30		+3.83			
		(monument)		1017	(86°55'40")	(-20°32'30)	149.37		-0-	—	—
"	"	1015			16°51'20"	0°0'0"					
			1016		33°42'40"	0°0'0"		+4.64			
				1018	(16°51'20")	(-0°0'0"	73.65		-0.16	—	—
					( )	( )					
					( )	( )					
					( )	( )					
					( )	( )					
					( )	( )					
					(VERT ∠)	( )					
8/27/79	JWW Rick P.	☼			+8°07'30"						
			1001		+8°07'30"			+5.01			
				1002	+8°07'30"	( )	243.55		+2.63		

REMARKS AND SKETCHES



SURVEY DATA - FIELD SHEET

SURVEY NO. _____ SHEET _____ OF _____

MINE _____ LEVEL _____ WORKING (S) _____

DATE	ENG. & HELPER	BS	INST	FS	HORIZONTAL ANGLE	VERTICAL ANGLE	SLOPE DISTANCE	HI	HP	PT. TO RAIL	COMPASS BEARING	REMARKS AND SKETCHES
8/27/79	JWW RAP	C-1			201°26'30"	-13°43'30"						
			C-0		42°33'30"	-13°43'		+4.72				
				T-1	(201°26'45")	(-13°43'15")	39.40		-0-	-0-		
8/28/79	JWW RAP	C-0			162°38'	-22°44'30"						162 38 162 38 324 76 325 16
			T-1		325°15'30"	-22°44'30"		+4.49				
				T-2	(162°37'45")	(-22°44'30")	214.62		-0-	-0-		
"	"	T-1			261°00'30"	-9°30'						261 00 30 261 00 30 522 01 60 360 01 162 01
			T-2		162°30'30"	-9°30'		+3.63				
				T-3	(261°00'15")	(-9°30')	185.98		-0-	-0-		
					( )	( )						
					( )	( )						
					( )	( )						
					( )	( )						
					( )	( )						
					( )	( )						
					( )	( )						
					( )	( )						
					( )	( )						
					( )	( )						
					( )	( )						
					( )	( )						
					( )	( )						
					( )	( )						

SURVEY DATA - FIELD SHEET

SURVEY NO. SURFACE — SHEET 1 OF _____

MINE QUEEN OF BRONZE — LEVEL SURFACE — WORKING(S) 'A', 'B', 'C' PORTALS; U.S.B.M. DDH-1, 2

DATE	ENG. & HELPER	BS	INST	FS	HORIZONTAL ANGLE	VERTICAL ANGLE	SLOPE DISTANCE	HI	HP	PT. TO RAIL	COMPASS BEARING
		1001			36°30'30"	+18°40'30"					
			C-0		73 01	+18°40'30"		+4.54			
				S-1	(36°30'30")	(+18°40'30")	161.94		-0-	-0-	
					( )	( )					
		C-0			193°09'30"	-0°59'30"					
			S-1		26°19'30"	-0°59'30"		+4.67			
				A-1	(193°09'45")	(-0°59'30")	59.28		-0.18	-0-	
		C-0			244° 19'	-4° 11'					
			S-1		128°37'30"	-4° 11'		+4.67			
				B-1	(244°18'45")	(-4° 11')	58.39		-0-	-0-	
		C-0			241° 21'	+11°40'30"					
			S-1		122°42'30"	+11°40'30"		+4.61			
				S-2	(241°21'15")	(+11°40'30")	77.11		-0-	-0-	
					( )	( )					
		S-1			121° 26'	+19°14'30"					
			S-2		242 52 30	+9° 15'		+4.43			
				S-3	(121°26'15")	(+19°14'45")	207.18		-0-	-0-	
		S-1			209°29'30"	-11°59'					
			S-2		58°59'	-11°59'		+4.43			
				DDH-2	(209°29'30")	(-11°59')	50.33		-0-	-0-	
					( )	( )					
		S-2			147°15'30"	-7°19'					
			S-3		294 31 30	-7°19'		+4.59			
				DDH-1	(147°15'45")	(-7°19')	69.18		-0-	-0-	
8/27	JW RAP	C-1			201°26'30"	-13°43'30"					
			C-0		42°33'30"	-13°43'		+4.72			
				T-1	(201°26'45")	(-13°43'15")	39.40		-0-	-0-	
					( )	( )					
					( )	( )					
					( )	( )					
					( )	( )					
		☺				+08°07'30"					
			1001			+08°07'30"		+5.01			
				1002	( )	(-08°07'30")	243.55		+2.63	-0-	

REMARKS AND SKETCHES

C-0 Steel fence post on 'c' level dump
S-1 Steel pin on 'A' level dump
A-1 Spad in 'A' level portal
B-1 Wood stake in front of hole to 'B' level (out 11/80)
S-2 Steel pin on dump at southwest end of glory hole
S-3 Steel pin along road above northeast end of glory hole
DDH-2 Wood stake at approximate location of hole
DDH-1 ditto

SURVEY DATA - FIELD SHEET

SURVEY NO. _____

MINE QUEEN OF BRONZE LEVEL A WORKING(S) East end above Cameron stope

SHEET 1 OF 2

DATE	ENG. & HELPER	BS	INST	FS	HORIZONTAL ANGLE	VERTICAL ANGLE	SLOPE DISTANCE	HI	HP	PT. TO RAIL	COMPASS BEARING	REMARKS AND SKETCHES
8/9/79	JWW RAP	M-12			138°29'	0°0'						
			A-16		276°58'30"	0°0'		+1.08				
				A-15	(138°29'15")	(0°0')	12.38		-1.02	-5.21	N 9° W	
		A-15			140°10'30"	0°0'						
			A-16		280°21'30"	0°0'		+1.08				
				A-17	(140°10'45")	(0°0')	38.89		-2.82	-6.10	S 47° E	
		A-16			130°04'30"	0°0'				NOT AVAILABLE FOR COMMENT		
			A-15		260°19'	0°0'		+3.18				
				A-14	(130°04'45")	(0°0')	26.15		-1.16		N 59° W	
		A-16			284°37'30"	-2°28'						
			A-17		20°15'30"	-2°28'		+1.08				
				A-1701	(284°37'45")	(-2°28')	14.30		-0.41	4.78	S 65° W	TO WOODEN SILL TIED TO LADDER
		A-16			147°38'30"	0°0'						
			A-17		295°17'30"	0°0'		+1.08				
				A-18	(147°38'45")	(0°0')	10.45		-1.84	6.42	S 79° E	
		A-17			147°56'	0°0'						
			A-18		295°52'	0°0'		+1.48				
				A-1811	(147°56')	(0°0')	23.58		0.74	N/A	N 72° W	
		A-17			229°52'30"	0°0'						
			A-18		78°22'30"	0°0'		+1.48				
				A-19	(229°52'45")	(0°0')	21.89		-1.97	6.45	S 35° E	
8/10/79	JWW RAP	A-18			173°14'30"	0°0'						
			A-19		346°29'30"	0°0'		+1.62				
				A-20	(173°14'45")	(0°0')	19.52		-1.50	N/A	S 90° E	
		A-18			96°04'30"	0°0'						
			A-19		192°8'30"	0°0'		+1.62				
				A-1911	(96°04'45")	(0°0')	19.83		-0.64	N/A	N 68° W	
		A-18			57°53'30"	0°0'						
			A-19		115°47'	0°0'		+1.62				
				A-1912	(57°53'30")	(0°0')	22.54		-2.35	N/A	N 34° E	
					( )	( )						
					( )	( )						
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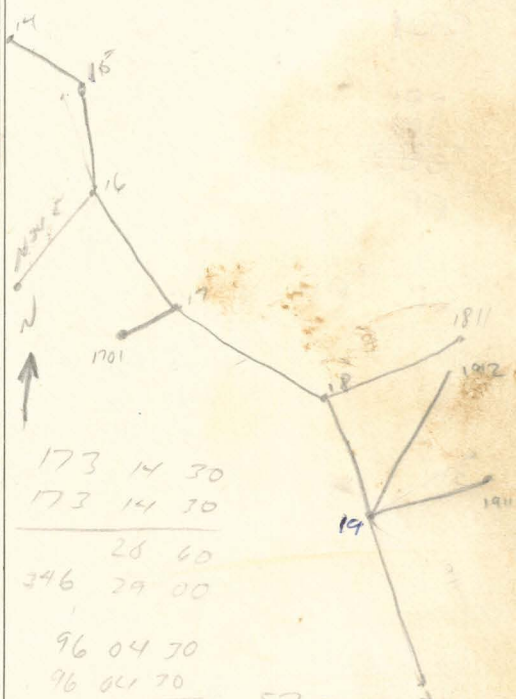


173 14 30
173 14 30
28 60
346 29 00
96 04 30
96 04 30
192 08 60
192 09 00
57 53 30
57 53 30
114 106 60
57 52 30
57 52 30
115 47
114 107 60
(115 45 00)

SURVEY DATA - OFFICE COMPUTATIONS

SURVEY NO. _____

SHEET _____ OF _____

BRG. OF INITIAL BACKSIGHT FROM:
( ) PRECEDING PAGE, THIS SURVEY
( ) SURVEY NO. _____, SHEET _____
S 32°01'30" W

COÖRDINATES AND ELEV. OF INITIAL INST. STA. FROM:
( ) PRECEDING PAGE, THIS SURVEY
( ) SURVEY NO. _____, SHEET _____

2052.08	1062.84	

COMP. BY	COSINE VERT. ANGLE	HORIZ. DISTANCE	SINE VERT. ANGLE	VERT. DIST.	BEARING	COSINE BEARING	SINE BEARING	LATITUDE OF FS	DEPARTURE OF FS	ELEVATION OF FS
								+ 41 77	+ 42 05	+
								—	—	—
		59.27		-1.03	N 45°11'15" E			2093 85	1104 89	
								+ 15 94	+ 22 84	+
								—	—	—
		27.85		-0-	N 55°05' E			2109 79	1127 73	
								+ 48 78	+ 62 09	+
								—	—	—
		78.96		-0-	N 51°50'45" E			2142 63	1166 98	
								+ 14	+ 30 42	+
								— 14 06	—	—
		33.51		-0-	S 65°11'15" E			2095 73	1158 15	
								+	+ 6 60	+
								— 4 78	—	—
		08.15		-0-	S 54°07'22" E			2137 85	1173 58	
								+ 10 03	+	+
								—	— 1 99	—
		10.23		-0-	N 11°13'22" W			2152 66	1164 99	
								+ 49 22	+	+
								—	— 19 51	—
		52.95		-0-	N 21°37'22" W			2191 85	1147 47	
								+ 15 54	+ 9 26	+
								—	—	—
		18.09		-0-	N 30°47'08" E			2168 20	1174 25	
								+	+	+
								—	—	—
								+	+	+
								—	—	—
								+	+	+
								—	—	—
								+	+	+
								—	—	—
								+	+	+
								—	—	—
								+	+	+
								—	—	—
								+	+	+
								—	—	—
								+	+	+
								—	—	—

SURVEY DATA - FIELD SHEET

SURVEY NO. _____ SHEET 2 OF 2

MINE Queen of Bronze LEVEL A WORKING(S) West end

DATE	ENG. & HELPER	BS	INST	FS	HORIZONTAL ANGLE	VERTICAL ANGLE	SLOPE DISTANCE	HI	HP	PT. TO RAIL	COMPASS BEARING
8/79	JWW RAP	C-0			193°09'30"	-59'30"					
			S-1		20°19'30"	-59'30"		+4.67			
				A-1	193°09'45")	(-59'30")	59.28		-0.18	-0-	
1/22/81	JWW RGE	S-1			189°53'30	0° 0'					
			A-1		19°47'30	0° 0'		-0.92			
				A-2	(189°53'45")	(0° 0')	27.85		-4.33	-0-	
	S-1 →	A-1			186°40'00	0° 0'					
			A-2		13°20'00	0° 0'		-0.92			
	A-1 →			A-3	(186°39'30")	(0° 0')	78.96		-3.74	-0-	
1/28/81	JWW RGE	A-1			239°43'30"	0° 0'					
			A-2		119°27'30"	0° 0'		-1.93			
				A-201	(239°43'45")	(0° 0')	33.51		-1.33	-0-	
	SEE ATTACHED NOTES	A-2			255°47'30"	0° 0'					
			A-3		151°35'00"	0° 0'		-2.13			
				A-202	(255°47'30")	(0° 0')	08.15		-3.13	-0-	
		A-2			118°41'30"	0° 0'					
			A-3		237°23'00"	0° 0'		-2.13			
				A-4	(118°41'30")	(0° 0')	10.23		-1.13	-0-	
		A-2			108°17'30"	0° 0'					
			A-3		216°35'00"	0° 0'		-2.13	2.292		
				A-5	(108°17'30")	(0° 0')	52.95		2.29	-0-	
		A-3			228°01'00"	0° 0'					
			A-4		96°01'00"	0° 0'		-2.10			
				A-203	(228°00'30")	(0° 0')	18.09		-1.42	-0-	
						( ) ( )					
						( ) ( )					
						( ) ( )					
						( ) ( )					
						( ) ( )					
						( ) ( )					
						( ) ( )					
						( ) ( )					

REMARKS AND SKETCHES

C-0 Steel fence post on 'C' level dump (no longer reliable, 10/80)

S-1 Steel pin on 'A' level dump

A-1 Spad in 'A' level portal

SEE SKETCH ON BACK FOR NOTES & BRG. A-2 — A-3

108 17 30
108 17 30
216 34 60
216 35

105 17 30
105 17 30
210 34 60
210 35 00

118 41 30
118 41 30
236 82 60
236 83 00
237 23 00

239 43 30
239 43 30
478 86 60
478 87 00
479 27 00
-360 00 00
119 27 00

255 47 30
255 47 30
510 94 60
510 95 00
511 35 00
-360 00 00
151 35 00

228 01 00
228 01 00
456 02 00
456 02 60

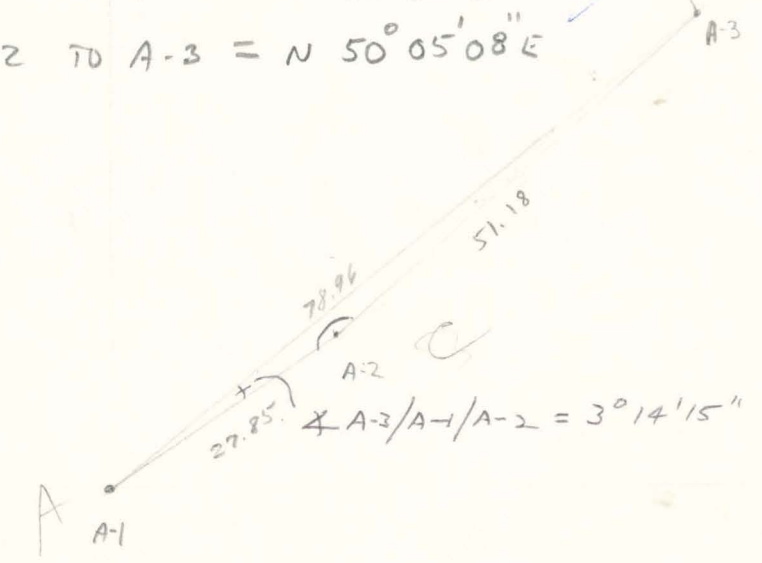
3
456 02 00
-360 00 00
96 02 00

$$\begin{array}{r} \angle A\ 3\ 14\ 15 \\ +\angle B\ 1\ 45\ 37 \\ \hline 4\ 59\ 52 \end{array} \qquad \begin{array}{r} 179\ 59\ 60 \\ -\ 4\ 59\ 52 \\ \hline 175^\circ 00' 08'' \end{array}$$

$\angle C$

BRG A-1 TO A-2 = N 55°05'E

BRG A-2 TO A-3 = N 50°05'08"E



SURVEY DATA - FIELD SHEET

SURVEY NO. 2 SHEET 7 OF

MINE Q of B LEVEL E WORKING(S) 2343, 2350, 2359, 2370

Inst # 28169 "Buff"

DATE	ENG. & HELPER	BS	INST	FS	HORIZONTAL ANGLE	VERTICAL ANGLE	SLOPE DISTANCE	HI	HP	PT. TO RAIL	COMPASS BEARING	REMARKS AND SKETCHES
4/77	TJF JWW	E-61A			340°27'	+33°40						0° Declination Elev. Rail
			E-61A1		320°54'	+33°41'		-0.79				
				150-2	(340°27')	(33°40.5')	14.49		-0-	-5.1	N34°W	N53°W 2344.12
	"	E-61A1			121°07'							
			150-2		242°13'			-0.77				
				150-3	(121°06.5')	(-0-)	18.15		-1.08	-6.1	S87W	S68W 2343.43
	"	E-61A1			312°24'							
			150-2		264°47'			-0.77				
				150-1	(312°23.5')	(-0-)	22.58		-0.13	-	S81E	N79.5E
	"	2343-A			115°33	+23°59'						
			150-3		231°05'	+24°00'		-1.57				
				150-4	(115°32.5')	(23°59.5')	10.68		-2.92	-5.2	S23W	S 3½° W 2350.02
	"	150-3			152°38'							
			150-4		305°15'			-0.87				
				150-5	(152°37.5')	(-0-)	38.42		-1.62	-	S4°E	S 23.5 E
5/77	TJF JWW	150-4			107°44'							
			150-5		215°28'			-1.42				
				150-6	(107°44')	(-0-)	16.61		-1.36	-6.0	S76°E	N84° E
		150-5			136°39'	+28°44'						Distance to 180° Spad
			150-6		273°17'	+28°45'		-1.03				2350E → 2350F = 100
				150-7	(136°38.5')	(28°44.5')	18.51		-0-	5.0	N60.5°E	N41° E BS INST FS
		150-6			172°36'	+4°01'						
			150-7		345°12'	+4°00'		-1.31				
				150-8	(172°36')	(4°01.5')	27.62		-3.56	-	N53°E	N38° E
		150-7			79°26'	+26°30'						
			150-8		158°52	+26°30'		-2.29				
				A-10	(79°26')	(26°30')	18.95		-0-	-	N49°W	N67.5°W
					( )	( )						
					( )	( )						
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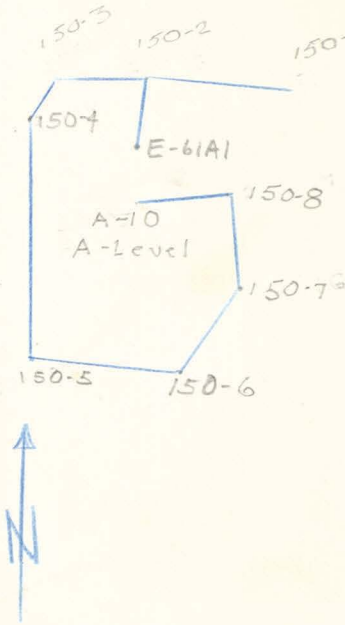

SURVEY DATA - OFFICE COMPUTATIONS

SURVEY NO. 2

SHEET 7 OF

BRG. OF INITIAL BACKSIGHT FROM:
( ) PRECEDING PAGE, THIS SURVEY
( ) SURVEY NO. 2, SHEET 6

N12°02'W

COÖRDINATES AND ELEV. OF INITIAL INST. STA. FROM:
( ) PRECEDING PAGE, THIS SURVEY
( ) SURVEY NO. 2, SHEET 6

1029.51	5333.25	2341.98

MP.	COSINE VERT. ANGLE	HORIZ. DISTANCE	SINE VERT. ANGLE	VERT. DIST.	BEARING	COSINE BEARING	SINE BEARING	LATITUDE OF FS	DEPARTURE OF FS	ELEVATION OF FS
								+ 10 27	+	+ 7 2
								—	— 6 32	—
		12.06		+8.03	N31°35'W			1039 78	5326 93	2349 2
								+	+	+ 3
								— 15	— 18 15	—
		18.15		-0-	S89°31'30"W			1039 63	5308 78	2349 5
								+	+ 22 18	+
						SHOULD BE →		— 4 23	—	— 6
		22.58		-0-	S 79°11'30"E	IN RED		1035 55	5349 11	2348 5
								+	+	+ 5 6
								— 8 84	— 4 14	—
		9.76		+4.34	S25°04'W			1030 79	5304 64	2355 2
								+	+ 1 55	+ 7
								— 38 39	—	—
		38.42		-0-	S 2°18'30"E			992 40	5306 19	2355 9
								+	+ 16 01	+
								— 4 42	—	— 0
		16.61		-0-	S74°34'30"E			987 98	5322 20	2355 9
								+ 7 60	+ 14 34	+ 7 8
								—	—	—
		16.23		+8.90	N 62°04' E			995 58	5336 54	2363 7
								+ 15 93	+ 22 48	+ 4 1
								—	—	—
		27.55		+1.94	N 54°40' E			1011 51	5359 02	2367 9
								+ 7 10	+	+ 6 1
								—	— 15 40	—
		16.96		8.46	N65°14 W			1018 61	5343 62	2374 1
								+	+	+
								—	—	—
								+	+	+
								—	—	—
								+	+	+
								—	—	—
								+	+	+
								—	—	—
								+	+	+
								—	—	—
								+	+	+
								—	—	—
								+	+	+
								—	—	—
								+	+	+
								—	—	—

	Lat.	Dep.
Origin (Sta. 1001)	2000 N	1000 E
Sta. E-64B	2162.90	1303.18

$+ \frac{162.90}{20} + \frac{303.18}{20}$

$= 8.145$ inches

$(+ 2.75 \text{ in})$ (to allow for 'A' Level)

10.895 inches

Minimum $11\frac{1}{2}$ inches from top border to Sta. 1001

	Lat.	Dep.
Sta. C-19	1945.51	1585.11

($30\frac{1}{2}$" would be better)

$\frac{585.11}{20} = 29.2$

Minimum 30" from right border to Sta. 1001

SURVEY DATA - FIELD SHEET

SURVEY NO. 1 — 'C' LEVEL – 'A' LEVEL VIA McCAULEY STOPE — SHEET 8 OF _____

MINE QUEEN of BRONZE — LEVEL _____ — WORKING (S) _____

DATE	ENG. & HELPER	BS	INST	FS	HORIZONTAL ANGLE	VERTICAL ANGLE	SLOPE DISTANCE	HI	HP	PT. TO RAIL	COMPASS BEARING	REMARKS AND SKETCHES
		C-13			~~156° 20'~~	–		~~-5.15~~				
			C-1311		~~302° 40~~	–	~~18.13~~	~~5.13~~	~~3.66~~			
				M-10	( )	(–)					~~N27E~~	
3/7/79	JWW RAP	C-13			151 18 30							
			C-1311		302 38		18.13	-5.13	~~3.66~~		~~N~~	
				M-10	(151° 19')	( )			-3.66		N27E	
					( )	( )						
3/13/79	JWW RAP	L-12			~~42°34'30"~~	0° 0'						
			L-13		5° 9'	0° 0'		-1.07				
				L-14	(~~42°34'30"~~)	(0° 0')	34.28		-2.29	N/A	S02°E	
					( )	( )						
					( )	( )						
					( )	( )						
					( )	( )						
					( )	( )						
					( )	( )						
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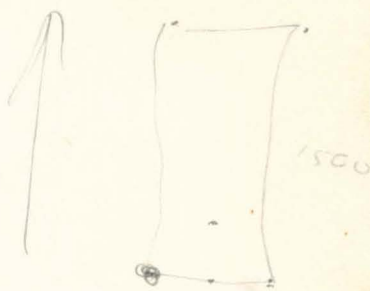
182 34 30
182 34 30
364 68 60
360 00 00
4 ← 68 ← 60
5 9



