J. McLAREN FORBES
Consulting Geologist

2275 MUELLER DRIVE
RENO, NEVADA 89509

TELEPHONE: AREA CODE 702 - 826 1545

(48)
ITEM 80

1210 0080

OFFICE REPORT

TEST LEACHING POSSIBILITIES

CORALTA RESOURCES PROJECT

CONTACT, NEVADA

J. McLaren Forbes

January 30, 1979

SUBJECT: PRELIMINARY OFFICE REPORT CONCERNING DEVELOPING PROCEDURES AND APPROXIMATE COSTS OF TESTS FOR MINING AND LEACHING OF OXIDE COPPER NEAR CONTACT, NEVADA.

INTRODUCTION

As a preface to this report, I understand that Coralta Resources has reason to believe that at its Contact, Nevada, properties there are oxide copper occurences which can be treated by either heap or in-place leaching. As far as I am concerned, much of the value of the proposed test work will be lost if a complete and extensive sampling program is not adhered to. This sampling must be done to determine the grade of acid soluble and total copper in the test area to be leached, as well as the percent of copper that is recovered by leaching. From these figures it may be possible to extrapolate the expected copper recovery that might be obtained from the leaching of comparable copper bearing areas on the property.

This report is based on the presumption that copper bearing outcrops sampled only along bulldozer cuts, by Fred Humphrey and Frank Lewis, may contain sufficient acid leachable copper to justify a 10,000 ton leach test. Samples from a cut, which crosses a band of N. 70° E. striking and 75° S. dipping mineralization, averaged 0.4585 copper over 41.5 feet. This zone may be extensive enough to be mined and used for leach testing.

I have not, as yet, seen the area under consideration; however, it appears from Mr. Lewis' verbal description, and from color pictures showing where samples 9-2-3 through 9-2-8 were taken, that there could be a possibility of developing sufficient copper mineralization from which to make the 10,000 ton leach test. This area will be referred to as Test Zone-1 (TZ-1).

Mr. Lewis wishes me to suggest a plan for testing TZ-1, or other such zones if TZ-1 does not contain the desired 10,000 tons of leaching material. The objectives of this plan will be: to evaluate the grade of both the total and leachable copper present, as well as to develop the TZ-1 area mineralization to make sure that there will be sufficient leachable copper present to warrant a small 10,000 ton test pit and a leaching installation; to suggest mining and leaching procedures, and predict assumed costs.

GENERAL TEST PLAN PROCEDURES

Samples:

Surface samples will be taken in the area under investigation wherever significant copper mineralization is exposed. All drill holes, both for exploration and blasting, will have ALL the cuttings saved from each 5 foot interval. Drill cuttings, after visual evaluation, will be selected for assay and/or for metallurgical testing.

Exploration:

Surface exploration, to determine the width and length of the outcrop, on either side of samples 9-2-3 through 9-2-8, will begin by visual inspection, followed by bulldozer cuts as needed. Surface samples are to be taken where copper showings outcrop or are exposed in the new cuts. There should be no more than six short cuts. Exploration at depth will be by no more than ten vertical drill holes, either by wagon (track) or rotary drilling.

Test hole exploratory drilling and blast hole drilling will be done by either a wagon (track) or rotary drill, equipped with a suitable sample collecting device, and capable of putting down a 3 inch to 4 inch diameter hole to a depth of 50 feet. The test holes will be located at sites that, hopefully, will show that mineralization continues to depth, or at other sites to find out if mineralization increases below low grade outcrops.

Mapping:

At this time, a tape and brunton survey will be adequate to outline the outcrop area, the bulldozer cuts, geology, the location of exploration drill holes, and proposed blast hole drill locations.

After the brunton survey has been completed, the geology mapped and bulldozer cuts, exploration drill holes, and sample locations plotted, it should be possible to decide whether location TZ-1 appears to be capable of supplying 10,000 tons of leaching material for the test pad, or whether in this area smaller tonnages are readily available for leach testing.

If site TZ-1 cannot supply the 10,000 tons desired for testing purposes, another site will have to be selected, using the same procedures as have been outlined, in order to determine whether the new test site contains the needed tonnage. At that time, it may be possible to determine whether or not leach testing of less than 10,000 tons will give satisfactory results.

For this report, it is now assumed that the values obtained from the samples, taken from the outcrop and the test hole exploratory drilling, indicate that there will be available the 10,000 tons needed for the leaching test. This material will be mined from an area approximately 100 feet long by 40 feet wide and 30 feet deep.

It is also assumed that continuing metallurgical testing has indicated that heap leaching of test site TZ-1 material when broken to -6 inches may probably be leached successfully.