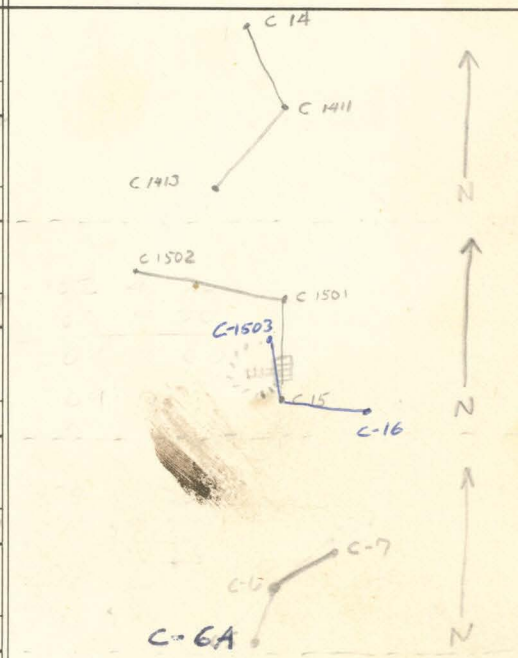
SURVEY DATA - FIELD SHEET

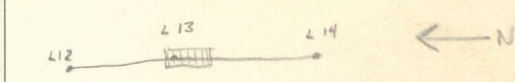
SURVEY NO. _____ SHEET _____ OF _____

MINE _____ LEVEL 'C' WORKING (S) MISCELLANEOUS

DATE	ENG. & HELPER	BS	INST	FS	HORIZONTAL ANGLE	VERTICAL ANGLE	SLOPE DISTANCE	HI	HP	PT. TO RAIL	COMPASS BEARING
8/8/79	JWW RAP	C-14	1411		229° 43'	-7° 35'					
			~~C-1401~~	1413	99° 26' 30"	-7° 35'		-1.74			
				~~C-1403~~	(229° 43' 15")	(-7° 35')	11.32		-0.54	N/A	S 50 W
					( )	( )					
8/13/79	JWW RAP	C-15			102° 4' 30"	-2° 37'					
			C-1501		204° 8'	-2° 37'		-1.89			
				C-1502	(102° 4' ~~30"~~ 00")	(-2° 37')	30.35		-3.11	N/A	N 85° W
					( )	( )					
8/14/79	JWW RAP	C-7			134° 50'	0° 0'					
			C-6		269 41	0° 0'		-1.15			
				C-6A	~~(134° 49' 30")~~	(0° 0')	10.56		-1.88	NA	S 25° W
					134° 50' 30"						
					( )	( )					
					( )	( )					
8/13/79	JWW RAP	L-12			182° 34' 30"	0° 0'					
			L-13		005° 09'	0° 0'		-1.07			
				L-14	(182° 34' 30")	(0° 0')	34.28		-2.29	N/A	S 02° E
					( )	( )					
					( )	( )					
		~~C-16~~			~~253° 39'~~	~~0° 0'~~					
			~~C-15~~		~~147° 18' 30"~~	~~0° 0'~~		~~-1.78~~			
				~~C-1503~~	~~(253° 39' 15")~~	~~(0° 0')~~	~~18.93~~		~~-1.49~~	~~6.85~~	~~N 02° E~~
		~~C-15~~			~~52° 56'~~	~~-47° 57' 30"~~					
			~~C-1503~~		~~105° 51' 30"~~	~~-47° 57'~~		~~-2.07~~			
				~~L-9~~	~~(52° 55' 45")~~	~~(-47° 57' 15")~~	~~25.56~~		~~-2.24~~	~~NA~~	~~S 51° W~~
		~~C-1503~~			~~206° 9'~~	~~-3° 10'~~					
			~~L-9~~		~~52° 18'~~	~~-3° 10'~~		~~-2.84~~			
				~~L-8~~	~~(206° 9')~~	~~(-3° 10')~~	~~31.44~~		~~-0-~~	~~NA~~	~~S 79° W~~
		C-16			253° 39'	0° 0'					
			C-15		147° 18' 30"	0° 0'		-1.78			
				C-1503	(253° 39' 15")	(0° 0')	18.93		-1.49	6.85	N 02° E
		C-15			52° 56'	-47° 57' 30"					
			C-1503		105° 51' 30"	-47° 57'		-2.07			
				L-9	(52° 55' 45")	(-47° 57' 15")	25.56		-2.24	NA	S 51° W
		C-1503			206° 9'	-3° 10'					
			L-9		52° 18'	-3° 10'		-2.84			
				L-8	(206° 9')	(-3° 10')	31.44		-0-	NA	S 79° W

REMARKS AND SKETCHES





206
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253 39
253 39
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360 00
146 78
147 18

52 56
52 56
104 112
105 52

253 39
253 39
506 78
507 18
360
147 18

52 56
52 56
104 112
105 52

206 9
206 9
412 18
360
52 18

SURVEY DATA - FIELD SHEET

SURVEY NO. 1 SHEET 1 OF

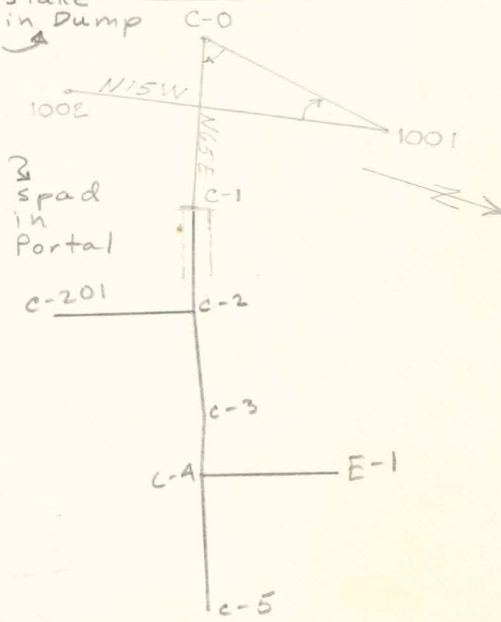
MINE Queen of Bronze LEVEL C WORKING (S)

INST 28169 "BUFF"

DATE	ENG. & HELPER	BS	INST	FS	HORIZONTAL ANGLE	VERTICAL ANGLE	SLOPE DISTANCE	HI	HP	PT. TO RAIL	COMPASS BEARING
3/11/77	TJF JWW	1002			30°54'	-10°14'					
			1001		61°48'	-10°16'		+4.29			
				C-0	(30°54')	(-10°15')	81.45		0.0		S 16° W
"	"	1001			49°59'						
			C-0		99°57'			+4.10			
				C-1	(49°58.5')	(-0-)	108.85		-2.39	-5.97	N 65° E
"	"	C-0			186°01'						
			C-1		12°03'			-1.45			
				C-2	(186°01.5')	(-0-)	53.18		-2.87	-6.54	N 70° E
3/14/77	TJF JWW	C-1			260°22'						
			C-2		160°43'			-1.75			
				C-201	(260°21.5')	(-0-)	48.82		-0.04	-4.90	S 29.5° E
"		C-1			181°00'						
			C-2		02°00'			-1.75			
				C-3	(181°00')	(-0-)	32.73		-2.18	-6.65	N 71° E
"		C-2			223°47'						
			C-3		87°34'			-2.03			
				C-4	(223°47')	(-0-)	16.17		-3.70	-8.15	S 65.5 E
"		C-3			178°21'						
			C-4		356°43'			-3.67			
				C-5	(178°21.5')	(-0-)	68.95		-1.20	-5.95	S 66 E
"		C-3			120°51'	+40°00'					
			C-4		241°41'	+40°00'		-3.67			
				E-1	(120°50.5')	(+40°00')	23.35		-1.21	-6.90	N 55° E
					( )	( )					
					( )	( )					
					( )	( )					
					( )	( )					
					( )	( )					
					( )	( )					
					( )	( )					
					( )	( )					

REMARKS AND SKETCHES

Steel Stake in Dump (SKETCH ON BACK)



— Point of Cave-In

{First shot from C-Level to E-Level

SURVEY DATA - OFFICE COMPUTATIONS

SURVEY NO. 1

SHEET 2 OF _____

BRG. OF INITIAL BACKSIGHT FROM:
( ) PRECEDING PAGE, THIS SURVEY
( ) SURVEY NO. 2, SHEET 2

N43°01'E

COORDINATES AND ELEV. OF INITIAL INST. STA. FROM
( ) PRECEDING PAGE, THIS SURVEY
( ) SURVEY NO. _____, SHEET

(C-10) | 2007.87 | 1361.53 |

COMP. BY	COSINE VERT. ANGLE	HORIZ. DISTANCE	SINE VERT. ANGLE	VERT. DIST.	BEARING	COSINE BEARING	SINE BEARING	LATITUDE OF FS	DEPARTURE OF FS	ELEVATION OF FS
								+	+	+
								− 5 70	− 29 83	−
		30.37		-0-	S 79°11'W		(C-9)	2002 17	1331 70	
					BS S79°11'W		(C-10)	+2007.87 7 84	+1361.53 39 32	+
								−	−	−
		40.09		-0-	N 78°43½'E		(C-11)	2015 71	1400 85	
					BS S79°11'W		(C-10)	+2007.87 5 75	+1361.53 9 06	+
								−	−	−
		10.73		-0-	N 57°36'E		(C-1011)	2013 62	1370 59	
					BS S57°36'W		↓	+ 7 98	+ 8 33	+
								−	−	−
		11.53	✓	-5.45	N46°13½'E		(C-1015)	2021 60	1378 92	
					BS S57°36'W		(C-1011)	+2013.62 11 78	+1370.59 2 38	+
								−	−	−
		12.02		-0-	N11°25'E		(C-1012)	2025 40	1372 97	
					BS S11°25'W		↓	+	+	+
								− 61	− 11 24	−
		11.26	✓	-1.27	S86°54½'W		(C-1013)	2024 79	1361 73	
					BS S11°25'W		(C-1012)	+2025.40 13 86	+1372.97 2 37	+
								−	−	−
		14.06	✓	-0.56	N09°41½'E		(C-1014)	2039 26	1375 34	
					BS S78°43½'W		(C-11)	+2015.71	+1400.85 6 68	+
								− 9 26	−	−
		11.42		-0-	S35°47½'E		↓	2006 45	1407 53	
								+	+ 59 56	+
								− 18 85	−	−
		62.47		-0-	S72°26'E		(C-13)	1987 60	1467 09	
								+	+ 2 90	+
								− 10 28	−	−
		10.68		-0-	S15°46½'E		(C-14)	1977 32	1469 99	
					BS N72°26'W		(C-13)	+1987.60 9 28	+1467.09 12 42	+
								−	−	−
		15.50	✓	+ 19.17	N53°13½'E		(C-1311)	1996 88	1479 51	
					BS N72°26'W		(C-13)	+1987.60 25 30	+1467.09	+
								−	− 17 23	−
		30.61	✓	- 1.72	N34°16'W			2012 90	1449 86	
					BS S46°13½'W		(C-1015)	+2021.60 1 62	+1378.92 12 63	+
								−	−	−
		12.73	✓	- 3.20	N82°41'E		↓	2023 22	1391 55	
								+	+ 3 49	+
								− 15 78	−	−
		16.16	✓	- 12.81	S12°29'E		(C-1017)	2007 44	1395 04	
								+ 9 58	+ 12 10	+
								−	−	−
		15.43	✓	+ 1.82	N51°37'E		(C-1018)	2017 02	1407 14	
					BS N12°29'W		(C-1017)	+2007.44	+1395.04 3 85	+
								− 17 09	−	−
		17.52	✓	- 24.19	S12°41½'E		(L-5)	1990 35	1398 89	

SURVEY DATA - OFFICE COMPUTATIONS

SURVEY NO. 1

SHEET 1 OF _____

C-Level

BRG. OF INITIAL BACKSIGHT FROM:
( ) PRECEDING PAGE, THIS SURVEY
( ) SURVEY NO. _____, SHEET _____
S 17°33'E True bearing from sunshots

COORDINATES AND ELEV. OF INITIAL INST. STA. FRO
( ) PRECEDING PAGE, THIS SURVEY
( ) SURVEY NO. _____, SHEET _____
2000 N | 1000 E

COMP. BY	COSINE VERT. ANGLE	HORIZ. DISTANCE	SINE VERT. ANGLE	VERT. DIST.	BEARING	COSINE BEARING	SINE BEARING	LATITUDE OF FS	DEPARTURE OF FS	ELEVATIO OF FS
								+	+	+
		80.15		-14.49 ✓				− 77 98	− 18 51	−
					S 13°21'W			1922 02	981 49	
								+ 48 87	+ 97 26	+
								−	−	−
		108.85		-0-	N 63°19½'E			1970 89	1078 75	
								+ 18 75	+ 49 76	+
								−	−	−
		53.18		-0-	N 69°21'E			1989 64	1128 51	
								+	+ 24 62	+
								− 42 15	−	−
		48.82		-0-	S 30°17½'E			1947 49	1153 13	
					BS S 69°21'W		(C-2)	+1989.64 11 01	+1128.51 30 82	+
								−	−	−
		32.73		-0-	N 70°21'E			2000 65	1159 33	
								+	+ 14 76	+
								− 6 61	−	−
		16.17		-0-	S 65°52'E			1994 04	1174 09	
								+	+ 63 71	+
								− 26 38	−	−
		68.95		-0-	S 67°30½'E			1967 66	1237 80	
					BS N 65°52'W		(C-4)	+1994.04 10 27	+1174.09 14 65	+
								−	−	−
		17.89 ✓		+15.01 ✓	N 54°58½'E		(E-1)	2004 31	1188 74	
								+	+	+
								−	−	−
								+	+	+
								−	−	−
								+	+	+
								−	−	−
								+	+	+
								−	−	−
								+	+	+
								−	−	−
								+	+	+
								−	−	−
								+	+	+
								−	−	−
								+	+	+
								−	−	−
								+	+	+
								−	−	−

SURVEY DATA - FIELD SHEET

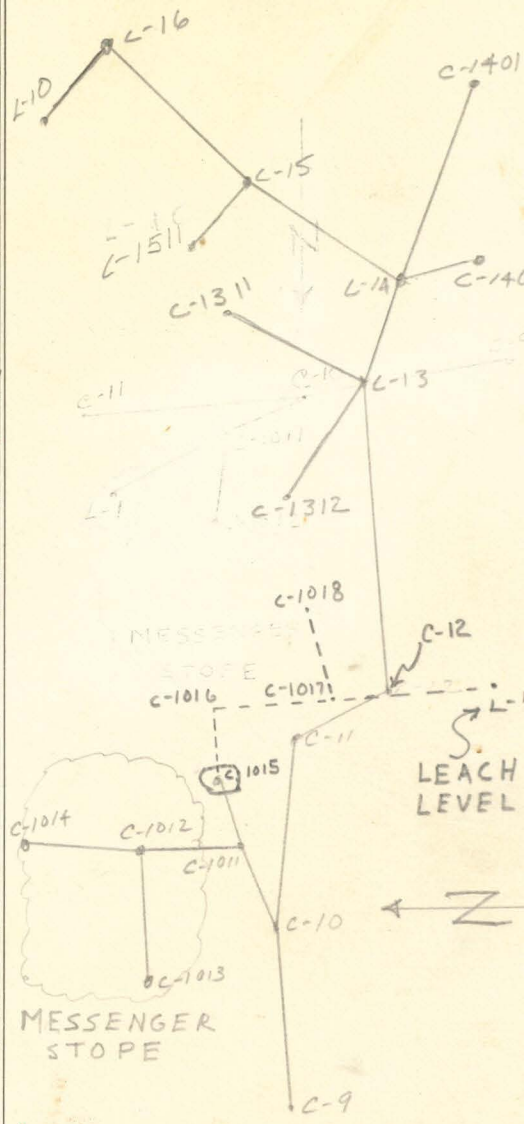
SURVEY NO. 1 SHEET 2 OF

MINE Queen of Bronze LEVEL C WORKING (S)

#28169 "Buff"

DATE	ENG. & HELPER	BS	INST	FS	HORIZONTAL ANGLE	VERTICAL ANGLE	SLOPE DISTANCE	HI	HP	PT. TO RAIL	COMPASS BEARING
6/3/77	TJF JWW	E-9			216°10'						0 Dec.
			C-10		72°20'			-1.50			S W
				C-9	(216°10')	(-0-)	30.37		-1.62	5.85	N60°E
	"	C-9			179°32						
			C-10		359°05'			-1.50			N E
				C-11	(179°32.5)	(-0-)	40.09		-1.65	6.10	S59°W
	"	C-9			158°25'						
			C-10		316°50'			-1.50			N E
				C-1011	(158°25')	(-0-)	10.73		-4.08	85	S38°W
	"	C-10			168°37'	-25°18'					
			C-1011		337°15'	-25°19'		-3.71			N E
				C-1015	(168°37.5')	(-25°18.5')	12.75		-0.30	—	S27°W
	"	C-10			135°49'						
			C-1011		267°38'			-3.71			N W
				C-1012	(133°49')	(-0-)	12.02		-5.56	—	S7°E
6/6/77	TJF JWW	C-1011			75°30'	-6°25'					
			C-1012		150°59'	-6°25'		-2.55			S W
				C-1013	(75°29.5')	(-6°25')	11.33		-0.97		N69°E
		C-1011			178°16'	2°16'					
			C-1012		356°33'	2°17'		-2.55			N W
				C-1014	(178°16.5')	(2°16.5')	14.07		-0.21		S 9°E
		C-16			245°29'						
			C-11		130°58'			-1.64			S E
				C-12	(245°29')	(-0-)	11.42		-1.21	6.3	N55°W
		C-11			143°21'						
			C-12		286°43'			-1.28			S E
				C-13	(143°21.5')	(-0-)	62.47		-4.39	—	S88°W
6/7/77	TJF JWW	C-12			236°40'						
			C-13		113°19'			-4.01			S E
				C-14	(236°39.5')	(-0-)	10.68		-0.99	6.3	N35°W
		C-12			125°40'	+51°02'					
			C-13	(RS)	251°19'	+51°02'		-4.01			N E
				C-1311	(125°39.5')	(+51°02')	24.65		-5.07	—	S33.5W
		C-12			38°10'	-3°13'					
			C-13		76°20'	-3°13'		-4.01			N W
				C-1312	(38°10')	(-3°13')	30.66		-1.62	—	S59°E
7/13/77	TJF JWW	C-1011			216°27'	-14°07'					
			C-1015		72°55'	-14°07'		-0.89			OK
				C-1016	(216°27.5')	(-14°07')	13.13		-1.14	—	N84°E
		C-1015			264°50'	-38°24'					
			C-1016		169°40'	-38°24'		-2.23			
				C-1017	(264°50')	(-38°24')	20.62		-0-	-0-	S12°E
		C-1016			64°06'	+6°44'					
			C-1017		128°12'	+6°44'		+2.19			
				C-1018	(64°06')	(+6°44')	15.54		-0.86	—	N53°E
		C-1016			179°47'	-54°05'					
			C-1017		359°35'	-54°06'		+2.19			
				C-5	(179°47.5')	(-54°05.5')	29.87		-4.69	—	S11.5°E

REMARKS AND SKETCHES



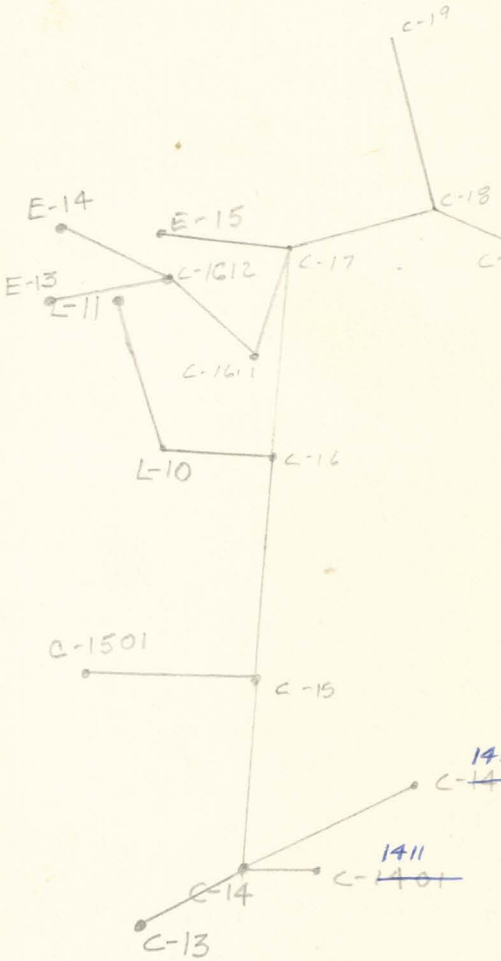
* Note ⊕ HI of Instrument

SURVEY DATA - FIELD SHEET

SURVEY NO. 1 SHEET 3 OF

MINE Queen of Bronze LEVEL C WORKING (S)

DATE	ENG. & HELPER	BS	INST	FS	HORIZONTAL ANGLE	VERTICAL ANGLE	SLOPE DISTANCE	HI	HP	PT. TO RAIL	COMPASS BEARING	REMARKS AND SKETCHES
7/5	TJF JWW	C-13			110°34'							
			C-14		221°07'			-1.64				
				C-15	(110°33.5')	(-0-)	23.94		-2.43	7.5	S86.5E	
		C-13			194°46'							
			C-14	1411	29°33'			-1.64				
				C-~~1401~~	(194°46.5')	(-0-)	8.63		-1.99	6.4	S1°E	
		C-13			164°46'	-1°09'						
			C-14	1412	329°33'	-1°08'		-1.64				
				C-~~1402~~	(164°46.5')	(-1°08.5')	45.75		-1.09	5.9	S31.5E	
		C-14			191°10'							
			C-15		22°19'			-2.08				
				C-16	(191°09.5')	(-0-)	13.64		-1.99	6.95	S73.5E	
		C-14			78°31'							
			C-15		157°02'			-2.08				
				C-1501	(78°31')	(-0-)	46.88		-1.91	-	N6°W	
		C-15			179°01'							
			C-16		358°02'			-2.09				
				C-17	(179°01')	(-0-)	40.72		-4.73	8.06	S79°E	
		C-15			80°21'	-57°48'						
			C-16		160°43'	-57°49'		-2.52				
				L-10	(80°21.5')	(-57°48.5')	26.17		-4.36	-	N7°E	
7/15/71	TJF JWW	C-16			233°02'							
			C-17		106°05'			-3.52				
				C-18	(233°02.5')	(-0-)	23.58		-2.16	-	S21.5E	
		C-16			86°24'	+40°25'						
			C-17		172°47'	+40°26'		-3.52				
				E-15	(86°23.5')	(+40°25.5')	33.68		-5.52	-	N10°E	
		C-16			6°06'							
			C-17		12°12'			-3.52				
				C-1611	(6°06')	(-0-)	17.16		-2.62	8.0	N69°W	
	*	C-16			122°13'							
			C-1611		244°25'			-3.41				
				C-1612	(122°12.5')	(-0-)	24.21		-3.20	-	N42.5E	
		C-17			100°06'							
			C-18		200°13'			-1.39				
				C-19	(100°06.5')	(-0-)	30.63		-2.18	-	N78°E	
		C-17			235°55'							
			C-18		111°49'			-1.39				
				C-20	(235°54.5')	(-0-)	8.87		-0.43	-	S34.5W	
		C-16			252°13'	-19°53'						
			L-10		144°27'	-19°53'		-4.36				
				L-11	(252°13.5')	(-19°53')	55.56		-2.39	-	N75°E	
		C-17			291°43'							
			C-1611		223°25'			-3.41				
				C-1612	(291°42.5')	(-0-)	24.21		-3.20	×	N42.5E	
		L-10			194°43							
			L-11		29°27'			-2.94				
				L-12	(194°43.5')	(-0-)	31.73		-2.73	-6.25	S88°E	



Needs checking

SURVEY DATA - OFFICE COMPUTATIONS

SURVEY NO. 1

SHEET 3 OF _____

BRG. OF INITIAL BACKSIGHT FROM:
(X) PRECEDING PAGE, THIS SURVEY
( ) SURVEY NO. _____, SHEET _____

COORDINATES AND ELEV. OF INITIAL INST. STA. FROM
(X) PRECEDING PAGE, THIS SURVEY
( ) SURVEY NO. _____, SHEET

N15°46½'W	(C-14)	1977.32	1469.99

COMP. BY	COSINE VERT. ANGLE	HORIZ. DISTANCE	SINE VERT. ANGLE	VERT. DIST.	BEARING	COSINE BEARING	SINE BEARING	LATITUDE OF FS	DEPARTURE OF FS	ELEVATION OF FS
								+	+ 23 86	+
								− 2 00	−	−
		23.94		-0-	S85°13'E		(C-15)	1975 32	1493 85	
					BS N15°46½'W		(C-14)	+1977.32	+1469.99 15	+
								− 8 63	−	−
		8.63		-0-	S01°00'E		(C-1411)	1968 69	1470 14	
					BS N15°46½'W		(C-14)	+1977.32	+1469.99 23 56	+
								− 39 21	−	−
		45.74 ✓		- 0.91	S31°00'E		(C-1412)	1938 11	1493 55	
					BS N85°13'W		(C-15)	+1975.32	+1493.85 13 12	+
								− 3 75	−	−
		13.64		-0-	S74°03½'E		(C-16)	1971 57	1506 97	
					BS N85°13'W		(C-15)	+1975.32	+1493.85	+
								− 46 56	− 5 47	−
		46.88		-0-	N06°42'W		(C-1501)	2021 88	1488 38	
					BS N74°03½'W		(C-16)	+1971.57	+1506.97 39 34	+
								− 10 51	−	−
		40.72		-0-	S75°02½'E		(C-17)	1961 06	1546 31	
					BS N74°03½'W		(C-16)	+1971.57	+1506.97	+
								− 13 86	− 1 53	−
		13.94 ✓		- 22.15	N06°18'E		(L-10)	1985 43	1508 50	
					BS N75°02½'W		(C-17)	+1961.06	+1546.31 8 83	+
								− 21 86	−	−
		23.58		-0-	S22°00'E		(C-18)	1939 20	1555 14	
					BS N75°02½'W		(C-17)	+1961.06	+1546.31	+
								− 25 14	− 5 05	−
		25.64 ✓		+ 21.84	N11°21'E		(E-15)	1986 20	1551 36	
					BS N75°02½'W		(C-17)	+1961.06	+1546.31	+
								− 6 17	− 16 01	−
		17.16		-0-	N68°56½'W		(C-1611)	1967 23	1530 30	
								+	+	+
								−	−	−
		24 X		-0-						
					BS N22°00'W		(C-18)	+1939.20 6 31	+1555.14 29 97	+
								−	−	−
		30.63 ✓		-0-	N78°06½'E		(C-19)	1945 51	1585 11	
					BS N22°00'W		(C-18)	+1939.20	+1555.14	+
								− 7 36	− 4 95	−
		8.87 ✓		-0-	S33°54½'W		(C-20)	1931 84	1550 19	
								+	+	+
								−	−	−
		52.25		- 18.90						
					BS S68°56½'E		(C-1611)	+1967.23 17 77	+1530.30 16 44	+
								−	−	−
		24.21		-0-	N42°46'E		(C-1612)	1985 00	1546 74	
								+	+	+
								−	−	−
		31.73		-0-						

SURVEY DATA - FIELD SHEET

SURVEY NO. 1 — 'C' LEVEL - 'A' LEVEL VIA McCAULEY STOPE — SHEET 8 OF _____

MINE QUEEN OF BRONZE — LEVEL _____ — WORKING (S) _____

DATE	ENG. & HELPER	BS	INST	FS	HORIZONTAL ANGLE	VERTICAL ANGLE	SLOPE DISTANCE	HI	HP	PT. TO RAIL	COMPASS BEARING	REMARKS AND SKETCHES
8/7/79	JWW	C-13			151°18'30"	0° 0'						
	RAP		C-1311		302°38'	0° 0'		-5.13				
				M-10	(151°19') ✓	(0° 0')	18.13		-3.66		N 29°E	
8/8/79	JWW	C-1311			230°55'	0° 0'						
	RAP		M-10		101° 50'	0° 0'		-1.24				
				M-11	(230°55') ✓	(0° 0')	16.60		-1.93		N 73°E	
		M-10			53°15'30"	+36°53'30"						
			M-11		106°32'	+36°53'00"		-2.39				
				M-12	(53°16') ✓	(+36°53'15") ✓	21.86		-5.76		N 47°W	
		M-11			264°20'	39°26'						
			M-12		168°40'30"	39°25'30"		-2.75				
				A-16	(269°20'15")	(39°25'45")	34.44		-1.12		N 34°E	
8/8/79	JWW	C-14			229°43'	-7°35'						
	RAP		C-1411		99°26'30"	-7°35'		-1.74				
				C-1413	(229°43'15")	(-7°35')	11.32		-0.54			
					( )	( )						
					( )	( )						
					( )	( )						
					( )	( )						
					( )	( )						
					( )	( )						
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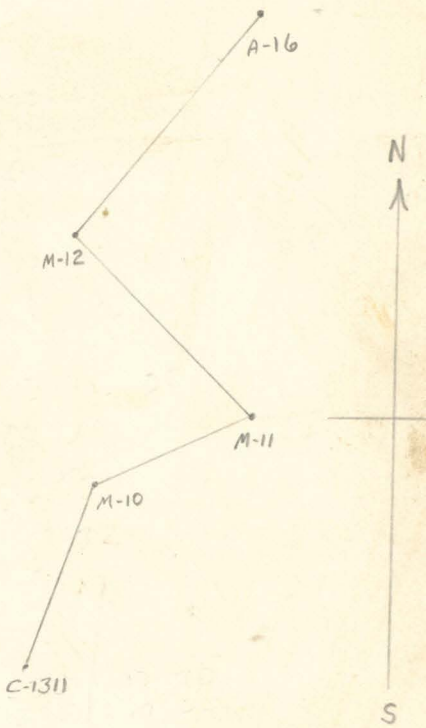
					( )	( )						
					( )	( )						
					( )	( )						



229 13 → 229 13
229 13 → 99 26 3
458 26 → 328 39
360 00
98 26

229 43
229 43
458 86
3
459 26
360 00
99 26

SURVEY DATA - OFFICE COMPUTATIONS

SURVEY NO. 2

SHEET 7 OF _____

BRG. OF INITIAL BACKSIGHT FROM:
( ) PRECEDING PAGE, THIS SURVEY
(X) SURVEY NO. 1, SHEET 3

S42°46'W (C-1612)

COORDINATES AND ELEV. OF INITIAL INST. STA. FROM
( ) PRECEDING PAGE, THIS SURVEY
( ) SURVEY NO. _____, SHEET

1985.00 | 1546.74

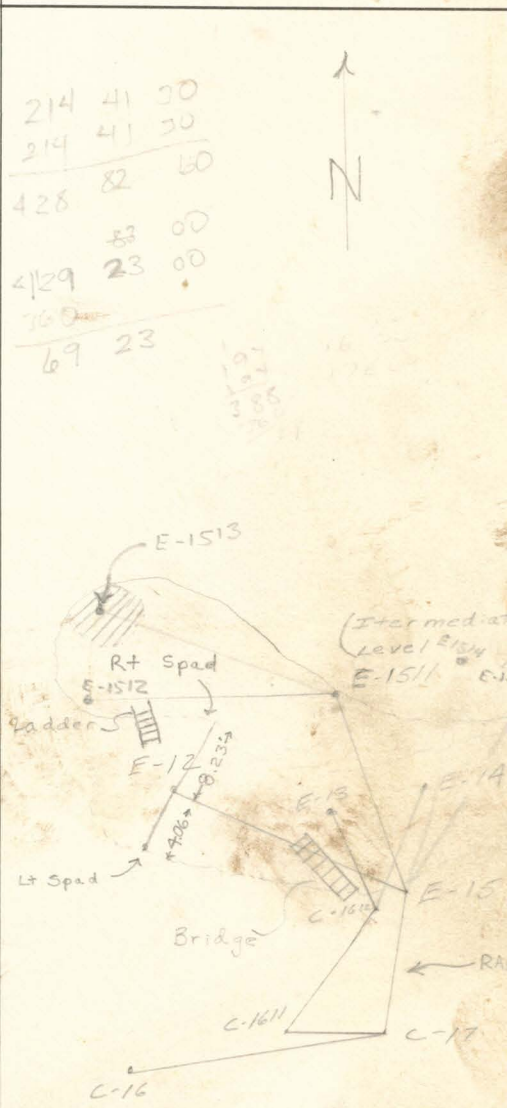
COMP. BY	COSINE VERT. ANGLE	HORIZ. DISTANCE	SINE VERT. ANGLE	VERT. DIST.	BEARING	COSINE BEARING	SINE BEARING	LATITUDE OF FS	DEPARTURE OF FS	ELEVATION OF FS
								+ 15 16	+ 7 17	+
								—	—	—
	✓	16.77 ✓		✓ +10.92	N25°19'E		(E-14)	2000 16	1553 91	
					BS S42°46'W		(E-1612)	+1985.00 14 72	+1546.74 — 6 12	+ —
	✓	15.94 ✓		✓ +11.53	N22°35½'W		(E-13)	1999 72	1540 62	
								+	+	+
								—	—	—
		29.74 ✓		+11.84						
								+	+	+
		35.19 ✓		-2.69				—	—	—
								+	+	+
		8.23 ✓		-0-				—	—	—
								+	+	+
								—	—	—
								+	+	+
		23.55 ✓						—	—	—
								+	+	+
		27.82						—	—	—
		22.93		15.86						
					BS S25°19'W		(E-14)	+2000.16	+1553.91	+
		21.21 ✓		-13.72	S69°58'E		(C-21)	— 7 27 1992 89	— 19 93 1573 84	—
							↓	+	+ 10 98	+
		11.26 ✓		+2.66	S77°06'E		(C-22)	— 2 51 1990 38	— 1584 82	—
								+	+	+
								—	—	—
								+	+	+
		22.94						—	—	—
								+	+	+
		19.25						—	—	—
								+	+	+
								—	—	—
								+	+	+
								—	—	—
								+	+	+
								—	—	—

SURVEY DATA - FIELD SHEET

SURVEY NO. 2 SHEET 7 OF

MINE Q of B LEVEL E WORKING (S) Camera Stope

DATE	ENG. & HELPER	BS	INST	FS	HORIZONTAL ANGLE	VERTICAL ANGLE	SLOPE DISTANCE	HI	HP	PT. TO RAIL	COMPASS BEARING
7/19/77	TJE JWW	C-1611			162°33'	+33°03'					
			C-1612		325°06'	+33°04'		-2.56			
				E-14	(162°33')	(+33°035)	20.01		—	—	N23.5°E
		C-1611			114°39'	+35°53'					
			C-162		229°17'	+35°53'		2.56			
				E-13	(114°38.5')	(35°53')	19.67		+.4		N22.5°W
7/26/77	TJE JWW	C-17			139°43'	+21°42'					
			E-15		279°26'	+21°42'		-5.88			
				E-1511	(139°43')	(+21°42')	32.01		-5.56	-	N28.5°W
		C-17			98°35'	-4°23'					
			E-15		197°11'	-4°22'		-5.88			
				E-12	(98°355')	(-4°22.5)	35.29		+044	-	N69°W
		E-15			265°09'						
			E-12		170°19'			—			
				Rt Spad	(265°09')	(—)	8.23		—	—	N15.5°E
					( )	( )					
12/2/77	JWW WSD	E-15			100°08'						
			E-1511		200°16'			-2.55			
				E-1512	(100°08')	(- 0 -)	23.55		-2.43	7.1	S 73°W
		E-15			126°27'	-3°40'					
			E-1511		242°54'	-3°41'		-2.55			
				E-1513	(126°27')	(-3°405')	27.88		-0-	—	N81°W
		C-1612			204°43'	-32°53'					
			E-14		169°26'	-32°54'		+1.67			
				C-21	(264°43')	(-32°535)	25.26		-2.87	5.7	S68°30'E
		E-14			172°52'	+13°16'					
			C-21		345°44'	13°16'		-1.94			
				C-22	(172°52')	(+13°16')	11.57		-0-	6.1	S79°E
					( )	( )					
		C-17			109°40'	-17°05'					
			E-15								
				Pt. A	(109°40')	(+17°05')	24.0		-0-		
		C-17			128°16'						
			E-15								
				Pt. B	(128°16')	(+19°22')	20.4		-0-		
		C-17			114°08'30"	+19°13'					
			E-15		28°18'	+19°12'30					
				E-1514	(114°09')	(19°12'45)	23.91	-4.36	-0-	—	
		C-17			214°41'30	+16°45'					
			E-15		69°23'30"	+16°45'		-4.36			
				E-1515	(214°41'45")	(+16°45')	27.16		+.02	—	N 43°E
					( )	( )					



SURVEY DATA - OFFICE COMPUTATIONS

SURVEY NO. 1

SHEET 4 OF _____

BRG. OF INITIAL BACKSIGHT FROM:
( ) PRECEDING PAGE. THIS SURVEY
(X) SURVEY NO. 1 . SHEET 2

N 79°11'E

COORDINATES AND ELEV. OF INITIAL INST. STA. FROM
( ) PRECEDING PAGE. THIS SURVEY
(X) SURVEY NO. 1 . SHEET 2

(C-9) 2002.17 1331.70

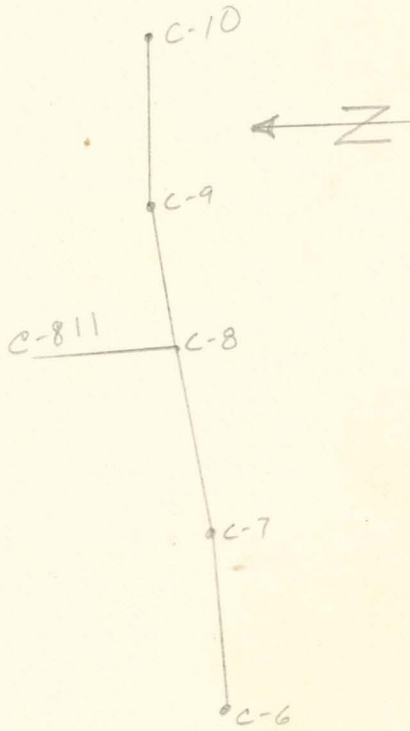
COMP. BY	COSINE VERT. ANGLE	HORIZ. DISTANCE	SINE VERT. ANGLE	VERT. DIST.	BEARING	COSINE BEARING	SINE BEARING	LATITUDE OF FS	DEPARTURE OF FS	ELEVATION OF FS
								+	+	+
								− 7 43	− 15 01	−
		16.75		-0-	S 63°39½'W		(C-8)	1994 74	1316 69	
								+ 10 60	+	+
								−	− 1 78	−
		10.75		-0-	N 09°32'W		(C-811)	2005 34	1314 91	
					BS		(C-8)	+1994.74	+1316.69	+
					N 63°39½'E		↓	− 7 92	− 14 95	−
		16.92		-0-	S 62°04½'W		↓	1986 82	1301 74	
							↓	+	+	+
							↓	− 4 49	− 21 07	−
		21.54		-0-	S 77°58'W		(C-6)	1982 33	1280 67	
RCP							↓	+	+	+
							↓	− 8 88	− 5 72	−
		10.56		-0-	S 32°48'30"W		(C-6A)	1973 45	1274 95	
								+	+	+
								−	−	−
					BS =		(C-1411)	+1968.69	+1470.14	+
					N 01°00'W			− 7 40	− 8 43	−
		11.22			S 48°43'15"W			1961 29	1461 71	
								+	+	+
								−	−	−
					BS =		(C-1501)	+2021.88	+1488.38	+
					S 06°42'E			− 2 84	− 30 19	−
		30.32			N 84°38'W			2024 72	1458 19	
								+	+	+
								−	−	−
							(C-15)	+1975.32	+1493.85	+
					S 74°03'30"E			− 18 93	− 13	−
		18.93			N 00°24'15"W			1994 25	1493 72	
								+	+	+
								−	−	−
								+	+	+
								−	−	−
								+	+	+
								−	−	−
								+	+	+
								−	−	−
								+	+	+
								−	−	−
								+	+	+
								−	−	−

SURVEY DATA - FIELD SHEET

SURVEY NO. 1 SHEET 4 OF

MINE Q of B LEVEL C WORKING (S)

DATE	ENG. & HELPER	BS	INST	FS	HORIZONTAL ANGLE	VERTICAL ANGLE	SLOPE DISTANCE	HI	HP	PT. TO RAIL	COMPASS BEARING	REMARKS AND SKETCHES
6/14/77	TJF JWW	C-10			164°28'							
			C-9		328°57'			-1.38				
				C-8	(164°28.5')	(-0-)	16.75		-0.91	5.35	N44E	
		C-9			286°49'							
			C-8		213°21'	0.37'?		-1.41				
				C-811	(286°485')	(-0-)	10.75		-2.36	6.85	S28E	
		C-9			178°25'							
			C-8		356°50'			-1.41				
				C-7	(178°25')	(-0-)	16.92		-2.73	6.65	N42E	
		C-8			195°53'							
			C-7		31°47'			-2.04				
				C-6	(195°53')	(-0-)	21.54		-1.22	6.3	N59E	
8/14/79	JWW/RAP	C-7			134°50'00"	-0-						
			C-6		269°41'00"	-0-		-1.15				
				C-6A	(134°50'30")	(-0-)	10.56		-1.88	N/A	S25°W	
					( )	( )						
8/8/79	JWW/RAP	C-14			229°43'00"	-7°35'						
			C-1411		99°26'30"	-7°35'		-1.74				
				C-1413	(229°43'15")	(-7°35')	11.32		-0.54	N/A	S50°W	
					( )	( )						
8/13/79	JWW/RAP	C-15			102°04'30"	-2°37'						
			C-1501		204°08'00"	+2°37'		-1.89				
				C-1502	(102°04'00")	(-2°37')	30.35		-3.11	N/A	N85°W	
					( )	( )						
8/?/79	JWW/RAP	C-16			253°39'00"	-0-						
			C-15		147°18'30"	-0-		-1.78				
				C-1503	(253°39'15")	(-0-)	18.93		-1.49	6.85	N02°E	
					( )	( )						
					( )	( )						
					( )	( )						
					( )	( )						
					( )	( )						



SURVEY DATA - OFFICE COMPUTATIONS

SURVEY NO. 1

SHEET 8 OF _____

BRG. OF INITIAL BACKSIGHT FROM:
( ) PRECEDING PAGE, THIS SURVEY
(X) SURVEY NO. 1, SHEET 2

S 55° 46½ W

COORDINATES AND ELEV. OF INITIAL INST. STA. FROM:
( ) PRECEDING PAGE, THIS SURVEY
( ) SURVEY NO. _____, SHEET

COMP. BY	COSINE VERT. ANGLE	HORIZ. DISTANCE	SINE VERT. ANGLE	VERT. DIST.	BEARING	COSINE BEARING	SINE BEARING	LATITUDE OF FS	DEPARTURE OF FS	ELEVATION OF FS
								+	+	+
								−	−	−
		18.13			N 27° 05½ E					
								+	+	+
								−	−	−
		16.60			N 78° 00½ E					
								+	+	+
								−	−	−
		17.48			N 48° 43½ W					
								+	+	+
								−	−	−
								+	+	+
								−	−	−
								+	+	+
								−	−	−
								+	+	+
								−	−	−
								+	+	+
								−	−	−
								+	+	+
								−	−	−
								+	+	+
								−	−	−
								+	+	+
								−	−	−
								+	+	+
								−	−	−
								+	+	+
								−	−	−
								+	+	+
								−	−	−
								+	+	+
								−	−	−
								+	+	+
								−	−	−
								+	+	+
								−	−	−

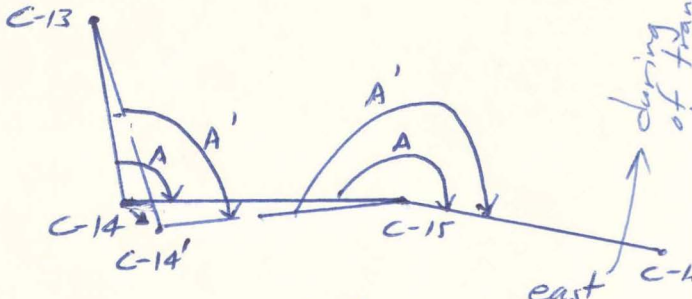
POSSIBLE REASON FOR DISCREPANCY

WESTERN INTERNATIONAL HOTELS

6/28/79

The Olympic

∠A = 'C' Level traverse
∠A' = Loop traverse



during setup

If Sta. C-14 shifted to ~~southeast~~, ∠A' > ∠A in both setups. Also, the distance C14-C15 would decrease slightly, but ~~distance C13-C~~ ~~would remain the same.~~

Looks like Sta. C-14 moved eastward

For reservations at any of our hotels worldwide, call 800-228-3000

~~between surveys.~~ or southeastward

No serious discrepancy in |C15| C16/L

C' Level traverse

Loop traverse

C10 – C11

179°32'
359°05'
179°32½

40.09

179°31'
359°03
179°31½

40.06

C11 – C12

245°29'
130°58'
245°29'

11.42

245°30'
131°00'
245°30'

11.43

C12 – C13

143°21'
286°43'
143°21½

62.97

143°21'
286°42'
143°21

6245

C13 – C14

927.22 Lat
5468.51 Dep

236°40'
113°19'
236°39½

10.68

927.24 Lat
5468.47 Dep

236°40'
113°21'
236°40½

10.68

C13 – C14 – C15

924.16 Lat
5492.25 Dep.

110°34'
221°07'
110°33½

23.94

924.16 Lat
5492.18 Dep

110°37'
221°13'
110°36½

23.91

+3'

C14 – C15 – C16

919.83 Lat.
5505.19 Dep.

191°10'
22°19'
191°09½

13.64

919.81 Lat
5505.12 Dep.

191°12'
22°24'
191°12'

13.65

+2½'