Mr. Lewis believes that there is oxidized copper ore around old workings, as well as ore that has not yet been mined and these possibly could be leached in place. If such leaching were to be done, the areas to be leached would undoubtedly have to be fractured or rubblized by blasting, in order to get the leach solutions through the rock mass. A rubblizing blast at test site TZ-1 would provide data regarding the feasibility of such an in-place leaching operation.

MINING

A properly designed blast to break up, fracture, or rubblize the TZ-1 copper bearing rock and then move it to the leach pad with load-haul-dump (LHD) equipment should be considered instead of the usual blasting or ripping, truck hauling to a crushing plant, and then moving the crushed material to the leach pad. Such a rubblizing blast, at TZ-1, and subsequent leaching of the blasted copper bearing rock would provide data regarding the feasibility of an in-place leaching operation.

Blasting:

Oscar Margraf, of Margraf Explosives in Sacramento, California, was asked to make a rough estimate of the cost of rubblizing a zone such as TZ-1, breaking the rock to under 6 inches in size, for a small pit 100 feet long, 40 feet wide, and 30 feet deep with a ramp entrance 50 feet long. He expects that such a blast would take, roughly, in the neighborhood of 16,000 pounds of explosives (Dupont

Tovex) at an explosive cost of about $8,000. A USBM test rubblizing blast breaking about 20,000 tons of rock used 51,500 pounds of explosive (ANFO) at a cost of approximately $6,000, and resulted in only 2% of the fragments being over 4 inches in their longest dimension.

This blast is to be designed so as to fracture or rubblize the ground, with a minimum of throw and surface rise. After the blast, all the drill holes, which have been assayed, will be relocated upon the surface of the blasted area; and if there has been very little heave and throw, they should indicate copper grades within the area of drill hole influence. By using these drill hole grades, visual inspection, and re-sampling and analysis where deemed needed, it should be possible to make a distinction between leaching material and waste. The grade of the material sent to the leach pad will be determined by the results of the metallurgical testing and assumed operating costs.

Leaching:

Leaching rock will be sent to the leach pad site by means of LHD (load-haul-dump) equipment or shovel and truck. At this time I feel that the LHD movement of leach rock will probably be the best method to use. LHD haulage equipment should not compact the leach dump as much as truck haulage, and LHD would probably be cheaper.

The site of the leach dump, which will determine the length of haul, will depend upon the topography, available water, permeability of the leach dump base material, and environmental considerations.

For a 10,000 ton leach dump the following assumptions have been made:

The rock in place will weigh approximately one ton per 12.5 cubic feet. The volume of 10,000 tons, in place, will be approximately 125,000 cubic feet.

The material in the proposed leach dump, according to Taggert's figures for volumes of rock in dumps and bins, is from 1.6 to 1.9 times the in-place volume, which averages about 22 cubic feet per ton of dump material; therefore, the volume of 10,000 tons of copper bearing material, when transferred to the leach dump, will now be about 220,000 cubic feet.

This proposed leach dump will be in the form of a truncated pyramid, with a square base 135 feet on a side, 20 feet high, and 75 feet square at the top. The slope angle of repose of the pyramid's sides will be about 35°.

The leach dump will be constructed upon a graded and compacted base 150 feet square, with two opposite sides sloping slightly, for drainage, toward the declining center line into a catchment basin at one edge of the base. The impervious base of the dump will be of tamped clay covered by a plastic liner. On top of this impervious base there will be a narrow drainage layer consisting of assorted sand and gravel, in which may be imbedded perforated plastic drain piping. The drainage layer will discharge copper bearing leach solutions (pregnant solution) into a catchment basin. From the catchment basin the solution will be recirculated back through the leach dump, if need be, to increase the copper content, or discharged into the cementation tanks (i.e. precipitation tanks) where the copper will be precipitated upon scrap iron.

Leach solutions can be applied either by trickle-leach or by ponding-leach methods. Leach solutions will consist of fresh water with sulfuric acid (H_2SO_4) added to obtain a pH of about 2. The trickle-leach application will be by means of sprinklers or wobblers (short lengths of plastic tubing) discharging solution, at a rate of 5 to 10 gallons per minute over each thousand square feet of dump surface. The maximum flow of leach solution, for the whole leach dump, should be in the neighborhood of 60 gallons per minute. The ponding-leach method, which consists of flooding the surface of the dump, will use about the same amount of solution. Trickle-leaching appears to be the preferable method since there should be better aeration of the heap during the leach cycles.