C16 – L10

80°21'
160°43'
80°21½

13.94

80°22'
160°45'
80°22½

13.94

933.60 Lat.
5507.33 Dep.

933.58 Lat
5507.29 Dep.

SURVEY DATA - FIELD SHEET

SURVEY NO. 4 SHEET 1 OF 3

MINE Q of B LEVEL C-D & Leach WORKING (S) Traverse Loop

#28169 Buff

DATE	ENG. & HELPER	BS	INST	FS	HORIZONTAL ANGLE	VERTICAL ANGLE	SLOPE DISTANCE	HI	HP	PT. TO RAIL	COMPASS BEARING	REMARKS AND SKETCHES
8/29/77	TJF JWW	C-9			179°31'							
			C-10		359°03'			-1.35				
				C-11	(179°31.5')	(-0-)	40.06		-1.49		N79E	
		C-10			245°30'							
			C-11		131°00'			-1.40				
				C-12	(245°30')	(-0-)	11.43		-1.77		S35E	
		C-11			143°21'							
			C-12		38°42'			-1.43				
				C-13	(143°21')	(-0-)	62.45		-4.54		S73E	
		C-12			236°40'							
			C-13		113°21'			-3.80				
				C-14	(236°40.5')	(-0-)	10.68		-0.77		S15E	
		C-13			110°37'							
			C-14		221°13'			-1.79				
				C-15	(110°36.5')	(-0-)	23.91		-3.56		S85E	
		C-14			191°12'							
			C-15		22°24'			-2.70				
				C-16	(191°12')	(-0-)	13.65		-2.63		S72E	
		C-15			80°22'	-58°47'						
			C-16		160°45'	-58°48'		-2.02				
				L-10	(80°22.5')	(-58°47.5')	26.90		-4.72		N7°E	
		C-16			252°14'	-19°51'						
			L-10		144°29'	-19°51'		-4.41				
				L-11	(252°14.5')	(-19°51')	55.54		-2.46		N79E	
		L-10			194°43'							
			L-11		29°27'			-3.07				
				L-12	(194°43.5')	(-0-)	31.73		-2.86		S89E	
		L-11			266°05'							
			L-12		172°10'			-1.45				
				L-13	(266°05')	(-0-)	32.98		-.087	-	S1°E	
8/30/77	TJF JWW				186°43'							
					13°27'							
					(186°43.5')	(-0-)						
		L-12			200°10'	-58°32'						
			L-13		26°55'	-58°33'		-1.54				
				D-49	(186°43.5')	(-58°32.5')	70.96		-7.61		S5°W	
		L-13			173°15'							
			D-49		346°31'		-	6.88				
				D-48	(173°15.5')	(-0-)	43.40		0.73		S0°00'	
		D-49			266°10'							
			D-48		172°20'		-	2.38				
				D-47	(266°10')	(-0-)	42.14		-1.30		S85°W	
		D-48			182°32'							
			D-47		5°04'		-	1.51				
				D-46	(182°32')	(-0-)	18.83		-0.84		S89°W	
		D-47			170°52'							
			D-46		341°45'		-	1.29				
				D-45	(170°52.5')	(-0-)	52.18		-0.64		S80°W	

SURVEY DATA - OFFICE COMPUTATIONS

SURVEY NO. 4

SHEET 1 OF 43

BRG. OF INITIAL BACKSIGHT FROM:
(X) PRECEDING PAGE, THIS SURVEY
( ) SURVEY NO. _____, SHEET _____

S 81°44 W

COÖRDINATES AND ELEV. OF INITIAL INST. STA. FROM:
(X) PRECEDING PAGE, THIS SURVEY
( ) SURVEY NO. _____, SHEET _____

962.58 (COS) | 5361.52 (SIN) | 2315.9

COMP. BY	COSINE VERT. ANGLE	HORIZ. DISTANCE	SINE VERT. ANGLE	VERT. DIST.	BEARING	COSINE BEARING	SINE BEARING	LATITUDE OF FS	DEPARTURE OF FS	ELEVATION OF FS
								+ 6 09	+ 39 60	+
								−	−	−
		40.06		-0-	N 81°15.5'E			968 67	5401 12	2316 1
								+	+ 6 27	+ 3
								− 9 56	−	−
		11.43		-0-	S 33°14.5'E			959 11	5407 39	2316 6
								+	+ 58 64	+ 3 1
								− 21 47	−	−
		62.45		-0-	S 69°53.5'E			937 64	5466 03	2319 6
								+	+ 2 44	+
								− 10 40	−	− 3 0
		10.68		-0-	S 13°13'E			927 24	5468 47	2316 5
								+	+ 23 71	+ 1 7
								− 3 08	−	−
		23.91		-0-	S 82°36.5'E			924 16	5492 18	2318 3
								+	+ 12 94	+
								− 4 35	−	− 0 0
		13.65		-0-	S 71°29.5'E			919 81	5505 12	2318 2
								+ 13 77	+ 2 17	+
								−	−	− 20 3
	−	13.94	−	23.01	N 8°58'E			933 58	5507 29	2297 9
								+ 7 98	+ 51 63	+
								−	−	− 20 8
		52.24	−	18.86	N 81°12.5'E			941 56	5558 92	2277 1
					BS = S81°04½'W (S84°12'E) (from Sur. 1, p 3)			+	+ 31 56	+
								− 3 28	−	− 2
		31.73		-0-	S 84°04'E			938 28	5590 48	2276 9
					BS = N84°12'W (S01°53'W)			+	+	+
								− 32 96	− 1 16	− 5
		32.98		-0-	S 02°01'W			905 32	5589 32	2276 3
								+	+	+
								−	−	−
		~~37.03~~		-0-				~~36 72~~		
								+	+	+
								− 36 60	− 5 63	− 54 4
	+	37.03	−	60.53	S 8°44.5'W			868 72	5583 69	2221 9
								+	+	+
								− 43 37	− 1 52	− 6 1
		43.40		-0-	S 02°00'W			825 35	5582 17	2215 7
								+	+	+
								− 1 35	− 42 12	− 1 0
		42.14		-0-	S 88°10'W			824 00	5540 05	2214 6
								+ 0 23	+	+
								−	− 18 83	− 6
		18.83		-0-	N 89°18'W			824 23	5521 22	2214 0
								+	+	+
								− 7 65	− 51 62	− 6
		52.18		-0-	S 81°39.5'W			816 58	5469 60	2213 3
		~~486~~ 65								

SURVEY DATA - FIELD SHEET

SURVEY NO. 4 SHEET 2 OF 3

MINE Q of B LEVEL C-D & Leach WORKING (S) Traverse Loop

DATE	ENG. & HELPER	BS	INST	FS	HORIZONTAL ANGLE	VERTICAL ANGLE	SLOPE DISTANCE	HI	HP	PT. TO RAIL	COMPASS BEARING	REMARKS AND SKETCHES
		D-46			205°20'							
			D-45		50°40'			-1.78				
				D-44	(205°20')	(-0-)	28.24		-1.70		N75W	
		D-45			152°56'	-0°38'						
			D-44		305°52'	-0°38'		-2.33				
				D-43	(152°56')	(0°38')	83.24		-7.62		S7W	
		D-44			261°14'							
			D-43		162°28'			-6.07				
				D-42	(261°14')	(-0-)	16.78		+1.47		N19°W	
		D-43			201°31'	+8°49'						
			D-42		43°02'	+8°50'		-1.68				
				D-41	(201°31')	(+8°49.5)	13.69		-3.79		N1°E	
		D-42			138°57'	-3°56'						
			D-41		277°53'	-3°56'		-2.09				
				D-40	(138°56.5')	(-3°56')	13.10		-1.72		N4°W	
		D-41			211°35'							
			D-40		63°08'							
				T-1	(211°34')	( )						
		D-41			274°43'	+56°29'						
			D-40		126°16'	+56°30'		-1.32				
				T-1	(211°34')	(+56°29.5')	42.62		+6.21		N8W	
		D-40			238°39'							
			T-1		117°15'							
				T-3	(238°37.5')	( )						
					355°53	+6°43'						
					234°29	+6°44'		+4.92				
					(238°37.5')	(+6°43.5')	15.28		-5.81		N51E	
		T-1			90°00	+32°58						
			T-3		179°59'	+32°59		-2.11				
				L-1	(89°59.5')	(+32°58.5')	26.84		-1.85		N40°W	
		T-3			235°37'							
			L-1		111°13'			-1.51				
				L-2	(235°36.5')	(-0-)	3.85		-1.09		N19E	
		L-1			210°36'							
			L-2		61°13'			-1.68				
				L-3	(210°36.5')	(-0-)	29.58		-1.93		N49E	
		L-2			198°24'							
			L-3		36°48'			-1.05				
				L-4	(198°24')	(-0-)	14.66		-2.03		N66E	
		L-3			187°47'							
			L-4		15°34'			-1.62				
				L-5	(187°47')	(-0-)	16.54		-4.28		N74E	
		L-4			95°43'	+50°08'						
			L-5		191°26'	+50°09'		-2.16				
				C-1017	(95°43')	(+50°08.5')	27.36		+1.58		N13W	
		L-5			180°15'	+31°29'						
			C-1017		100°30'	+31°29'		+2.14				
				C-1016	(180°15')	(+31°28.5')	18.91		-2.97		N10W	

SURVEY DATA - OFFICE COMPUTATIONS

SURVEY NO. 4

SHEET 2 OF 3

BRG. OF INITIAL BACKSIGHT FROM:
(X) PRECEDING PAGE. THIS SURVEY
( ) SURVEY NO. _____. SHEET _____

N81 34.5E

COORDINATES AND ELEV. OF INITIAL INST. STA. FROM:
(X) PRECEDING PAGE. THIS SURVEY
( ) SURVEY NO. _____. SHEET

816.58	5469.60	2213.36
COS	SIN	

COMP. BY	COSINE VERT. ANGLE	HORIZ. DISTANCE	SINE VERT. ANGLE	VERT. DIST.	BEARING	COSINE BEARING	SINE BEARING	LATITUDE OF FS	DEPARTURE OF FS	ELEVATION OF FS
								+ 8 21	+	+
								−	− 27 02	−
		28.24		-0-	N 73°05.5'W			824 79	5442 58	2213 2
								+	+	+ 4 3
								− 14 68	− 81 93	−
		83.23		-.92	S 79°50.5'W			810 [illegible]	5360 65	2217 6
								+ 15 87	+	+
								−	− 5 44	− 4 6
		16.78		-0-	N 18°55.5'W			825 98	5355 21	2213 0
								+ 13 52	+ 61	+ 4 2
								−	−	−
		13.53		+2.10	N 2°35.5'E			839 50	5355 82	2217 2
								+ 10 23	+	+
								−	− 8 13	− 1 2
		13.07		-0.90	N 38°28'W			849 73	5347 69	2215 9
								+	+	+
								−	−	−
		—		—	~~N 6°54'W~~					
								+ 23 36	+	+ 28 0
								−	− 2 83	−
		23.53	+	35.54	N 6°54'W			873 09	5344 86	2244 0
								+	+	+
								−	−	−
		—		—						
								+ 9 40	+ 11 92	+ 12 5
								−	−	−
		15.18		+1.79	N 51°43.5'E			882 49	5356 78	2256 5
								+ 17 67	+	+ 14 3
								−	− 13 95	−
		22.52		+14.61	N 38°17'W			900 16	5342 83	2270 8
								+ 13 22	+ 4 12	+
								−	−	− 4
		13.85		-0-	N 17°19.5'E			913 38	5346 95	2270 4
								+ 19 82	+ 21 96	+ 2
								−	−	−
		29.58		-0-	N 47°56'E			933 20	5368 91	2270 7
								+ 5 88	+ 13 43	+ 9
								−	−	−
		14.66		-0-	N 66°20'E			939 08	5382 34	2271 6
								+ 4 53	+ 15 91	+ 2 6
								−	−	−
		16.54		-0-	N 74°09'E			943 61	5398 25	2274 3
								+ 17 27	+	+ 17 2
								−	− 3 10	−
		17.54	+	21.00	N 10°10'W			960 88	5395 15	2291 6
								+ 15 89	+	+ 14 9
								−	− 2 78	−
		16.13	+	9.87	N 09°55'W			976 77	5392 37	2306 5

324.38

SURVEY DATA - FIELD SHEET

SURVEY NO. 4 SHEET 3 OF 3

MINE Q of B LEVEL C-D & Leach WORKING (S) Traverse loop

DATE	ENG. & HELPER	BS	INST	FS	HORIZONTAL ANGLE	VERTICAL ANGLE	SLOPE DISTANCE	HI	HP	PT. TO RAIL	COMPASS BEARING	REMARKS AND SKETCHES
		C-1017			95°11'	+9°46'						95° 10.5
			C-1016		190°21'	+9°47'		-2.29				
				C-1015	(95°10.5')	(+9°46.5')	12.93		-3.04		S83W	
		C-1016			143°36'	+28°34'						
			C-1015		287°12'	+28°33'		-0.81				
				C-1011	(143°36')	(+28°33.5')	13.17		-3.39		S47W	
		C-1015			191°32'							
			C-1011		23°04'			-3.96				
				C-10	(191°32')	(-0-)	10.75		-1.38		S79W	
		C-1011			21°11'							
			C-10		42°22'							
				C-11	(21°11')	( )						
					( )	( )						
					( )	( )						
					( )	( )						
					( )	( )						
					( )	( )						
					( )	( )						
					( )	( )						
					( )	( )						
					( )	( )						
					( )	( )						
					( )	( )						
					( )	( )						

SURVEY DATA - OFFICE COMPUTATIONS

SURVEY NO. 4

SHEET 3 OF 3

BRG. OF INITIAL BACKSIGHT FROM:
(X) PRECEDING PAGE, THIS SURVEY
( ) SURVEY NO. _____, SHEET _____

S 09°55'E

COÖRDINATES AND ELEV. OF INITIAL INST. STA. FROM:
(X) PRECEDING PAGE, THIS SURVEY
( ) SURVEY NO. _____, SHEET

976.77 | 5392.37 | 2306.58

COMP. BY	COSINE VERT. ANGLE	HORIZ. DISTANCE	SINE VERT. ANGLE	VERT. DIST.	BEARING	COSINE BEARING	SINE BEARING	LATITUDE OF FS	DEPARTURE OF FS	ELEVATION OF FS
								+	+	+ ✓ 2 9
							✓	− 1 05	− ✓ 12 70	−
	✓	12.74	✓	42.20	S 85°15'30"W			975 92	5379 67	2309 5
						✓ Was 48 42 30		+	+	+ ✓ 8 8
								− 7 61	− 8 71	−
	✓	11.57	✓	46.30	S 48°51'30"W			968 11	5370 96	2318 4
								+	+	+
								− 5 31	− 9 35	− ✓ 2 5
	✓	10.75			S 60°23.5W			962 80	5361 61	2315 8
								+ 5 87	+ 39 63	+
								−	−	−
	✓	40.06			N 61°34.5'E			968 67	5401 24	
								+	+	+
								−	−	−
								+	+	+
								−	−	−
		35.06			Σ E. O. C.			022	009	01
		324.38						+	+	+
		486.65						−	−	−
		846.09			~~Angular~~					
								+	+	+
								−	−	−
					Angle Error 19'					
					@ 33 shots = 34" per shot			+	+	+
										−
								+	+	+
								−	−	−
					Σ E.O.C. = $\sqrt{(.22)^2 + (.09)^2}$					
					= .2377 / 850			+	+	+
								−	−	−
					= 1/3560			+	+	+
								−	−	−
								+	+	+
								−	−	−
								+	+	+
								−	−	−
								+	+	+
								−	−	−
								+	+	+
								−	−	−

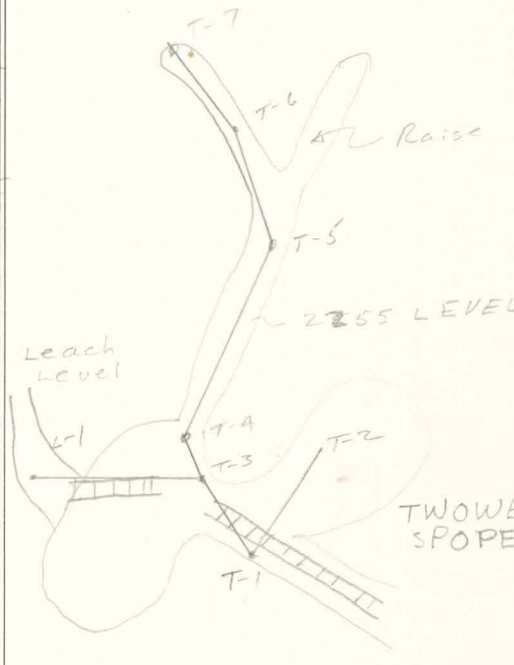
SURVEY DATA - FIELD SHEET

SURVEY NO. 6 SHEET 1 OF 1

MINE Q of B LEVEL 2255 WORKING (S) 2255 Level

DATE	ENG. & HELPER	BS	INST	FS	HORIZONTAL ANGLE	VERTICAL ANGLE	SLOPE DISTANCE	HI	HP	PT. TO RAIL	COMPASS BEARING	REMARKS AND SKETCHES
8/4/77	TJF JWW	T-1			171°06'							
			T-3		342°17'							
				T-4	(171°08.5')	( )						
		T-1			153°25'							
			T-3		324°34'			-2.16				
				T-4	(171°08.5')	(-0-)	7.77		-1.43	-6.25	N41°E	
10/11/77	11	T-3			242°10'							
			T-4		124°21'			-1.35				
				T-5	(242°10.5')	(-0-)	25.76		-1.41	-6.5	N65E	
		T-4			150°12'							
			T-5		300°24'			-1.18				
				T-6	(150°12')	(-0-)	24.64		-2.12	-6.0	N34°E	
		T-5			143°16'							
			T-6		286°31'			-1.54				
				T-7	(143°15.5')	(-0-)	18.85		-2.63	—	N4°W	
					( )	( )						
					( )	( )						
					( )	( )						
					( )	( )						
					( )	( )						
					( )	( )						
					( )	( )						
					( )	( )						
					( )	( )						
					( )	( )						
					( )	( )						

} Quad



SURVEY DATA - OFFICE COMPUTATIONS

SURVEY NO. 6

SHEET 1 OF 1

BRG. OF INITIAL BACKSIGHT FROM:
( ) PRECEDING PAGE, THIS SURVEY
(X) SURVEY NO. 4, SHEET 2

S51°43.5W

COÖRDINATES AND ELEV. OF INITIAL INST. STA. FROM:
( ) PRECEDING PAGE, THIS SURVEY
(X) SURVEY NO. 4, SHEET 2

882.49	5356.78	2256.52

COMP. BY	COSINE VERT. ANGLE	HORIZ. DISTANCE	SINE VERT. ANGLE	VERT. DIST.	BEARING	COSINE BEARING	SINE BEARING	LATITUDE OF FS	DEPARTURE OF FS	ELEVATION OF FS
								+	+	+
								−	−	−
								+ 5 69	+ 5 29	+
								−	−	− 0 7
		7.77		-0-	N42°52E			888 18	5362 07	2255 7
								+	+ 24 88	+ 00
								− 6 68	−	− 2
		25.76		-0-	S74°57'30"E			881 50	5386 95	2255 8
								+ 6 28	+ 23 83	+ 09
								−	−	−
		24.64		-0-	N75°14'30"E			887 78	5410 78	2256 7
								+ 14 75	+ 11 73	+ 1 0
								−	−	−
		18.85		-0-	N38°30'E			902 53	5422 51	2257 8
								+	+	+
								−	−	−
								+	+	+
								−	−	−
								+	+	+
								−	−	−
								+	+	+
								−	−	−
								+	+	+
								−	−	−
								+	+	+
								−	−	−
								+	+	+
								−	−	−
								+	+	+
								−	−	−
								+	+	+
								−	−	−
								+	+	+
								−	−	−
								+	+	+
								−	−	−

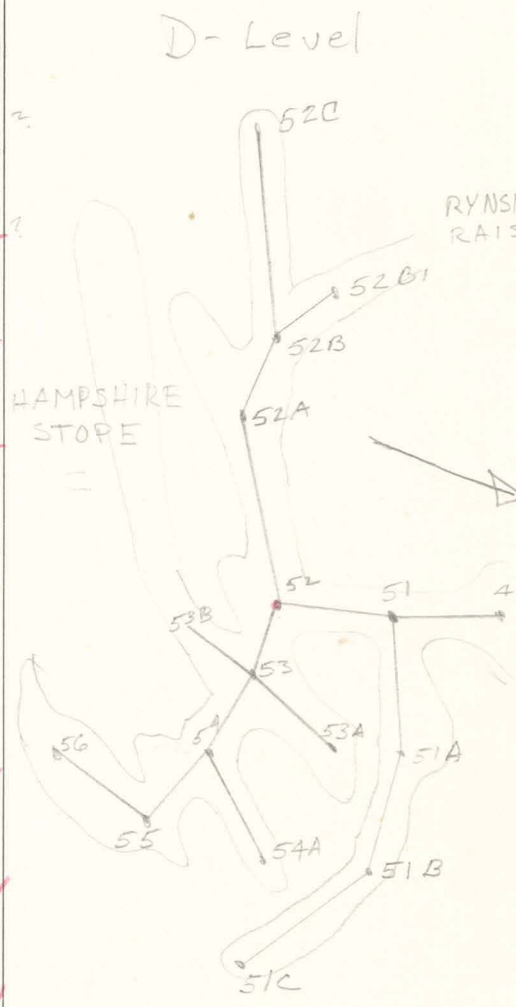
SURVEY DATA - FIELD SHEET

SURVEY NO. 5 SHEET 1 OF 1

MINE Q of B LEVEL D WORKING (S) Hampshire Stope & Rynski Raise

DATE	ENG. & HELPER	BS	INST	FS	HORIZONTAL ANGLE	VERTICAL ANGLE	SLOPE DISTANCE	HI	HP	PT. TO RAIL	COMPASS BEARING
9/3/77	TJF JWW	D-44			81°18'						
			D-43		162°36'			5.73			
				D-51	(81°18')	(-0-)	31.52		-1.83	-	S19.5E
10/10/77	TJF JWW	D-43			233°05						
			D-51		106°11'			3.40			
				D-52	(233°05.5')	(-0-)	30.91		-4.73	-9.5	S4°E
		D-43			143°17'						
			D-51		286°33'			3.40			
				D-51A	(143°16.5')	(-0-)	16.41		-1.92	-6.6	N83°E
		D-51			198°07'						
			D-51A		36°15'			1.62			
				D-51B	(198°07.5')	(-0-)	31.22		-3.62	-	S7°E
		D-51A			216°56'						
			D-51B		73°53'			3.86			
				D-51C	(216°56.5')	(-0-)	49.57		-0.99	-	S42E
		D-51			106°59'						
			D-52		213°58'			4.48			
				D-53	(106°59')	(-0-)	23.58		-6.13	-	S81°E
		D-51			248°47'						
			D-52		137°34'			4.48			
				D-52A	(248°47')	(-0-)	58.78		-2.42	-	N61°W
		D-52			221°12'						
			D-52A		82°25'			1.47			
				D 52B	(221°12.5')	(-0-)	11.48		-2.16	-	N77W
		D-52A			173°56'						
			D-52B		347°53'			2.13			
				D 52C	(173°56.5')	(-0-)	48.16		-2.27	-6.3	N84W
		D-52A			201°57'	+30°09'					
			D-52B		43°54'	+30°10'		2.13			
				D 52B1	(201°57')	(+30°09.5')	18.12		-0.74	-	N56W
10/11/77	"	D-52			211°54'						
			D-53		63°48'			5.72			
				D-54	(211°54)	(-0-)	16.00		-3.85	-8.3	S49E
		D-52			130°12'	+12°39'					
			D-53		260°24'	+12°39'		5.72			
				D 53A	(130°12')	(+12°39')	26.32		-4.21	-	N49E
		D-52			269°25'						
			D-53		178°49'			5.72			
				D 53B	(269°24.5')	(-0-)	11.40		-4.42	-	S9°W
		D-53			187°40						
			D-54		15°20'			3.42			
				D-55	(187°40')	(-0-)	24.02		-2.89	-7.0	S40E
		D-53			101°09'	+32°23'					
			D-54		202°18'	+32°23'		3.42			
				D-54A	(101°09')	(+32°23')	25.45		-2.76	-	N53E
		D-54			290°32'						
			D-55		221°04'			2.36			
				D-56	(290°32')	(-0-)	30.39		-2.05	-6.0	S70°W

REMARKS AND SKETCHES



SURVEY DATA - OFFICE COMPUTATIONS

SURVEY NO. 5

SHEET 1 OF 1

BRG. OF INITIAL BACKSIGHT FROM:
( ) PRECEDING PAGE, THIS SURVEY
(X) SURVEY NO. 4, SHEET 2

N79°50.5E

COÖRDINATES AND ELEV. OF INITIAL INST. STA. FROM:
( ) PRECEDING PAGE, THIS SURVEY
(X) SURVEY NO. 4, SHEET 2

810.11 | 5360.65 | 2217.65

COMP. BY	COSINE VERT. ANGLE	HORIZ. DISTANCE	SINE VERT. ANGLE	VERT. DIST.	BEARING	COSINE BEARING	SINE BEARING	LATITUDE OF FS	DEPARTURE OF FS	ELEVATION OF FS
								+	+ 10 19	+
								− 29 83	−	− 3 9
		31.52		-0-	S18°51'30"E			780 28	5370 84	2213 7
								+	+	+ 1 3
								− 25 55	− 17 39	−
		30.91		-0-	S34°14' W			754 73	5353 45	2215 0
								+	+ 13 54	+ 1 4
								− 9 27	−	−
		16.41		-0-	S55°35'E			771 01	5384 38	2215 2
								+	+ 18 99	+ 2 0
								− 24 78	−	−
		31.22		-0-	S37°27'30"E			746 23	5403 37	2217 2
								+	+ 0 45	+
								− 49 57	−	− 2 8
		49.57		-0-	S0°31'E			69 666	5403 82	2214 3
								+	+ 14 77	+ 1 6
								− 18 38	−	−
		23.58		-0-	S38°47'E			736 35	5368 22	2216 7
								+ 13 24	+	+
								−	− 57 27	− 2 0
		58.78		-0-	N76°59'W			767 97	5296 18	2213 0
								+ 9 31	+	+ 0 6
								−	− 6 71	−
		11.48		-0-	N35°46'30"W			777 28	5289 47	2213 7
								+ 35 88	+	+ 0 1
								−	− 32 12	−
		48.16		-0-	N41°50'W			813 16	5257 35	2213 8
								+ 15 24	+	+ 7 7
								−	− 3 75	−
		15.67		+9.10	N13°49'30"W			792 52	5285 72	2221 4
								+	+ 192	+
								− 15 88	−	− 1 8
		16.00		-0-	S 6°53' E			720 47	5370 14	2214 8
								+	+ 25 67	+ 4 2
								− 0 63	−	−
		25.68		+5.76	S 88°35' E			735 72	5393 89	2220 9
								+	+	+
								− 7 23	− 8 81	− 1 3
		11.40		-0-	S50°37'30"W			729 12	5359 41	2215 4
								+	+	+
								− 24 02	− 0 33	− 0 5
		24.02		-0-	S 0°47' W			696 45	5369 81	2214 3
								+	+ 21 43	+ 12 9
								− 1 59	−	−
		21.49		+13.63	S85°44'E			718 88	5391 57	2227 8
								+ 11 05	+	+
								−	− 28 31	− 0 3
		30.39		-0-	N68°41 W			707 50	5341 50	2214 0

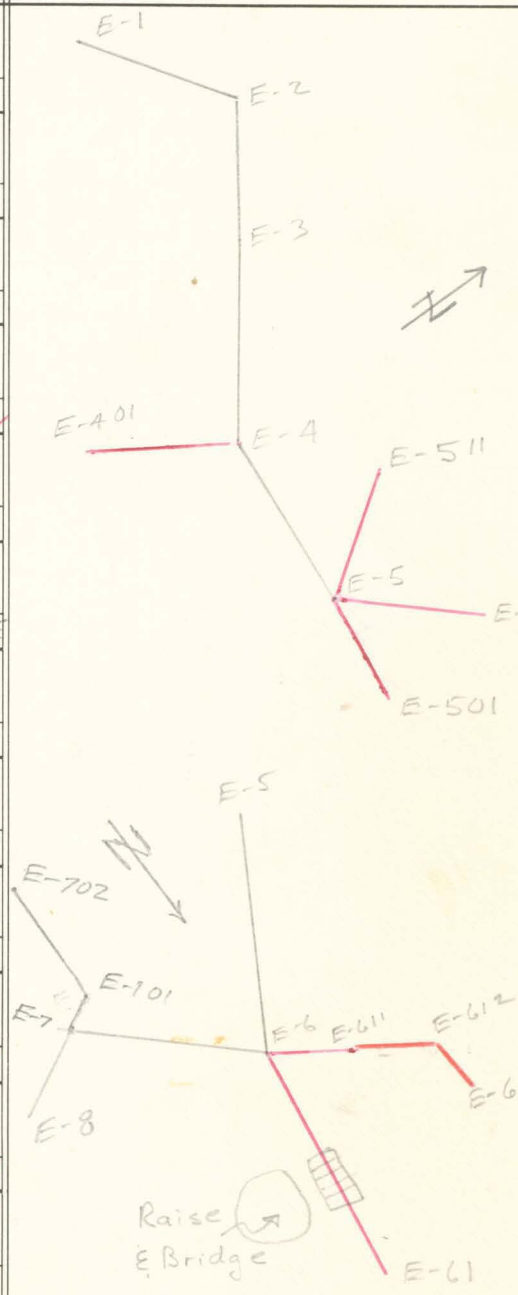
SURVEY DATA - FIELD SHEET

SURVEY NO. 2 SHEET 1 OF

MINE QUEEN OF BRONZE LEVEL E WORKING (S)

INST 28169 "BUFF"

DATE	ENG. & HELPER	BS	INST	FS	HORIZONTAL ANGLE	VERTICAL ANGLE	SLOPE DISTANCE	HI	HP	PT. TO RAIL	COMPASS BEARING
3/14/77	TJF JWW	C-4			161°58'						
			E-1		323°55' ✓			-2.31			
				E-2	(161°57.5')	(-0-)	39.79		-2.30	-7.25	N 37° E
"		E-1			250°20'						
			E-2		140°41' ✓			-2.61			
				E-3	(250°20.5')	(-0-)	28.70		-2.68	-7.40	S 72° E
3/15/77	TJ F JWW	E-2			191°42						
			E-3		23°23' ✓			-2.41			
				E-4	(191°41.5')	(-0-)	29.82		-1.66	-6.75	S61.5°E
"	"	E-3			248°14'						
			E-4		136°27'			-1.59			
				E-401	(248°13.5') ✓	(-0-)	25.25		-2.26	—	S7.5°W
"	"	E-3			144°40'	-5°					
			E-4		289°19'	-5°		-1.59			
				E-5	(144°39.5') ✓	(-5°) ✓	30.76		-3.05		N84°E
3/17/77	TJ F JWW	E-4			119°57'						
			E-5		239°53' ✓			-2.36			
				E-6	(119°56.5')	(-0-)	22.98		-2.75	-	N24.5°E
"	"	E-4			181°12'						
			E-5		02°25'			-2.36			
				E-501	(181°12.5')	(-0-)	10.50		-0.87	-	N56°E
"	"	E-4			61°50	+17°17'					
			E-5		123°40'	+17°17'		-2.36			
				E-511	(61°50')	(+17°17')	22.43		-0-	-	N34°W
"	"	E-5			290°06'						
			E-6		220°12' ✓			-3.21			
				E-7	(290°06')	(-0-)	40.03		-3.88	-	S46°E
"	"	E-5			93°22'	+7°00'					
			E-6		186°43'	+7°00'		-3.21			
				E-611	(93°21.5')	(+7°00')	11.44		-0-	-	N62°W
"	"	E-5			154°11'						
			E-6		308°23' ✓			-3.21			
				E-61	(154°11.5')	(-0-)	39.70		-1.88	-	N 1°W
					( )	( )					
"	"	E-6			127°14'	-6°00'					
			E-7		254°28'	-6°00' ✓		-1.67			
				E-8	(127°14') ✓	(-6°00') ✓	16.05		-1.57	-	
					( )	( )					
3/23/77	TJF JWW	E-6			290°20'	-25°33'					
			E-7		220°41'	-25°32'		-1.49			
				E-701	(290°20.5')	(-25°32.5')	7.36		-2.82	—	
"		E-7			106°46'						
			E-701		213°32'			-1.12			
				E-702	(106°46')	(-0-)	17.36		-1.89	—	



HI(+/-) -(HP)

SURVEY DATA - OFFICE COMPUTATIONS

SURVEY NO. 2

SHEET 1 OF

E - Level

BRG. OF INITIAL BACKSIGHT FROM:
( ) PRECEDING PAGE, THIS SURVEY
(X) SURVEY NO. 1, SHEET 1

S54°58½'W

COORDINATES AND ELEV. OF INITIAL INST. STA. FRO
( ) PRECEDING PAGE, THIS SURVEY
(X) SURVEY NO. 1, SHEET 1

(E-1) | 2004.31 | 1188.74

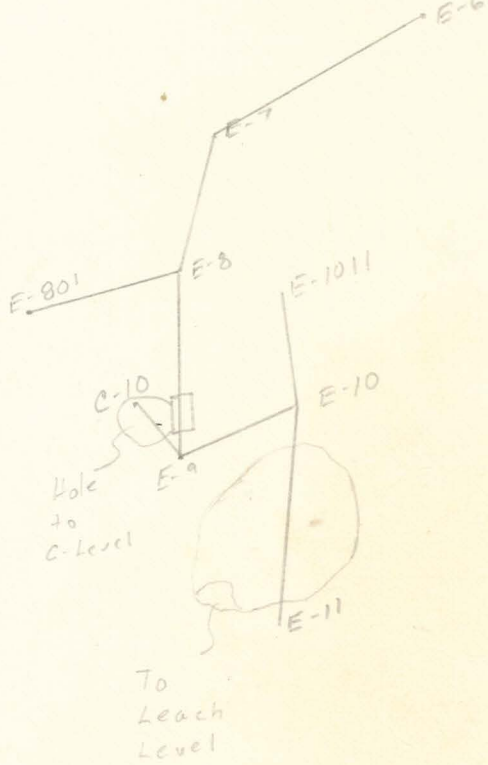
COMP. BY	COSINE VERT. ANGLE	HORIZ. DISTANCE	SINE VERT. ANGLE	VERT. DIST.	BEARING	COSINE BEARING	SINE BEARING	LATITUDE OF FS	DEPARTURE OF FS	ELEVATIO OF FS
								+ 31 81	+ 23 91	+
								—	—	—
		39.79		-0-	N36°56'E			2036 12	1212 65	
								+	+ 27 41	+
								— 8 52	—	—
		28.70		-0-	S72°43½'E			2027 60	1240 06	
								+	+ 26 09	+
								— 14 44	—	—
		29.82		-0-	S61°02'E		(E-4)	2013 16	1266 15	
					BS N61°02'W			+	+	+
								— 25 05	— 3 16	—
		25.25		-0-	S07°11½'W		(E-401)	1988 11	1262 99	
					BS N61°02'W		(E-4)	+2013.16 3 40	+1266.15 30 45	+
								—	—	—
		30.64 ✓		-2.68	N83°37½'E			2016 56	1296 60	
								+ 21 06	+ 9 19	+
								—	—	—
		22.98		-0-	N23°34'E		(E-6)	2037 62	1305 79	
					BS S83°37½'W			+	+	+
								—	—	—
		10.50		-0-						
					BS S83°37½'W			+	+	+
								—	—	—
		21.42		+6.66						
							(E-6)	+2037.62	+1305.79 28 96	+
								— 27 64	—	—
		40.03		-0-	S46°20'E		(E-7)	2009 98	1334 75	
					BS S23°34'W		(E-6)	+2037.62 5 14	+1305.79	+
								—	— 10 12	—
		11.35 ✓		+1.34	N63°04½'W		(E-611)	2042 76	1295 67	
					BS S23°34'W		(E-6)	+2037.63 39 67	+1305.79	+
								—	— 1 55	—
		39.70		-0-	N02°14½'W		(E-61)	2077 29	1304 24	
								+	+	+
								—	—	—
							(E-7)	+2009.98 2 52	+1334.75 15 76	+
								—	—	—
		15.96 ✓		-1.68	N80°54'E		(E-8)	2012 50	1350 51	
								+	+	+
								—	—	—
								+	+	+
								—	—	—
		6.64 ✓		-3.17						
								+	+	+
								—	—	—
		17.36		-0-						

SURVEY DATA - FIELD SHEET

SURVEY NO. 2 SHEET 2 OF _____

MINE Queen of Bronze #28169 "Buff" LEVEL E WORKING (S) _____

DATE	ENG. & HELPER	BS	INST	FS	HORIZONTAL ANGLE	VERTICAL ANGLE	SLOPE DISTANCE	HI	HP	PT. TO RAIL	COMPASS BEARING	REMARKS AND SKETCHES
3/24/77	TJF JWW	E-7			174°23'							
			E-8		348°47'			-1.66				
				E-9	(174°23.5')	(-0-)	21.00		-1.21	-3.80	N76°E	
		E-7			235°41'	+1°10'						
			E-8		111°22'	+1°10'		-1.66				
				E-801	(235°41')	(+1°10')	35.80		-0-	—	S43°E	
3/28/77	TJF JWW	E-8			62°31'							
			E-9		125°01'			-1.12				
				E-10	(62°30.5')	(-0-)	16.92		-3.57		N41°W	
"	"	E-8			327°43'	-45°00						
			E-9		295°27'	-45°00		-1.12				
				C-10	(327°43.5')	(-45°00')	19.26		-1.77	6.10	S43°W	
"	"	E-9			294°37'							
			E-10		229°14'			-3.35				
				E-11	(294°37')	(-0-)	24.90		-1.02	—	N72°E	
"	"	E-9			94°46'							
			E-10		189°31'			-3.35				
				E-1011	(94°45.5')	(-0-)	13.28		-2.79	—	S52W	
					( )	( )						
					( )	( )						
					( )	( )						
					( )	( )						
					( )	( )						
					( )	( )						
					( )	( )						
					( )	( )						
					( )	( )						
					( )	( )						



SURVEY DATA - OFFICE COMPUTATIONS

SURVEY NO. 2

SHEET 2 OF _____

BRG. OF INITIAL BACKSIGHT FROM:
(X) PRECEDING PAGE. THIS SURVEY
( ) SURVEY NO. _____. SHEET _____

S80°54'W

COORDINATES AND ELEV. OF INITIAL INST. STA. FROM
(X) PRECEDING PAGE. THIS SURVEY
( ) SURVEY NO. _____. SHEET

(E-8) 2012.50 | 1350.51 |

COMP. BY	COSINE VERT. ANGLE	HORIZ. DISTANCE	SINE VERT. ANGLE	VERT. DIST.	BEARING	COSINE BEARING	SINE BEARING	LATITUDE OF FS	DEPARTURE OF FS	ELEVATION OF FS
								+ 5 33	+ 20 31	+
								—	—	—
		21.00		-0-	N75°17½'E		(E-9)	2017 83	1370 82	
					BS S80°54'W		(E-8)	+2012.50	+1350.51 24 60	+
		35.79						— 26 00	—	—
		35.80		+.73	S43°25'E			1986 50	1375 11	
							(E-9)	+2017.83 12 53	+1370.82	+
								—	— 11 37	—
		16.92		-0-	N42°12'W		(E-10)	2030 36	1359 45	
							(E-9)	+2017.83	+1370.82	+
								— 9 96	— 9 29	—
		13.62		-13.62	S43°01'W		(E-10)	2007 87	1361 53	
							(E-10)	+2030.36	+1359.45	+
								—	—	—
		24.90		-0-						
								+	+	+
								—	—	—
		13.28		-0-						
								+	+	+
								—	—	—
								+	+	+
								—	—	—
								+	+	+
								—	—	—
								+	+	+
								—	—	—
								+	+	+
								—	—	—
								+	+	+
								—	—	—
								+	+	+
								—	—	—
								+	+	+
								—	—	—
								+	+	+
								—	—	—
								+	+	+
								—	—	—

SURVEY DATA - FIELD SHEET

SURVEY NO. 2 SHEET 3 OF _____

MINE Q of B LEVEL E WORKING (S) E-400 Xcut

28169 "Buff"

DATE	ENG. & HELPER	BS	INST	FS	HORIZONTAL ANGLE	VERTICAL ANGLE	SLOPE DISTANCE	HI	HP	PT. TO RAIL	COMPASS BEARING	REMARKS AND SKETCHES
3/16/77	TSF JWW	E-4			78°02'	+ 6°00'						
			E-401		156°04'	+ 6°00'		-1.54				
				E-402	(78°02')	(+ 6°00')	18.66		-2.04	—	N85.5°E	
		E-401			120°12'	+ 25°00'						
			E-402		240°25'	+ 25°00'		-0.82				
				E-403	(120°12.5')	(+25°00')	20.20		-1.52	—	N25.5°E	
		E-4			339°40'	+ 15°58'						
			E-401		319°21'	+ 15°58'		-1.54				
				E-404	(339°40.9')	(+ 15°58)	33.16		-0-	—	N12.5°W	← Vertical & Horz. ∠ taken directly on spad.
					( )	( )						
					( )	( )						
					( )	( )						
					( )	( )						
					( )	( )						
					( )	( )						
					( )	( )						
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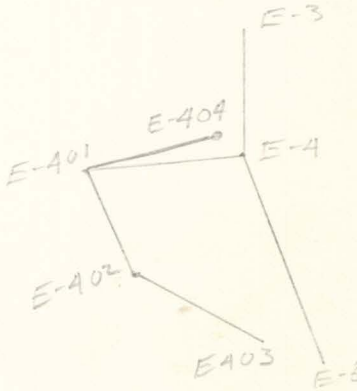
					( )	( )						
					( )	( )						
					( )	( )						



SURVEY DATA - OFFICE COMPUTATIONS

E-401 2332.02

SURVEY NO. 2

SHEET 3 OF

BRG. OF INITIAL BACKSIGHT FROM: Comp.
( ) PRECEDING PAGE, THIS SURVEY
( ) SURVEY NO. 2, SHEET 1

N9°44'30"E ✓

COORDINATES AND ELEV. OF INITIAL INST. STA. FROM:
( ) PRECEDING PAGE, THIS SURVEY
( ) SURVEY NO. 2, SHEET 1

9472.22	5262.20	2332.02

COMP. BY	COSINE VERT. ANGLE	HORIZ. DISTANCE	SINE VERT. ANGLE	VERT. DIST.	BEARING	COSINE BEARING	SINE BEARING	LATITUDE OF FS	DEPARTURE OF FS	ELEVATION OF FS
								+ 72	+ 18 55	+ 2 4
								—	—	—
		18.56		+1.95	N87°46'30"E	✓		947 94	5280 75	2334 4
								+ 16 17	+ 8 59	+ 9 2
								—	—	—
		18.31		+8.54	N27°59'E	✓		964 11	5289 34	2343 7
								+ 31 34	+	+ 7 5
								—	5 85	—
		31.88		+9.12	N10°35'W	✓		978 56	5256 35	2339 6
								+	+	+
								—	—	—
								+	+	+
								—	—	—
								+	+	+
								—	—	—
								+	+	+
								—	—	—
								+	+	+
								—	—	—
								+	+	+
								—	—	—
								+	+	+
								—	—	—
								+	+	+
								—	—	—
								+	+	+
								—	—	—
								+	+	+
								—	—	—
								+	+	+
								—	—	—
								+	+	+
								—	—	—

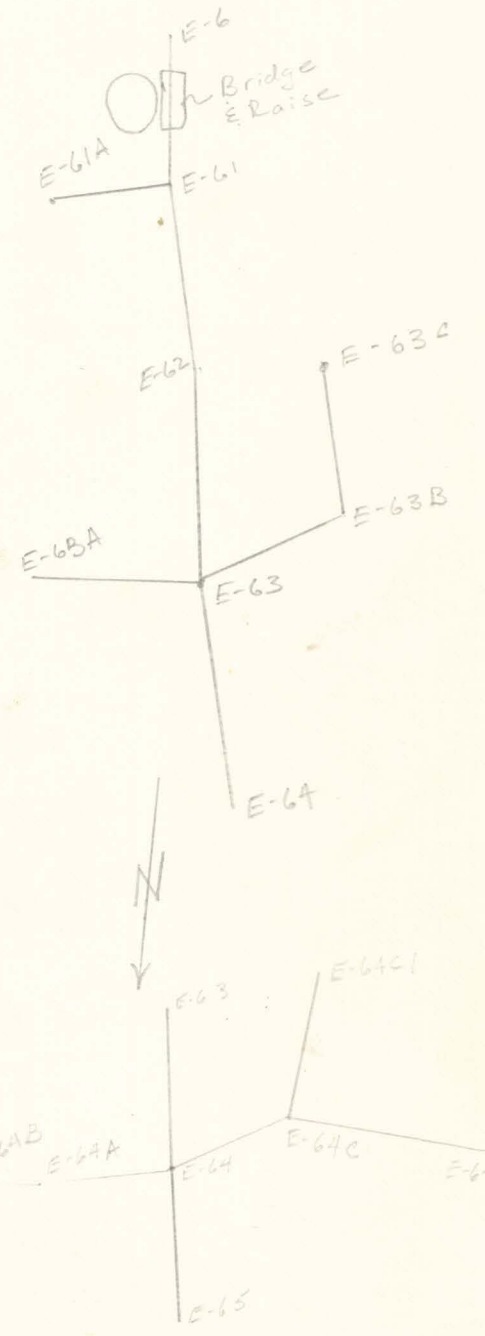
SURVEY DATA - FIELD SHEET

SURVEY NO. 2 SHEET 4 OF

MINE Q of B LEVEL E WORKING (S)

DATE	ENG. & HELPER	BS	INST	FS	HORIZONTAL ANGLE	VERTICAL ANGLE	SLOPE DISTANCE	HI	HP	PT. TO RAIL	COMPASS BEARING
3/28/77	TJF JWW	E-6			248°56'						
			E-61		137°52'			-2.08			
				E-61A	(248°56')	(-0-)	24.56		-2.89	-	N67°E
"	"	E-6			159°35'	-12°31'					
			E-61		319°09'	-12°32'		-2.08			
				E-62	(159°34.5')	(-12°31.5')	16.86		-5.36	-	N22°W
3/30/77	TJF JWW	E-61			186°45'						
			E-62		13°31'			-2.52			
				E-63	(186°45.5')	(-0-)	16.77		-6.77	-9.5	N14°W
"	"	E-62			167°59'						
			E-63		335°58'			-4.80			
				E-64	(167°59')	(-0-)	21.53		-1.05	-	N28°W
"	"	E-62			65°00'						
			E-63		130°00'			-4.80			
				E-63B	(65°00')	(-0-)	21.38		-0.97	5.45	S50°W
"	"	E-63			112°34'	+0°49'					
			E-63B		225°07'	+0°49'		-1.08			
				E-63C	(112°33.5')	(+0°49')	13.59		-0-	4.4	S18E
"	"	E-62			285°09'	+2°40'					
			E-63		210°18'	+2°39'		-4.8			
				E-63A	(285°09')	(+2°39.5')	17.61		-0.26	—	N89°E
3/31/77	TJF JWW	E-63			157°10'	-0°32'					
			E-64		314°19'	-0°33'		-1.44			
				E-65	(157°9.5')	(-0°32.5')	19.12		-0-	-	N50.5°W
"	"	E-63			224°41'						
			E-64		89°22'			-1.44			
				E-64A	(224°41')	(-0-)	26.51		-2.76	-	N16.5°E
"	"	E-63			83°22'						
			E-64		166°44'			-1.44			
				E-64C	(83°22')	(-0-)	14.07		-1.92	-	S56°W
"	"	E-64			214°28'	-7°43'					
			E-64A		68°56'	-7°43'		-1.98			
				E-64B	(214°28')	(-7°43')	15.94		-0-	-	N5°E
"	"	E-64			211°38'						
			E-64C		63°16'			-1.35			
				E-64D	(211°38')	(-0-)	31.55		-0.08	-	S82°W
"	"	E-64			142°10'	+41°49'					
			E-64C		284°19'	+41°49'		-1.35			
				E-64C1	(142°9.5')	(+41°49')	12.68		-0-	-5.7	S19.5°W
					( )	( )					
					( )	( )					
					( )	( )					

REMARKS AND SKETCHES



SURVEY DATA - OFFICE COMPUTATIONS

SURVEY NO. 2

SHEET 4 OF _____

BRG. OF INITIAL BACKSIGHT FROM:
( ) PRECEDING PAGE. THIS SURVEY
( ) SURVEY NO. _____. SHEET _____

$S2°14\frac{1}{2}'E$

COORDINATES AND ELEV. OF INITIAL INST. STA. FROM
( ) PRECEDING PAGE. THIS SURVEY
( ) SURVEY NO. _____. SHEET _____

(E-61) 2077.29 1304.24

COMP. BY	COSINE VERT. ANGLE	HORIZ. DISTANCE	SINE VERT. ANGLE	VERT. DIST.	BEARING	COSINE BEARING	SINE BEARING	LATITUDE OF FS	DEPARTURE OF FS	ELEVATION OF FS
								+ 9 72	+ 22 56	+
								−	−	−
		24.56		-0-	$N66°41\frac{1}{2}'E$		(E-61A)	2087 01	1326 80	
							(E-61)	2077.29 15 19	1304.24	+
								−	− 6 34	−
		16.46 ✓		+3.66	$N22°40'W$			2092 48	1297 90	
								+ 16 13	+	+
								−	− 4 60	−
		16.77		-0-	$N15°54\frac{1}{2}'W$			2108 61	1293 30	
								+ 19 02	+	+
								−	− 10 08	−
		21.53		-0-	$N27°55\frac{1}{2}'W$		(E-64)	2127 63	1283 22	
								+	+	+
								−	−	−
		21.38		-0-						
								+	+	+
								−	−	−
		13.59 ✓		+.19						
								+	+	+
								−	−	−
		17.59 ✓		+.82						
					BS $S27°55\frac{1}{2}'E$		(E-64)	+2127.63 17 09	+1283.22	+
								−	− 14 81	−
		19.12 ✓		-.18	$N50°46'W$		(E-65)	2139 72	1268 41	
					BS $S27°55\frac{1}{2}'E$		(E-64)	+2127.63	+1283.22	+
								− 25 38	− 7 64	−
		26.51		-0-	$N16°45\frac{1}{2}'E$		(E-64A)	2153 01	1290 86	
					BS $S27°55\frac{1}{2}'E$		(E-64)	+2127.63	+1283.22	+
								− 7 98	− 11 59	−
		14.07		-0-	$S55°26\frac{1}{2}'W$		(E-64C)	2119 65	1271 63	
					BS $S16°45\frac{1}{2}'W$		(E-64A)	+2153.01	+1290.86	+
								− 9 89	− 12 32	−
		15.80 ✓		-2.14	$N51°13\frac{1}{2}'E$		(E-64B)	2162 90	1303 18	
					BS $N55°26\frac{1}{2}'E$		(E-64C)	+2119.65	+1271.63	+
								− 1 61	− 31 51	−
		31.55		-0-	$S87°04\frac{1}{2}'W$		(E-64D)	2118 04	1240 12	
					BS $N55°26\frac{1}{2}'E$		(E-64C)	+2119.65	+1271.63	+
								− 9 01	− 2 86	−
		9.45 ✓		+8.45	$S17°36'W$			2110 64	1268 77	
								+	+	+
								−	−	−
								+	+	+
								−	−	−
								+	+	+
								−	−	−

SURVEY DATA - FIELD SHEET

SURVEY NO. 2 SHEET 5 OF

MINE R of B LEVEL E WORKING (S)

Inst 28169 "Buff"

DATE	ENG. & HELPER	BS	INST	FS	HORIZONTAL ANGLE	VERTICAL ANGLE	SLOPE DISTANCE	HI	HP	PT. TO RAIL	COMPASS BEARING
3/23/77	TJF JWW	E-6			178°00'	-1°30'					
			E-611		356°01'	-1°30'		-1.12			
				E-612	(178°00.5')	(-1°30')	16.55		-2.42	-	
"	"	E-611			239°09'						
			E-612		118°19'			-1.21			
				E-613	(239°09.5')	(-0-)	6.49		-0.71	-	
3/25/77	TJF JWW	E-612			195°04'						
			E-613		30°08'			-0.72			
				E-614	(195°04')	(-0-)	11.76		-1.72	-	
"	"	E-612			279°18'						
			E613		198°35'			-0.72			
				E-615	(279°17.5')	(-0-)	12.45		-1.34	-	
"	"	E-612			66°15'						
			E-613		132°30'			-0.72			
				E-616	(66°15')	(-0-)	20.61		-0.44	-	
					( )	( )					
					( )	( )					
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					( )	( )					
					( )	( )					
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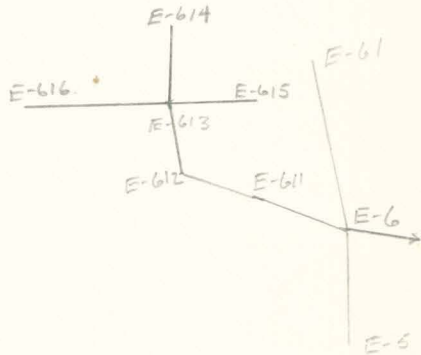
					( )	( )					

REMARKS AND SKETCHES

Horz Vert
16.54 -.43



SURVEY DATA - OFFICE COMPUTATIONS

E-611 2328.65

SURVEY NO. 2

SHEET 35 OF

BRG. OF INITIAL BACKSIGHT FROM:
( ) PRECEDING PAGE, THIS SURVEY
(X) SURVEY NO. 2, SHEET 1

S 60° 31½ E

COORDINATES AND ELEV. OF INITIAL INST. STA. FROM:
( ) PRECEDING PAGE, THIS SURVEY
(X) SURVEY NO. 2, SHEET 1

1000.36	5297.28	2328.65

COMP. BY	COSINE VERT. ANGLE	HORIZ. DISTANCE	SINE VERT. ANGLE	VERT. DIST.	BEARING	COSINE BEARING	SINE BEARING	LATITUDE OF FS	DEPARTURE OF FS	ELEVATION OF FS
								+ 7 63	+	+ 0 8
			✓					−	− 14 67	−
		16.54		-0.43	N62°31'W	✓		1007 99	5282 61	2329 5
								+ 6 48	+	+
								−	− 38	− 0 5
		6.49		-	N3°21'30"W	✓		1014 47	5282 23	2329 0
								+ 11 52	+ 2 39	+ 1 0
								−	−	−
		11.76		-	N 11°42.5'E	✓		1025 99	5284 62	2330 0
								+	+ 12 38	+ 6
								− 1 29	−	−
		12.45		-	S 84°04'E	✓		1013 18	5294 61	2329 6
								+	+	+
								− 9 39	− 18 35	− 2
		20.61		-	S 62°53.5'W	✓		1005 08	5263 88	2328 7
								+	+	+
								−	−	−
								+	+	+
								−	−	−
								+	+	+
								−	−	−
								+	+	+
								−	−	−
								+	+	+
								−	−	−
								+	+	+
								−	−	−
								+	+	+
								−	−	−
								+	+	+
								−	−	−
								+	+	+
								−	−	−

SURVEY DATA - FIELD SHEET

SURVEY NO. 2 SHEET 6 OF _____

MINE Queen of Bronze LEVEL E WORKING (S) _____

INST 28169 "Buff"

DATE	ENG. & HELPER	BS	INST	FS	HORIZONTAL ANGLE	VERTICAL ANGLE	SLOPE DISTANCE	HI	HP	PT. TO RAIL	COMPASS BEARING
3/30/77	TSF JWW	E-61			184°23'	+1°17'					
			E-61A		8°45'	+1°17'		-2.52			
				E-61B	(184°22.5')	(+1°17')	61.95		-0-	-	N70.5E
"	"	E-61			278°44	+46°02'					
			E-61A		197°27	+46°03'		-2.52			
				E-61A1	(278°43.5')	(+46°02.5')	20.14		-0-		S13.5'E
		E-61A			242°46'						
			E-61B		125°32'			-1.26			
				E-61C	(242°46')	(-0-)	11.36		-2.38	-	S46°E
					( )	( )					
					( )	( )					
					( )	( )					
					( )	( )					
					( )	( )					
					( )	( )					
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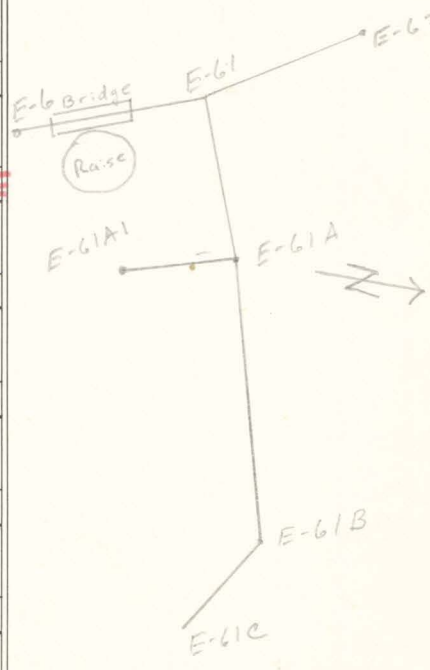
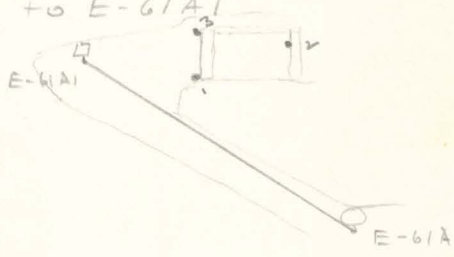
					( )	( )					
					( )	( )					

REMARKS AND SKETCHES



Set 3 spads on line established by E-61A to E-61A1



SURVEY DATA - FIELD SHEET

IRVEY NO. 2 SHEET 6 OF

NE Queen of Bronze LEVEL E WORKING (S)

INST 28169 "Buff"

E	ENG. & HELPER	BS	INST	FS	HORIZONTAL ANGLE	VERTICAL ANGLE	SLOPE DISTANCE	HI	HP	PT. TO RAIL	COMPASS BEARING
/77	TSF	E-61			184°23'	+1°17'					
	SWW		E-61A		8°45'	+1°17'		-2.52			
				E-61B	(184°22.5')	(+1°17')	61.95		-0-	-	N70.5E
	"	E-61			278°44	+46°02'					
			E-61A		197°27	+46°03'		-2.52			
				E-61A1	(278°43.5')	(+46°02.5')	20.14		-0-		S13.5°E
		E-61A			242°46'						
			E-61B		125°32'			-1.26			
				E-61C	(242°46')	(-0-)	11.36		-2.38	-	S46°E
					( )	( )					
					( )	( )					
					( )	( )					
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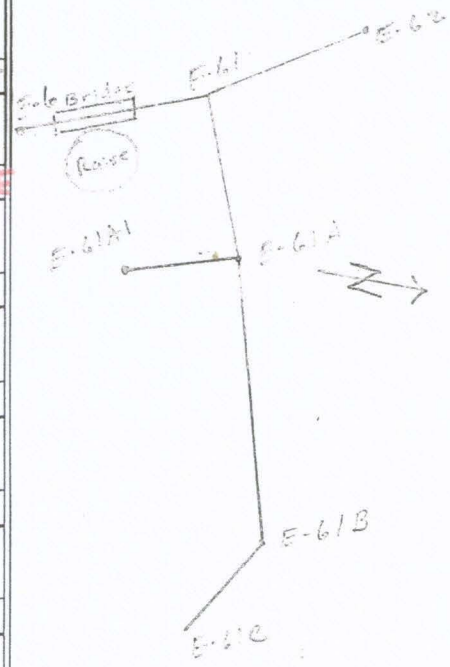
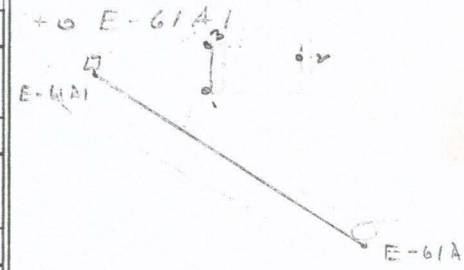
REMARKS AND SKETCHES



Set 3 spads on line established by E-61A to E-61A1



SURVEY DATA - OFFICE COMPUTATIONS

E-61A = 2330.00

SURVEY NO. 2

SHEET 6 OF _____

BRG. OF INITIAL BACKSIGHT FROM:
(X) PRECEDING PAGE, THIS SURVEY
( ) SURVEY NO. _____, SHEET _____
S 69°14'30"W

COORDINATES AND ELEV. OF INITIAL INST. STA. FROM:
(X) PRECEDING PAGE, THIS SURVEY
( ) SURVEY NO. _____, SHEET _____

1043.18	5330.34	2330.00

COMP. BY	COSINE VERT. ANGLE	HORIZ. DISTANCE	SINE VERT. ANGLE	VERT. DIST.	BEARING	COSINE BEARING	SINE BEARING	LATITUDE OF FS	DEPARTURE OF FS	ELEVATION OF FS
								+ 17 47	+ 59 42	+
								−	−	− 1 1
		61.93		+1.39	N 73°37'E			1060 65	5389 76	2328 8
								+	+ 2 91	+ 11 9
								− 13 67	−	−
		13.98		+14.50	S 12°02'E			1029 51	5333 25	2341 9
								+	+ 7 84	+ 1 1
								− 8 22	−	−
		11.36		-0-	S 43°37'E			1052 43	5397 60	2329 9
								+	+	+
								−	−	−
								+	+	+
								−	−	−
								+	+	+
								−	−	−
								+	+	+
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								−	−	−
								+	+	+
								−	−	−

Tests, Reports & Maps

Queen of Bronze Group of Mines

Takilma District
Josephine County - Oregon

Contributors: Walter A Richelsen
George C Heikes
Cushing Moore
Philip J. Shenon
Dr. Francis Wells
American Smelting & Refining Co.
American Cyanamid
Mace Smelter Co.
Denver Equipment Co.
U. S. Bureau of Mines
Oregon State Dept. Geology
Roy E. Clark
C. G. Stowell
and others.

Please return to:
L. R. Davenport
Waite Minerals, Inc.
P. O. Box 2 202 Doyle Bldg., Grants Pass,
Oregon

Suggestions of Mr. George C. Heikes, on Queen of Bronze and Associated Properties

According to Mr. George C. Heikes, consulting mining geologist, who visited the area on April 6 and 7, 1956, "there appears to be a NE x N trending zone extending from the Cowboy Extension to the lime quarry beyond the North End of the Queen of Bronze, a distance of approximately 4 miles. In this zone appears to be some mineralized greenstones of unknown age, probably pre-Paleozoic, which were intruded by peridotite, probably during the Jurassic age. The area was subsequently mineralized and faulted.

Judging from the existing surface and underground workings, a systematic drilling of the greenstones might reveal a body of disseminated copper of commercial grade and size. Detailed underground geologic mapping should be made of the existing mine workings, with some sampling. This work would very likely reveal a definite pattern of mineralization which has not been understood by previous operators. The information obtained from this work would immeasurably reduce the cost of exploration and development and should help in locating some large commercial ore bodies.

There seems to be a good chance for an adequately financed and experienced organization to find some worthwhile ore deposits in this area.

In the event of discovery of a disseminated deposit, it would probably be amenable to find grinding and selective flotation; whereas the gold-cobalt-nickel-copper bearing fine grained pyrite would be amenable to the production of matte by direct smelting.

The kind of regional geologic study indicated above is possible for the first time in the history of the area. This is due to the fact that all the known deposits or existing mines are in the hands of one individual, Major E. R. Waite, President of Waite Minerals, Inc. These existing mines are:

Patented - North End Queen of Bronze, South End Queen of Bronze, Mable, Lyttle, East Cowboy, Cowboy and Waldo.

Unpatented - Lilly and Cowboy Extension."

Walter A. Richelsen
Consulting Mining Engineer
709 Central Building
Seattle 4, Washington

REPORT ON THE QUEEN OF BRONZE PROPERTY

TAKILMA DISTRICT
JOSEPHINE COUNTY - OREGON

Seattle, Washington
November 7, 1949

by: W. A. Richelsen

ASSAY RETURNS

Queen of Bronze Property
Takilma District
Josephine County-Oregon

Sample No.	Ounces per ton Au.	Ag.	% Cu.	Remarks
	Samples in May, 1949			
49-17	Tr.	Nil	1.28	Open Cut-North of pit, chalcopyrite and carbonates.
49-19	Tr.	Nil	0.51	Open Pit-North End-Altered andesite and gs. -pyrite with some chp.
49-20	0.27	Nil	1.10	Open Pit- Opposite Sta. 10 andesite some chp. and cb.
49-21	Tr.	Nil	1.48	Surface showing-Sta. 9, mostly carbonate copper.
49-22	0.08	Nil	6.54	B Tunnel-Dump---- chalcopyrite and pyrrhotite - pile on dump.
49-23	0.10	0.22	6.78	C Tunnel-Dump-hi- gr. - pyrite with chalcopyrite.
49-24	0.08	Nil	4.13	C Tunnel-Dump- between portal and ore chute-oxidized pyrr. and chp.
49-25	0.04	0.12	2.14	Big Dump- above D Tunnel-grab sample 20 feet along cat trail pyrrhotite and chalcopyrite.
	Samples in October, 1949			
				D Tunnel
49-58	0.08	Nil	5.17	4 E. Drift-Hampshire stope chp. with pyrr.
49-59	0.08	0.56	8.34	4 E Dr.-Incline raise-Hampshire stope, 3' ore-chp. pyr.
49-60	0.06	0.14	5.58	4 W. Dr.-Hampshire stope 1' ore-chp. and pyrr.
49-61	Tr.	Nil.	2.28	3 E. Drift- No. 6 raise-ore in back about 2'
				C Tunnel
49-62	Tr.	Nil.	0.32	Back of small stope-Stevens X-Cut apparently pinched out.
				A Tunnel
49-63	Tr.	Nil.	1.61	1 E. Drift- 3rd one in-grab sample from muck in foot-oxidized-chp. and malachite.
49-64	0.04	Nil	4.24	1 N. X-Cut at turn-2" stringer in foot-chp. and pyrr.
49-65	Tr.	Nil	1.41	1 N. X-Cut grab sample- ore from caved chute-oxidized-from Open Pit above.

ESTIMATED SMELTER COSTS

Queen of Bronze Property
Takilma District
Josephine County-Oregon

Smelter Construction	Weight	Amount
Proposal of July 23, 1949-Equipment f.o.b.Denver	87,000	$26,000
Proposal of October 5, 1949 " " "	10,300	5,800
Total Pounds	97,300	
Truck freight - Denver to Takilma, Oregon at $2.90 per cwt.		2,842
Total Cost f.o.b. Takilma		35,542

Additional Equipment - Supervision	
2 Double Snyder samplers- $250 each- 1 trommel at $1,000	$ 1,500
Sample Grinder- $250, Wet Laboratory at $1,000	1,350
Steel smelter building erected	3,500
Installation of machinery- 50 tons at $60 per ton	3,000
Foundations- 2 furnaces, blowers and sintering hearth	1,000
Electrical wiring and water piping	2,000
Flues and 60 foot stack, 5' diameter	2,000
Concrete dust chamber, 10' x 12' x 70' at $20 per cu. yd.	1,860
Water Tank- 15,000 gallons	2,420
Reconditioned crusher, $750- 20 H.P. Motor and starter, $450	1,200
Supervision-Superintendent 3 months at $750 per month	2,250
Bunkers- 100 ton capacity at mine dump and 500 ton capacity at smelter, Straight line bins with partitions Fir lumber at $20 per M, at site	2,000
Total	$ 24,080

Miscellaneous	
Rehabilitation of underground workings	$ 5,000
Power line extension -California Oregon Power Co.	3,800
Surface jig-back slope tramway-dump to smelter bins 1,600 feet at $3.75 per foot	6,000
Total	$ 14,800
Grand Total	$ 74,422

Estimated Daily Operating Costs

Treating 100 tons of ore daily, may include 90 tons dump ore and 10 tons mine ore.

One shift daily, delivering ore with Cat or Drag line into grizzly; trommel screen and crusher, then to storage bins. Then by 1,600 foot surface jig-back tram to smelter.

ESTIMATED SMELTER COSTS

Queen of Bronze Property
Takilma District
Josephine County - Oregon

Estimated Daily Operating Costs - Continued

	Daily	Cost Per Ton	Per Unit
Ore Handling			
Labor			
1 Cat Operator and fuel	$ 20.00		
1 Tram man- one shift	12.00		
	$ 32.00	$0.32	0.32
Smelting and Rematting			
Labor and Supervision:			
2 feeder men per shift- 6 men at $10 each	60.00		
1 furnaceman " " 3 " 1.75 per hr.	42.00		
1 " helper " " 3 " $10 each	30.00		
1 utility man " " 3 " $10 "	30.00		
1 Mech-elect. " " 1 " 1.65 per hr	13.00		
1 sinter man " " 3 " $10 each	30.00		
	$205.00	2.05	
Smelter Superintendent at $500 per mo. per day	17.00		
Assayer and chemist $300 " " " "	10.00		
	$ 27.00	0.27	
Coke:			
Heads- 20% sulphur-5% coke			
5 1/2 tons at $25.00 per ton-per charge	138.00		
Rematting furnace at 3/4 ton at $25.00 per ton	19.00		
	$157.00	1.57	
Limestone: 10 tons at $2.50 per ton	25.00	0.25	
Power:			
Sinter fan 10 h.p.			
Turbo blower 40 h.p.			
Connerville blower 10 h.p.			
Crusher 7 h.p.			
Electric hoist 4 h.p.			
Lighting 3 h.p.			
Trommel 10 h.p.			
1 pump -			
84 h.p. 63 k/w 3 shifts at 1.5¢ per k.w.h.	$ 23.00	0.23	4.37
Marketing			
Takilma - Grants Pass, Oregon			
Trucking 3 1/2 tons matte at $3.50 per ton	12.25		
Freight-Grants Pass, Ore. to Tacoma, Wash.-$6.57	23.63		
Refining- $9.00 per ton	31.50		
	$67.38	0.67	0.67
Total Cost			

ORE PRODUCTION ANALYSIS

Queen of Bronze Property
Takilma District
Josephine County-Oregon

Production at 100 tons per day, with smelter heads maintained at 3% copper. May be entirely dump ores or combined underground and dump ores. Following costs based on price of copper paid for by smelter on 18 ½ ¢ copper less smelter deduction of 2 ½ ¢ or 16¢ net per pound. Gold at $35.00 per ounce, silver at 90.5¢ per ounce. Recovery at 95%. Taxes are not included.

Dump Ores

100 tons ore at 3% copper, 0.04 ounces of gold, and 0.04 ounces of silver

Copper	$912
Gold	133
Silver	3
Total Gross Return - per day	$1,048

Less operating expense-		
Ore transportation, smelting and marketing- per ton	$5.36	
Mining- dump ores	1.15	
	$6.51	
Cost per 100 tons- per day		$ 651
Net returns per day		397
Net returns per month		11,910
Net returns per ton		$ 3.97
Estimated yearly production-less delays		35,000 tons
Net returns per year at 35,000 tons		$138,900
Tonnage- dumps alone		111,000 tons
Estimated life of dumps- combined		3.2 years
Net returns- dump ores- total		$440,000

Tonnage- Queen of Bronze dump	46,000 tons
Estimated life of dump at 35,000 tons per year	1.3 years
Net returns- Queen of Bronze dump	$182,000
Capital investment- plus 7 months interest	$ 77,024
Net returns per month	11,910
Months to retire capital investment and interest	6.5 months

ORE PRODUCTION ANALYSIS

Queen of Bronze Property
Takilma District
Josephine County- Oregon

Underground Ores

100 tons of ore at 7% copper with 0.04 ounces of gold and 0.04 ounces of silver per ton. Net price of copper 16 ¢ with recovery at 95%. Taxes not included.

	Copper	$2,125
	Gold	133
	Silver	3
Total Gross Return- Per day		$2,261

Less operating expense-		
Ore transportation, smelting and marketing- per ton	$ 5.36	
Mining costs per ton	4.70	
	$10.06	

Cost per 100 tons- per day	$ 1,006	
Net returns per day	1,255	
Net returns per month	$ 37,650	
Net returns per ton	$ 12.55	
Estimated yearly production- less delays	35,000	tons
Net returns per year at 35,000 tons	$439,000	
Tonnage underground- Queen of Bronze	90,000	tons
Estimated life- underground ore	2.6	years
Net return- underground ores- total	$ 1,129,000	
Cost per ton- less gold and silver values	$ 8.70	
Minimum or cut-off copper content- underground ores		
at 18 ½ ¢ copper	2.9%	
at 17 ½ ¢ copper	3.1%	
at 16 ½ ¢ copper	3.3%	

Combined Ores: Dump and Underground

Estimated yearly production less delays	35,000	tons
Total tonnage- combined ores	201,000	
Estimated life- combined ores	5.8	years
Net returns- combined ores	$ 1,518,000	

Net returns on the combined ores is based on the combined profits in each individual operation. It is not considered on the basis of a 5% copper content as shown in the estimated reserves.

TAX ANALYSIS

Queen of Bronze Property
Takilma District
Josephine County- Oregon

The following tax analysis is based on the Corporation Income Tax which includes both the Normal and Surtax. It shows the inadequacy of our present tax laws for encouraging venture capital to invest in mining properties. It compares unfavorably to the Canadian Tax Laws which impose no tax for a period of 3 years on mining properties in the stage of development. This allows the company to build up a small surplus to meet exploration risks.

Tax Analysis

Capital investment- to retire in 1 year		$74,422
Plus- Interest at 6% for 1 year		4,460
	Total capital charge	$78,882
Net returns per year- dump ores		$ 138,900
Less- total capital charge		78,882
Net return- for first year- dump ores		60,018
Over $50,000 and taxable for total of 38% and this will apply on total net profit.		
Total net profit- dump ores		440,000
Less total capital charge	$78,882	
" 2.78% proportionate royalty-Waldo ore	12,250	
	91,132	91,132
Total net return- dump ores		348,868
Less 40% tax - 38% Federal- 2% State		139,547
Total Net Profit- after taxes- dump ores		209,321

Net return- underground ores- Queen of Bronze	$ 1,129,000
Less 40% tax - 38% Federal- 2% State	451,600
Net profit- underground ores- after taxes	677,400
Total net profit-combined ores- after taxes	886,721

Taxes such as Social Security and Employment Security and Workman's Compensation, Pension and Insurance benefits have not been considered as this depends on the personnel involved in the operation.

A 15% depletion charge is recommended in the first year of operation. A Limited Partnership is also recommended rather than a corporation.

The legal description of the Queen of Bronze Group of Mines, known as the North and South Workings of the Queen of Bronze, The Cowboy, East Cowboy, Mabel, and the Lyttle Mines total 532 acres patented, and are designated as:

Parcel I: The Northwest Quarter of Section 36, Township 40 South, Range 8 West of the Willamette Meridian.

Parcel II: The West half of the Northeast Quarter and the East Half of the Northwest Quarter of Section 35, Township 40 South, Range 8 West of the Willamette Meridian. Less the Northwest Qurater of the Southeast Quarter of Section 35, Township 40 South, Range 8 West of Willamette Meridian. ALSO less and excepting the following: Begin at the Northwest corner of the East half of the Northwest Quarter of Section 35, Township 40 South, Range 8 West of the Willamette Meridian; thense South 207.4 feet to corner No. 2; thence South 89° 42' East 210 feet to corner No. 3; thence North 207.4 feet to corner No. 4; thence North 89° 42' West, 210 feet to place of beginning, as shown in Deed recorded in Volume 120, page 120, Deed Records of Josephine County, Oregon.

Parcel III: Mineral Lot 416 according to the official Government survey as described in Mineral Patent from the United States of America, recorded in Volume 24, page 272, of Josephine County Deed Records.

Parcel IV: Mineral Lot 522 according to the official Government survey as described in Mineral Patent from the United States of America, recorded in Volume 25, page 346-49 of Josephine County Deed Records;

with right of ingress and egress to; right of ownership and removal of all minerals contained therein; and the use of as much of the surface as is necessary for such removal being vested in Waite Minerals, Inc.

The vegetative surface rights belong to a local timber company.

AMERICAN CYANAMID COMPANY

Ore Dressing Laboratory

T. I. 242
Problem 463

December 1, 1947

Heavy-Media Separation Tests on Gold-Copper Ore
From Queen of Bronze Mine Submitted by Mr. E. R. Waite

Introduction:

Mr. E. R. Waite of Grants Pass, Oregon shipped a 500 pound sample of dump ore from the Queen of Bronze Mine, Takilma, Oregon, to our Mineral Dressing Laboratory for Heavy-Media Separation tests. Mr. Waite advised that a substantial tonnage was available but that it was too low grade to ship to a smelter. It was his thought that concentration by Heavy-Media would produce a shipping grade concentrate.

Object of Investigation:

We were asked to determine the amenability of the sample of dump ore to concentration by Heavy-Media Separation.

Description of Shipment:

Four kegs having a gross weight of 500 pounds were received at the Mineral Dressing Laboratory in good condition on October 7, 1947. The sample consisted of pices ranging in size from 6" to fines.

Preparation of Sample:

The contents of all four kegs were screened in 1". The screen oversize was crushed to pass 1" and the entire sample was then screened on 10 mesh. The minus 1" plus 10 mesh ore was mixed and representative 50 pound charges were cut out for testing. The minus 10 mesh fraction was weighed and sampled for assay. The above treatment resulted in the following weight distribution.

AMERICAN CYANAMID COMPANY

Mineral Dressing Laboratory

Table I

Weight Distribution

Prepared Ore

Product	% Weight
Original	100.00
-1" / 10 mesh	89.95
-10 mesh	10.05

Heavy-Media Test Procedure:

Each 50 pound charge of -1" ∕ 10 mesh ore was wet screened on 10 mesh to remove adhering slime. The west minus 10 mesh fraction was dried, weighed and assayed. The -1" ∕ 10 mesh ore was then fed continuously at the rate of 50 pounds per hour to a 20" diameter Heavy-Media Separatory cone. Preliminary examination of the ore had indicated the optimum specific gravity of separation to be slightly below 3.00. The Heavy-Meidia Separatory cone was therefore charged with a medium consisting of a mixture of finely ground ferrosilicon and magnetite suspended in water. The specific gravity of the medium was adjusted to 3.00 and the feed was started.

The ore particles having a specific gravity lower than 3.00 floated on the surface of the medium and were discharged ofrom the cone over a weir. The heavier fraction of the ore sank and was discharged from the ore continuously by means of an air lift. Both products were drained free of medium on a vibrating screen. The drained medium was returned to the cone. The separated ore products were washed free of medium on the lower part of the screen. Each product was dried, weighed and assayed.

Separating mediums having specific gravities of 2.93 and 2.88 were employed in subsequent tests.

The metallurgical results of these tests are presented in Tables II, III and IV.

AMERICAN CYANAMID COMPANY
Ore Dressing Laboratory

Table No. II — Tech. Invest. No. 242-463

Test No. I — Sample -1"

PRODUCT	% WEIGHT In Size	% WEIGHT In Total	ASSAYS oz/ton Au	ASSAYS % Cu
Ore		100.00	0.044	2.41
Total -10 mesh		10.25	0.050	1.98 +
Cone Feed -1" / 10 mesh	100.00	89.75	0.043	2.46
Sink in 3.00	32.58	29.24	0.090	5.26 +
Float on 3.00	67.42	60.51	0.020	1.11

REMARKS:

PRODUCT	% DISTRIBUTION IN SIZE Au	IN SIZE Cu	IN TOTAL Au	IN TOTAL Cu
Ore			100.00	100.00
Total -10 mesh			11.78	8.41
Cone Feed -1" / 10 mesh	100.00	100.00	88.22	91.59
Sink in 3.00	68.51	69.60	60.44	63.75
Float in 3.00	31.49	30.40	27.78	27.84

AMERICAN CYANAMID COMPANY
Ore Dressing Laboratory

Table No. III — Tech. Invest. No. 242-463

Test No. 2 — Sample -1"

PRODUCT	% WEIGHT In Size	% WEIGHT In Total	ASSAYS oz/ton Au	ASSAYS % Cu
Ore		100.00	0.050	2.54
Total - 10 mesh		10.25	0.050	1.98
Cone Feed -1" / 10 mesh	100.00	89.75	0.050	2.61
Sink in 2.93	53.41	47.94	0.076	4.11
Float in 2.93	46.59	41.91	0.020	0.88

REMARKS:

PRODUCT	% DISTRIBUTION In Size Au	In Size Cu	In Total Au	In Total Cu
Ore			100.00	100.00
Total -10 mesh			10.28	7.99
Cone Feed -1" / 10 mesh	100.00	100.00	89.72	92.01
Sink in 2.93	81.34	84.27	72.98	77.53
Float in 2.93	18.66	15.73	16.74	14.48

AMERICAN CYANAMID COMPANY
Ore Dressing Laboratory

Table No. IV — Tech. Invest. No. 242-463

Test No. 3 — Sample -1"

PRODUCT	% WEIGHT In Size	% WEIGHT In Total	ASSAYS oz/ton Au	ASSAYS % Cu
Ore		100.00	0.045	2.30
Total -10 mesh		10.25	0.050	1.98
Cone Feed -1" ∕ 10 mesh	100.00	89.75	0.044	2.33
Sink in 2.88	65.23	58.54	0.060	3.18
Float in 2.88	34.77	31.21	0.015	0.74

REMARKS:

PRODUCT	% DISTRIBUTION In Size Au	In Size Cu	In Total Au	In Total Cu
Ore			100.00	100.00
Total -10 mesh			11.42	8.84
Cone Feed 1-" ∕ 10 mesh	100.00	100.00	88.58	91.16
Sink in 2.88	88.24	88.96	78.17	81.09
Float in 2.88	11.76	11.04	10.41	10.07

Discussion of Results:

The metallurgical results given in Tables II, III and IV show that as the specific gravity of separation was decreased the ratio of concentration decreased and metal recoveries increased.

The highest grade concentrate was made in Test 1 in which a medium having a specific gravity of 3.00 was employed. The sink product amounted to 20.24% weight of the original ore and assayed 0.09 oz. Au per ton and 5.26% Cu. Recoveries of 60.44% of the gold and 63.75% of the copper contained in original ore were made.

Conclusion:

Heavy-Media Separation tests conducted on a sample of dump ore from the Queen of Bronze Mine have shown the amenability of this material to concentration. The optimum specific gravity of separation will depend largely on the cost of marketing the concentrate. Heavy-Media Separation at 3.00 specific gravity produced the highest grade concentrate. If shipping costs are moderate the use of a lower specific gravity of separation might be indicated so as to obtain higher metal recoveries even though at the expense of concentrate grade.

AMERICAN CYANAMID COMPANY

R. A. Pickens
Mineral Dressing Laboratory

COPY

ESTIMATED COST OF OPERATION
with No. 5 matting furnace and No. 2 rematting furnace
FOR
WAITE MINERALS, INC., GRANTS PASS, OREGON

Smelting 100 tons ore & Re-matting 12 tons Matte

	Daily	Cost/T/Ore
Labor:		
6 furnace feeders @ 1.25 per hr.	60.00	
6 furnacemen helpers @ 1.25 per hr.	60.00	
3 furnacemen @ 1.65 per hr.	39.60	
1 crusherman @ 1.25 per hr.	10.00	
	$169.60	$ 1.696
Coke:		
10 tons for #5 furnace @ 21.35 per T.	213.35	2.133
0.75 tons for #2 furnace rematting @ 21.35	16.01	0.161
Flux:		
10 tons limestone @ 2.50 per ton for #5 furnace	25.00	0.250
4.25 tons limestone @ " " " " #2 "	10.62	0.106
*9 ton silica for #2 furnace	—	—
Power:		
40 HP motor on turbo blower —29.84 KW x 24 hrs. at 1½¢	10.74	0.107
10 HP motor small blower #2 furnace —7.46KW x 24 hrs. at 1½¢	2.68	0.026
15 HP motor on crusher 8 hrs. — 11.19KW x 8 hrs. @ 1½¢	1.34	0.013
4 HP motor on elevator — 3KW x 6 hrs. @ 1½¢	0.27	0.003
Supervision: @ $450.00	15.00	0.150
Chemist $300.00	10.00	0.100
	$474.61	$4.745

MARKETING

Haul 3.25 tons per day to Grants Pass @ 3.50/T	11.375	0.114
Freight to Tacoma $7/T (estimated x 3¼	22.750	0.227
Treatment $7/T (estimated) x 3¼	22.750	0.228
	$ 56.875	$0.569
		4.745
	Total	$5.314

* Marginal silica flux--no cost--some value (as per E.R.W. letter)

3.25 tons #2 matte per day to ship

THE MACE COMPANY
2763 Blake St.
Denver 5, Colorado

August 22, 1949

COPY

METALLURGICAL REPORT
to
WAITE MINERALS, INC. GRANTS PASS, OREGON

1958 lbs. ore was received for test purpose. The sample was crushed to 1½" size and a head sample taken. The moisture content was 3% -- leaving 1900 lbs. dry ore. The analysis of the ore was as follows:

Au oz per ton	Ag oz per ton	Cu %	SiO_2 %	CaO %	MgO %	Al_2O_3 %	S %	Fe %	Value per ton
.02	0.98	1.82	32.74	0.63	5.91	9.48	12.0	25.55	$7.79

Smelting Test: The ore was smelted direct with addition of limestone and coke as follows:

Furnace charge:

Ore 100 lbs.)
Limestone 10 lbs.)---19½ charges
Coke 11 lbs.)

Blast pressure 6 ozs. Slage very hot and liquid. The furnace products were:

	Weight	Au oz per ton	Ag oz per ton	Cu %	SiO_2 %	FeO %	MgO %	Al_2O_3 %	CaO %
Slag	1044 lbs.	0.01	0.28	0.12	50.0	22.67	3.6	10.52	10.52
Matte	158¼ lbs	0.31	9.69	11.23	-	-	-	-	-
Shaft bearings & crucible clean up	474 lbs.	0.03	0.67	1.39	-	-	-	-	-

Re-matting Test

The 158½ lbs. of matte was crushed to ¼"m mixed with 108½ lbs. of silica (98%) and sintered -- producing 246 lbs of sintered matte. This was smelted with addition of limestone at the furnace also the 447 lbs of barrings and crucible slag of smelting test and 10% coke as follows:

Sintered matte 100 lbs.)
Limestone 20 lbs.)_____ 2½ charges
120 lbs.)
Coke (10%) 12 lbs.)

Barrings & crucible clean up 100 lbs.)
Coke 10 lbs.)_____ 4-3/4 charges

The products of this test were:

COPY

	Au oz per ton	Ag	Cu. %	Value/ton
49 lbs. tapped matte				
11.9 lbs matte skim in crucible bottome	0.78	16.32	29.95	$147.56
60.9 lbs Total 31.2 Ratio by weight				
Tapped slag 447 lbs.				
Clean up of crucible 123 lbs.	0.01	1.49	0.40	

Extraction and Slag Losses

On the direct ore smelting test the copper recovery was 94.4% and on the re-matting test it was 97.7% or a combined recovery of 92.22%. The combined gold, silver and copper recovery on the two tests was 85.0% by values. Had the gold been .09 ozs and copper 2.9% instead of .02 ozs and 1.82%, the final recovery would have been over 95%, as the slag assay would be practically the same for the higher grade ore. The two slag assays indicate the true loss, hence the overall recovery is exceptionally good on such a low grade ore.

Ratio of Concentration

The ratio of concentration by weight in the smelting test was 11.6 into 1 and on the re-matting test was 2.7 into 1 or a combined ratio of 31.2 into 1 by weight.

Recommendations for Practice

The amount of matte (158½ lbs.) was insufficient for a standard re-matting test so it was performed in the test by sintering it with silica and smelting the sinter in the blast furnace. In practice the first matte will be charged direct to the No. 2 furnace with silicious ore flux and 3% coke.

Appended to the report is an estimated cost sheet treating 100 tons of crude ore per day in a No. 5 Mace matting furnace and concentrating the matte in a No. 2 Mace matting furnace.

Respectfully submitted

THE MACE COMPANY

(signed) BY J. McAuliffe
Chief Engineer

Denver, Colo.
August 23, 1949

1	Spencer turbo-blower with direct connected 10 HP motor	800	840
1	Fairbanks type S dial smelter scale	2,800	1,240
1	Electro-Lift hoist with 4 HP motor and steel platform elevator	2,000	1,000
	Total F. O. B. Denver	89,800	$ 38,580
	Freight rate to Grants Pass — $2.22 +/- 15% on 898 cwt.		2,290
	Approx. Cost-Grants Pass		$ 40,870

Delivery about 90 days from receipt of formal order.

Usual terms: 50% with order, balance S/D attached to B/L.

You could have everything ready to install the equipment when it arrived, i. e. the excavations, foundations, smelter building, water & electric lines, etc. and in that th way the smelter could be started up in a months time thereafter.

It is difficult to arrive at the total cost of the plant. We estimate erection of the machinery at $100 ton or say $5,000 for 50 tons, the rest of it would just be a guesstimate.

You mention the nickel & cobalt - especially at the Cowboy — These metals would all go to the matte.

Catalog Mace Smelters covering our line of sintering, smelting and refining equipment is enclosed as requested.

With kindest regards, we are,

Yours very truly,

Clem H. Mace
President

CHM:hac
Encl.

This was dump ore - N Q

METALLURGICAL REPORT
to
WAITE MINERALS, INC., GRANTS PASS, OREGON

1958 lbs. ore was received for test purpose. The sample was crushed to 1½" size and a head sample taken. The moisture content was 3% — leaving 1900 lbs. dry ore. The analysis of the ore was as follows:

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Smelting Test: The ore was smelted direct with addition of limestone and coke as follows:

Furnace charge:

Ore	100 lbs.)	
Limestone	10 lbs.)	—— 19½ charges
Coke	11 lbs.)	

Blast pressure 6 ozs. Slag very hot and liquid. The furnace products were:

	W Weight	Au oz per ton	Ag oz per ton	Cu %	SiO_2 %	FeO %	MgO %	Al_2O_3 %	CaO %
Slag	1044 lbs.	0.10	0.28	0.12	50.0	22.67	3.6	10.52	10.52
Matte	158½ lbs.	0.31	9.69	11.23	-	-	-	-	-
Shaft bearings & crucible clean up	474 lbs.	0.03	0.67	1.39	-	-	-	-	-

Re-matting Test

The 158½ lbs. of matte was crushed to ¼" mixed with 108½ lbs. of silica (98%) and sintered — producing 246 lbs. of sintered matte. This was smelted with addition of limestone at the furnace also the 447 lbs. of barrings and crucible slag of smelting test and 10% coke as follows:

Sintered matte	100 lbs.)	
Limestone	20 lbs.)	2½ charges
	120 lbs.)	
Coke (10%)	12 lbs.)	
Barrings & crucible clean up	100 lbs.)	
Coke	10 lbs.)	4-3/4 charges

The products of this test were:

COPY

	Au oz per ton	Ag	Cu %	Value/ton
49 lbs. tapped matte 11.9 lbs. matte skim in crucible bottom	0.78	16.32	29.95	$147.56
60.9 lbs Total 31.2 Ratio by weight				
Tapped slag 447 lbs. clean up of crucible 123 lbs.	0.01	1.49	0.40	

Extraction and Slag Losses

On the direct ore smelting test the copper recovery was 94.4% and on the re-matting test it was 97.7% or a combined recovery of 92.22%. The combined gold, silver and copper recovery on the two tests was 85.0% by values. Had the gold been .09 ozs and copper 2.9% instead of .02 ozs and 1.82%, the final recovery would have been over 95%, as the slag assay would be practically the same for the higher grade ore. The two slag assays indicate the true loss, hence the overall recovery is exceptionally good on such a low grade ore.

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Recommendations for Practice

The amount of matte (158½ lbs.) was insufficient for a standard re-matting test so it was performed in the test by sintering it with silica and smelting the sinter in the blast furnace. In practice the first matte will be charged direct to the No. 2 furnace with silicious ore flux and 3% coke.

Appended to the report is an estimated cost sheet treating 100 tons of crude ore per day in a No. 5 Mace matting furnace and concentrating the matte in a No. 2 Mace matting furnace.

Respectfully submitted

THE MACE COMPANY

(signed) By J. McAuliffe
Chief Engineer

Denver, Colo.
August 23, 1949

E-A

COPY

ESTIMATED COST OF OPERATION
with a No. 5 matting furnace and No. 2 rematting furnace
for
WAITE MINERALS, INC., GRANTS PASS, OREGON

Smelting 100 tons ore & Re-matting 12 tons Matte

		Daily	Cost/T/Ore
Labor:			
6 furnace feeders	@ 1.25 per hr.	60.00	
6 furnacemen helpers	@ 1.25 " "	60.00	
3 furnacemen	@ 1.65 " "	39.60	
1 crusherman	@ 1.25 " "	10.00	
		$169.60	$ 1.696
Coke:			
10 tons for #5 furnace at 21.35 per T.		213.35	2.133
0.75 tons for #2 furnace rematting		16.01	0.161
Flux:			
10 tons limestone at 2.50 per T-#5 furn.		25.00	0.250
4.25 tons limestone " " " #2 "		10.62	0.106
* 9 ton silica for #2 furnace		-	-
Power:			
40 HP motor on turbo blower - 29.84 KW x 24 hrs. at 1½¢		10.74	0.107
10 HP motor small blower #2 furnace - 7.46 KW x 24 hrs. at 1½¢		2.68	0.026
15 HP motor on crusher 8 hrs.---11.19 KW x 8 hrs. at 1½ ¢		1.34	0.013
4 HP motor on elevator --- 3 KW x 6 hrs. at 1½¢		0.27	0.003
Supervision:	at $450 per mo.	15.00	0.150
Chemist:	at $300 per mo.	10.00	0.100
		$474.61	$ 4.745

MARKETING

Haul 3.25 tons per day to Grants Pass at $3.50 /T	11.375	0.114
Freight to Tacoma $7/T (estimated) x 3¼	22.750	0.227
Treatment $7 /T (estimated) x 3¼	22.750	0.228
	$ 56.875	$ 5.689
		4.745
Total		$ 5.314

* Marginal Silica flux- no cost - some value(as per E.R.W. letter)

3.25 tons #2 matte per day to ship

THE MACE COMPANY
2763 Blake St.
Denver 5, Colorado

August 22, 1949

34

PROPOSED DIAMOND DRILLING AT THE COWBOY MINE, JOSEPHINE COUNTY, OREGON

September 1949

Exploration of the ore bearing shear zone as found in the Winze and in the adit and small stope 30 feet beneath the Winze (see diagram No. 1) at the Cowboy Mine, Josephine County, Oregon, should be attempted by a diamond drill hole as described below.

In the Winze and stope the mineralized shear zone trends N.20° - 30°E. and dips 65° - 75°E. Considerable shearing and some mineralization have been found on the downward extension of the zone where it is crossed by the Lower aAdit. The broken nature of the rocks in the shear zone and the hard dike material which occurs in and near the zone, limit the angle at which the zone can be penetrated by a drill hole. A diamond drill entering the zone at less than 70° would undoubedly be deflected if hard dike rock were encountered which was bounded by sheared and much softer serpentine. The drill hole, therefore, should be oriented so that the drill meets the plane of shearing at or near 90°.

To meet the limitations both of the drilling equipment available and of the rock types and structures that are present, the following drill site and hole orientation are recommended.

Location of the drill station is oin the Lower Adit 644 feet from the portal of the adit. (See diagram No. 1.) Measuring toward the portal fromm sta. "CC", which is located at the "Y" in the adit, the distance to the proposed drill station is 64 feet. From this station the drill hole should bear S 59° E. (see diagram No. 1) and be elevated 30° from the horizontal (see diagram No. 2). In obtaining the bearing of the drill hole it must be remembered that the surrounding rocks are noticeably magnetic. It is necessary, therefore, to turn an angle of 65° toward the east from the adit center line passing through the drill station and to sta. "CC".

The area explored by the drill hole lies approximately 100 feet above sta. "DD" on the Lower Adit. The mineralized shear zone, projected downward from the upper workings, should be found between 203 feet and 223 feet measured along the drill hole (see diagram No. 2.)

If the recommended exploration finds favorable showings of ore the lateral extent of the ore should be tested by drill holes parallel to the drill hole described above. (Drawn in pencil on diagram no. 1). Drill stations should be located along the Lower Adit at 32 feet and at 97 feet from sta. "CC" measured toward the portal. The holes should have the same bearing (S 59° E.) and the same elevation (30° above the horizontal) as the drill hole recommended above. Of these two subsequent holes the first should be meet the mineralized shear zone at from 190 to 210 feet and the second hole at from 216 to 237 feet.

THESE TESTS WERE ALL MADE SEVERAL YEARS AGO. PRESENT METHODS OF MINERAL RECOVERY WILL FAR SURPASS RESULTS OBTAINED AT THAT TIME.

IT WOULD SEEM THAT SERIOUS CONSIDERATION SHOULD BE GIVEN TO USING SINK*FLOAT ON THE LOW GRADE ORES PRIOR TO DIRECT SMELTING.

THE HIGH*GRADE WOULD ALSO BE GOING TO THE SMELTER * PREFERABLY ONE IN THIS AREA.

IN ADDITION TO THE QUEEN OF BRONZE GROUP THERE ARE MANY OTHER FINE PROPERTIES AWAITING THE INEVITABLE TIME WHEN A GOOD SMELTER IS ERECTED SOMEWHERE NEAR.

THE QUEEN OF BRONZE PROPERTY HAS A GOOD SITE FOR THIS PLANT, AS IT IS IN THE CENTER OF THIS LARGE, MINERALIZED COPPER*GOLD* SILVER*NICKEL*IRON*COBALT ZONE WHICH RUNS FOR MANY MILES THROUGH SOUTHERN OREGON AND NORTHERN CALIFORNIA.

The complete report may be examined by visitors to Waite Minerals, Inc. office ~~202 Doyle Bldg.~~ P.O. Box 2., Grants Pass.

QUEEN OF BRONZE GROUP

Phil Shenon (336:163-167) (Excerpts) August, 1930

Location and History:

The Queen of Bronze Group of Mines is in Township 40 S., Range 8 West, about 1½ miles by good road, east of Takilma, on a prominent ridge between Elder Creek and the East Fork of the Illinois River, at an altitude of about 2,400 feet. A road connects the mines with the town of Takilma.

Copper was discovered in the Takilma district in 1860, by a miner named Hawes, and the Queen of Bronze was discovered and located October 22, 1862, by P. Androit. Little development was done, however, until 1903, when C. L. Tutt and Associates of Colorado bought the property. Their company, known as the Takilma Smelting Company, erected a smelter of a semipiritic type in 1904. It had a capacy of 100 tons daily and operated more or less continously until 1910. More than 20,000 tons of ore, with an average copper content of 8½ per cent, was smelted and in 1907 the mattes from the smelted ores contained about 40 per cent of copper. Lessees shipped a few thousand tons of ore during the period from 1910 to 1916. In April of 1916, the property was sold to John Twohy, John Hampshire and others, with Hamsphire acting as trustee. They operated continuously, under the direction of Roy B. Clark, until 1919, when the price of copper became too low for profitable mining. During this period of three years, according to G. E. Stowell, mining engineer, the mine shipped 183 cars (9,992.3 tons) of ore averaging in metal content from 5.16 to 16.33 percent of copper and from 0.04 to 0.44 ounces of gold to the ton. * * * * * * *

Mineralogy:

The abundant hypogene sulphides are cobaltite, chalcopyrite, cubanite, spalerite and pyrrhotite. Chalcocite occurs as a supergene sulphide and malachite, cuprite, tenorite, hematite and limonite are the more common oxidation products. In order of abundance the gangue minerals are serpentine, calcite, quartz and epidote. With the exception of serpentine, the gangue minerals are not readily visible in hand specimens, although post-sulphide calcite is in some places evident along fracture surfaces. The microscope shows, however, that calcite consitutes a considerable part of the ore and of the wall rocks next to the ore.

The cobaltite resembles pyrite in hardness and crystal outline but differs from it in color. Although microchemical tests reveal considerable iron in the cobaltite, the crystal form and lack of anisotropism distinguish it from glaucodot (a cobalt-iron-arsenic sulphide.) Cubanite and chalcpyrite differ considerably in color and degree of anisotropism. Pyrrhotite resembles cubanite but it is readily distinguished from it in polished sections by a greater relief. Sphalerite is fairly abundant and is readily distinguished by its grey color. In addition, it almost everywhere contains oriented blebs of chalcopyrite or pyrrhotite.

The succession in the deposition of the sulphide minerals is the normal one as defined by Lindgren, despite the fact that repetition occurs. There appears, however, to be a reversal in the succession of the gangue minerals. According to Lindgren, the normal order of mineral deposition in dpposits of this general class is silicates, quartz, carbonates, and other ganuge minerals - cobaltite,

pyrrhotite, sphalerite and chalcopyrite. The order of formation as actually determined is serpentine, calcite, epidote, quartz, cobaltite, sphalerite, chalcopyrite, cubanite, pyrrhotite, sphalerite and calcite. The succession was thrice interrupted by fracturing - - once after the deposition of the gangue minerals, again after the formation of the cobaltite and again after the deposition of the sulphites, but before the deposition of the later calcite. Serpentine was formed before the deposition of the older calcite, because veinlets of this calcite clearly cut felted plates of the serpentine, thus illustrating that serpentinization had taken place - in part at least - before deposition of the ore. Epidote appears to have formed after the older calcite posibly in part from the reaction of hydrothermal solutions upon it. QUARTZ veinlets clearly cut the epidote, and veinlets of sulphides, in turn, cut all three of these gangue minerals. The sulphides have replaced calcite more readily than the other gangue minerals and in most places this differential replacement of calcite is noticeable. Cobaltite was the first sulphide mineral deposited. A period of fracturing followed and then later sulphides were introduced, for the most part along the fractures. Sphalerite is the first sulphide known to have formed after the cobaltite. If other sulphides preceded the sphalerite, the evidence of them is completely destroyed. Chalcopyrite and cubanite formed after the sphalerite and, where associated, they occure as blade-like intergrowths. Of the two, chalcopyrite is considerably more abundant. Pyrrhotite succeeded the cubanite and chalcopyrite. It occurs as irregular masses, as veinlets in or along grains of older minerals, as lentils in chalcopyrite and cubanite and as oriented blebs and laths in sphalerite. The blebs and laths were certainly form d by replacement along cleavage directions in the sphalerite, as they occur only where sphalerite is known to be replaced by pyrrhotite.

At other places, oreinted blebs of chalcopyrite are numerous, but blebs and laths of pyrrhotite are missing. Lentils of pyrrhotite cut outgrowths of cubanite and chalcopyrite at various angles and in some places are parallel to the intergrowths. Some of the lentils merge into irregular shaped masses; others terminate at grain boundaries and still others appear isolated within grains of cubanite and chalcppyrite. Veinlets of pyrrhotite cut chalcopyrite and are numerous along grain boundaries of sphalerite and of cubanite and chalcopyrite.

Some be veinlets clearly cut across twined crystals of chalcopyrite. The more massive bodies of pyrrhotite replace cobaltite, chalcopyrite and sphalerite, whereas the pyrrhotite in turn is replaced by a later generation of sphalerite. This sphalerite, the last of the sulphides to form, replaces both chalcopyrite an and pyrrhotite and masses and veinlets of it transverse the boundaries of the two minerals. Fracturing followed the deposition of the later sphalerite and at some time later, calcite was introduced along the fractures.

The proportions of the different sulphides vary greatly from place to place. In general, the shipping ore averages about 1¼ percent of copper, $1 of gold to the ton, a considerable amount of zince and a little silver. A sample by G. E. Stowell, mining engineer, as taken in the Rose stope across a lens measuring 3 feet by 10 feet, assayed 15.1 per cent copper and 0.08 ounce of silver to the ton. No analysis are known to have been made of the run-of-mine ore for cobalt, nickel, zinc, arsenic or the platinum group. A partial analysis of one of the boulders containing no visible cobaltite, made by E. T. Erickson in the chemical laboratory of the U. S. Geological Survey, gave copper 18.65 per cent, zinc 0.24 per cent, cobalt 0.15 per cent, arsenic 0.11 per cent, nickel 0.11 per cent and chromium 0.04 per cent.

* * * * *

From "Contributions of to Economic Geology, 1908, Part I.," concerning the Mineral resources of the Grants Pass quadrangle and bordering districts, Oregon, by J. S. Diller and G. F. Kay

COPPER MINES

According to returns from the smelters, Oregon produced 545,859 pounds of copper in 1907. Of this amount, by far the larger part was derived from the ores of Josephine County and the remainder from the ores of Grant County. The entire output of Josephine County came from one mine, the Queen of Bronze, in the Waldo district. Some good ore had been taken from several small mines adjacent to this one, but operations on these mines have been suspended for some time. During the summer of 1908 very little work was being done at the Queen of Bronze mine, but preparations were being made for more extensive developments.

Queen of Bronze Mine

The Queen of Bronze mine is located in Sec. 36, T. 40 S., R 8 W., about 6 miles from Waldo and 2 miles from Takilma. It is owned by the Takilma Smelting Company, of Colorado Springs, Colorado, and was superintended by C. E. Tucker.

The rocks with which the ores are associated are gabbros, peridotites, and serpentine. These rocks are widespread in southwestern Oregon. The peridotite, which consists chiefly of enstatite and olivine, is in many places intruded in the gabbro; in other places the peridotite and gabbro appear to be products of differentiation of the same magma, there being no sharp line between those rocks which are olivine gabbros and gabbros, the last named containing chiefly plagioclase feldspar and pyroxene but no olivine. Locally the peridotite grades into pyroxenite. The compositions and field relationships of these rocks clearly indicate that they are closely related genetically. In some places the peridotites and related rocks are fairly free from decomposition, but over large areas they have been altered to serpentine, which usually has a greenish-yellow color and shows slickensided surfaces. At a few places the serpentine is itself undergoing decomposition, and one ofthe products of alteration being magnesite. The peridotites, gabbros and serpentine almost everywhere shows the effects of earth movements. They are fractured, fissured, and jointed, and in many localities are decidedly brecciated. The soil formed from these rocks is in general of a reddish color and supports a scant vegetation.

The outcrops of the ore deposits consist of gossan, the oxidized materials varying in depth from a few feet to more than 100 feet. The ore bodies have no definite form but occur as irregular masses in the gabbro, the peridotite and the serpentine. These masses or pockets of ore appear to have no definite relationships to one another but occur irregularly in the fractured and fissured rocks. Most of the ore bodies, however, that have been found on the Queen of Bronze and adjacent claims lie in a zone that extends for several miles in a north-south direction and has a width of less than 1 mile. The largest single body of unoxidized ore mined at the Queen of Bronze mine contained about 10,000 tons. Practically all of it came from a depth of less than 30 feet. Other masses of unoxidized ore have been taken from depths of about 100 feet. Although depths of about 300 feet have been reached in the workings, no important ore body has

been found below 125 feet. Several occurrences of slickensided ores were ob observed, and in some places the ore contains small beinlets of calcite.

The unoxidized ore is chalcopyrite, with which are associated pyrite, pyrrhotite, and subordinate amounts of quartz and calcite. In the low-grade ores pyrite and pyrrhotite are the most abundant minerals. In addition, to the copper content, the ores carry some gold and silver.

The oxidized ores are malachite, azurite, cuprite, tenorite (?), and chrysocolla. Of these the black ores, containing tenorite or chalcocite, are most abundant. Several thousand tons of oxidized ore has been mined. The average content in copper was more than 10 per cent. The lower limit of the oxidized ores is usually less than 90 feet from the surface, but some have been found at greater depths. In a small opening about 105 feet below the surface black oxide and small amounts of native copper were observed. The zone of oxidation is invariably deeper where the rocks have been serpentinized than where the country rocks are fairly fresh.

These ore bodies are apparently the result of precipitation fofrom mineral-bearing solutions which entered the rocks after they had been fractured and fissured by earth movements. Whether these solutions werd set free from cooling magmas as they solidified to form igneous rocks or whether they were of meteoric origin it is impossible to determine. Although dikes cutting the peridotite and gabbro were not observed in the vicinity of the mine, their presence in other areas of these rocks would suggest that the solutions may have been associated with the magmas from which the dikes were formed. In places in the serpentine below the zone of oxidation chalcopyrite with slickensided surfaces has frequently been found. The chalcopyrite appears to have been subjected to all the movements which accompanied the process of serpentinization. This indicates that the ores are older than the serpentine.

The mine is more than 20 miles from Grants Pass, which is the most accessible point on the railway. The only means of transportation between Grants Pass and the mine is by wagon; consequently the rates for hauling machinery, provisions, and other materials for the mine and coke for the smelter have been high. This fact has been unfavorable to the development of the property. The mine is situated on the slope of a ridge. The smelter is at the base of the slope 500 feet below the mine and 1½ miles from it. The ores, when taken from the workings, are trammed to bins, from which they are transferred to wagons and hauled to the smelter.

The equipment at the mine consists of three boilers, and air compressor, a hoist and two machine drills. The mine has been developed by tunnels, drifts and open cuts. The chief workings are related to two gossan-covered areas on the claim. The northern and more extensive workings are near the north boundary of the claim; the other workings are about 1,200 feet farther south.

The northern workings consist of two tunnels, from which considerable drifting has been done, and a large open pit. The upper tunnel, which is about 400 feet long, enters the west slope of the ridge and runs eastward

beneath an area of decomposed and breciated gabbro, in which are oxidized ores. At no place does this tunnel have a vertical depth of more than 90 feet from the surface. A In this tunnel and in drifts and winzes from it some large irregular-shaped masses of chalcopyrite, but practically no oxides, were obtained. From the tunnel an upraise was made to the oxidized ores. This upraise was then used as a chute. The oxidized ores were mined to the surface by overhand stoping, passed thru the chute, and carried out by tram thru the tunnel. Several thousand tons of oxidized ores were mined in this way, the large open pit thus formed having an area of about 120 by 120 by 80 feet. Where the tunnels and other workings were in the serpentine, great care had to be taken in timbering. The lower tunnel also enters the west slope and is about 190 feet below the upper tunnel. In it and in drifts from it more than 1,100 feet of work has been done. Only a small amount of ore was found in these workings.

The southern workings consist of a large open cut, a tunnel, which runs underneath this cut, and a 106 foot shaft. From the open cut about 10,000 tons of unoxidized ore, carrying about 7 per cent in copper, were taken. The zone of oxidation was only a few feet in depth. The ores mined wrer were passed thru a chute from the bottom of the pit to the tunnel and then trammed to the bins. From the tunnel and the shaft only a small amount of profitable ore was mined.

All the ores that have been mined have been smelted at the Takilma smelter, which is under the same ownership as the mine. The smelter is of the pyritic matte type and has a capacity of 100 tons a day. The charge used iwas about 1,500 pounds of ore, 350 pounds of limestone, and 200 pounds of coke. The limestone used had to be hauled about 2 miles. The matte from the ores smelted in 1907 contained about 40 percent of copper.

The Queen of Bronze property was acquired in 1903 by the present owner. Only a small amount of development had been done on the property previous to its acquisition. In all about 30 claims are owned by the ccompany. Including the cost of the smelter, more than $150,000 has been spent on the properties. Mr. Tutt, the president of the company, stated that more than 20,000 tons of ore had been smelted and that the average copper content had been about 6½ per cent. The gold content of the ores has been about worth more than $3 a ton, the silver content about 17 cents a ton. Ore was first smelted from this property in 1904. The greatest production was in 1907.

OTHER COPPER MINES

As already stated, there are several small mines adjacent to the Queen of Bronze mine, and owned by Mr. Tutt and his associates. Considerable development has been done on these properties, and from three of them - the Cow Boy, the Lyttle, and the Mabel - about 4,000 tons of ore has been smelted. The characters of the ores, their modes of occurrence, and their associations are similar to those of the Queen of Bronze mine.

Some distance to the northeast of the Queen of Bronze mine is a prospect owned by Doctor Spence. On this property considerable work has been done and some good ore has been found.

* * * *

This report was written in 1909.

REPORT OF ASSAY

ABBOT A. HANKS, INC.

ASSAYERS, CHEMISTS, ENGINEERS
624 SACRAMENTO STREET
SAN FRANCISCO. February 19, 1954

DEPOSITED BY White Minerals, Inc.
P. O. Box 2
Grants Pass, Oregon

MPLE OF PRODUCTS

abty. No.	Mark	GOLD, per ton of 2,000 lbs.		SILVER, per ton of 2,000 lbs.		Percentages
		Troy Ounces	Value @ $35.00 oz.	Troy Ounces	Value @ $	COPPER
64658	Shipping Ore	.09	3.15			18.51 %
59	Medium Grade	.06	2.10			15.35 %
60	Mill Feed					7.48 %

REPORT OF ASSAY

ABBOT A. HANKS, INC.

ASSAYERS, CHEMISTS, ENGINEERS
624 SACRAMENTO STREET
SAN FRANCISCO. April 29, 1954

DEPOSITED BY Waite Minerals, Inc.
P. O. Box 2
Grants Pass, Oregon

PLE OF ORE

bty. No.	Mark	GOLD, per ton of 2,000 lbs.		SILVER, per ton of 2,000 lbs.		Percentages
		Troy Ounces	Value @ $35.00 oz.	Troy Ounces	Value @ $	COPPER
7311	No. 1					1.54 %
12	" 2					20.46
13	" 3					6.55

ABBOT A. HANKS, INC.

Schedule No. 30

GENERAL SERVICE

APPLICABILITY

Applicable to single phase or three phase alternating current electric service, at such voltage as the Company may have available at the Customer's premises, for all purposes except those for which specific Schedules are provided.

TERRITORY

Within the entire territory served in Oregon by the Company.

RATES

The sum of the following Demand and Energy Charges:

	Per Meter Per Month
Demand Charge:	
First 20 kw of Billing Demand, per kw	No Charge
Excess kw of Billing Demand, per kw	60 ¢
Energy Charge:	
First 300 kwhr, per kwhr	4.0 ¢
Next 450 kwhr, per kwhr	3.5 ¢
Next 1,050 kwhr, per kwhr	3.0 ¢
Next 8,400 kwhr, per kwhr	1.8 ¢
Next 49,800 kwhr, per kwhr	0.9 ¢
Excess kwhr, per kwhr	0.6 ¢

Minimum Charge: The amount of the Demand Charge per meter per month, but in no event less than:

$1.80 per meter per month for single phase service
$7.20 per meter per month for three phase service

Determination of Billing Demand: The Billing Demand shall be the highest of the following: (a) the maximum measured 15 minute integrated demand in kilowatts occurring during the month, or (b) 50 per cent of the highest demand which has occurred during the preceding eleven months, or (c) the demand which the Customer has requested the Company in writing to stand ready to furnish for standby or auxiliary service. At the Company's option, a demand meter will be installed whenever the Customer's monthly consumption has exceeded 4,000 kilowatt hours for three consecutive months, or when the Company estimates that a Customer's consumption will exceed 4,000 kilowatt hours or that his demand will exceed 20 KW per month.

(Continued)

Schedule No. 30

GENERAL SERVICE
(Continued)

SPECIAL CONDITIONS

1. Temporary disconnection of any portion of load will not be considered as affecting the monthly minimum charge.

2. For recurrent seasonal or intermittent service to a permanently established business or enterprise, when service is disconnected before the expiration of a twelve months period, the closing bill will include the minimum charge which would have been charged for the remainder of the twelve months period had service not been disconnected.

3. Customers described under Special Provisions 4 and 5 of Schedule 10, may be served under Schedule 30.

4. For commercial buildings, apartment houses, court groups, auto camps, and the like, for which individual customers are submetered, the charge to individual customers must be at the Company's regular tariff rate for the type of service which such individual customer may actually receive.

QUEEN OF BRONZE

Data given by Mr. William Cox, A. S. & R.
Taken from an A. S. R. File Report by Noel Justin Davenport

Carload shipped in September, 1941:

Dry ton - - 61

Cu	13.76%	As	.65%
Zn	1.00%	$Si\ O_2$	7.00%
Fe	37.90%	Ag	.30 oz.
S	27.30%	Ni	.21%
Au	.08 oz.	Ca O	1.00%

Carload gave net returns of $1,083.79

Carload shipped in November, 1941:

Dry tons - - 52

Cu	13.73%	As	.24%
Zn	1.00%	Ag	.38 oz.
Au	.06 oz.	Ni	.20%

Carload gave net returns of $838.06

Carload shipped August 1940:

Dry tons - - 53

Cu	12.42%	$Si\ O_2$	6.20%
Zn	.50%	Al_2O_3	2.90%
Fe	39.00%	Ag	.27 oz.
S	28.00%	Ni	.27%
Au	.05 oz.	Ca O	1.60%
As	.24%		

Data given by Mr. William Cox, A. S. R.,
From an A. S. R. file report by E. G. Spillsbury

Cu	13.21%	Zn	0.23%
Fe	25.53%	Au	.30 oz.
S	0.23%		

Waite Minerals, Inc.
Grants Pass, Oregon

INVENTORY - December 1, 1955

Located at Queen of Bronze North End Workings, Takilma, Oregon

1	Delco Generator, 100 KVA #47G51
1	General Motors Diesel Engine #67190773
1	1500 Gal. Diesel Tank
1	Ingersoll Rand, Model 50 Type 40 T, 2 stage Air compressor, #250SA179,
1	General Electric 50 HP induction motor, #796-630, both on steel subframe
1	8 strand V belt drive, complete w/ pullies
1	4 x 6 air receiver tank
1	Myers safety switch and box
1	Westinghous 50 HP Motor #1889762
1	General Electric safety switch and box
2	Electric Motors, 5 HP
3	Electric Motors, 3 HP
1	High Pressure Pump
1	Set Ore Car Trucks
4	Side Dump Ore Cars
600 ft	2 1/2 inch galvanized and iron pipe, installed
700 ft	4 inch black spiral pipe, installed
1800 ft	2 1/2 inch galvanized and iron pipe, not installed
400 ft	mine rail, not installed
1850 ft	double track mine rail,
6 lengths	20' each air hose
1	Blacksmith Forge
2	Orchard type camp heaters
1	oil burning stove
1	Desk and chair

1	Steel frame, Miscellaneous V wheels and pulleys
1	Kubit Impact Breaker, No. 3, #S4969
1	Universal Jaw Crusher, 8" x 12"
1	Truax End dump ore car
1	Grizzley, 9' x 20'
1	Roller carriage, 6'
1	Cabin - used as office, tin roof
1	Tool shed
1	compressor house, tin roof
2	8' x 10' open water tanks
20,000	bd feet of timbers, mostly 8 x 8 inch
	Various small tools, etc.