Precipitation:

The precipitation plant will consist of two wooden or concrete launders, with false bottoms to aid in washing out precipitate, about 2 feet by 4 feet in cross section and 10 feet long. Instead of wooden launders, it may be possible to use lengthwise split halves of large diameter concrete culvert pipe. The cementation launders should be set up so that they can be readily drained of leach solution and cleaned of cement copper and iron scrap. They will be operated alternately to allow for clean-up of one launder while the other one is in use.

The cement copper will be sluiced off the iron daily, or as needed, by high pressure water, all of which will be discharged to a settling-drainage-drying pad. The drained cement copper will be sun dried and sacked for shipment. Discharge solution from cementation will either go to barren solution storage or be bled off as waste solution, if it is no longer suitable to be used for leaching.

Product:

If the proposed 10,000 tons of rock to be leached contains:

0.50% total copper there will be:

100,000 pounds of copper present that at:

50% recovery of acid soluble copper will produce:

50,000 pounds of copper which at a price of:

$0.60 per pound = $30,000

$0.70 per pound = $35,000

$0.80 per pound = $40,000

Costs:

Estimate of probable costs for test leaching site TZ-1.

Pre-leaching:

Bulldozer, sample cuts, pad building, roads, etc.	$ 5,000
Assaying, surface samples for total copper, oxide copper and/or soluble copper, and gold and silver, some groups of samples will be composited, 50 samples at $10 each	500
Drill hole samples, to be sampled for selected intervals and composited for grouped sample intervals, and assayed as for surface samples, maximum 700 samples	7,000
Preparation of leach pad, ½ acre at $12,000/acre	6,000
Cementation launders, and drying pad of wood or concrete	2,000
Water, well with pump, etc.	5,000
Two acid proof pumps, 100 gallons per minute each with piping	2,500
Power supply, generator, etc.	2,000
Sub-total	$30,000

Mining-Leaching:

Drill holes for a rubblizing blast, spacing as per Margraf's estimate 5,000 feet @ \$3.00 per foot	\$15,000
Powder for blast, as per Margraf's estimate 16,000 pounds @ \$0.50 per pound	8,000
Moving 10,000 tons from blast site to the leach pad @ \$0.50 per ton	5,000
H_2SO_4 for 50,000 pounds of recoverable copper, at 3 pounds acid per pound of copper recovered, 75 tons of acid @ \$30 per ton	2,250
Iron at 3 pounds per pound of recovered copper 75 tons of iron @ \$130 per ton	9,750
Solution make up water at \$0.10 per 1,000 gallons at $\pm$ 60 gallons per minute for 100 days	1,000
Smelting, refining, selling costs at \$0.30 per pound of copper produced, for 50,000 pounds	15,000
Sub-total	\$56,000

Miscellaneous:

Trailer for office and laboratory	\$ 3,000
Laboratory equipment, chipmunk crusher, pulverizer, scales, glassware, reagents, etc.	4,000
Change room with shower in case of acid spill, first aid kit, wire basket, etc.	1,000
Pick up truck	10,000
Sub-total	\$18,000

Personnel:

TZ-1 test, 4 months:

Supervision, retainer for 10 days per month at $200 per day and $150 per day for extra days, also field expenses and transportation, probable maximum for 4 months	$ 15,000
Technical assistant, recent graduate or advanced student capable of setting up a minimum laboratory and running wet assays at a maximum of $2,000 per month	8,000
Labor, 4 people at $50 each per day for 120 days	24,000
Possible metallurgical consulting	4,000
Sub-total	$ 51,000
Grand total	$155,000

The attached tables, regarding leaching costs, came from a paper presented at the Inplace Leaching and Solution Mining Short Course given by the Mackay School of Mines, University of Nevada, Reno, in November, 1975, by Roshan B. Bhappu of Mountain States Research and Development. The figures he gave, with modification for inflation, are certainly applicable today, even if they are for larger tonnages than may be mined and leached by Coralta Resources at Contact, Nevada.

These tables consider copper ores with grades of 0.50% and 1.00% copper and a 60% recovery to be mined and leached, either in heaps or in place (in situ). Copper recovery will be either by cementation on scrap iron or by solvent

(Continued on page 20)

COPPER ORE - 1,650,000 TONS PER YEAR
PRODUCTION AND SALES

OPERATING DAYS PER YEAR	330
TONS ORE TREATED PER OPERATING DAY	5,000

PRODUCTION

POUND COPPER RECOVERED PER YEAR IN THOUSANDS OF POUNDS

ORE-PERCENT COPPER	0.50	1.00
HEAP LEACHING (60% recovery)	9,900	19,800
IN SITU LEACHING (60% recovery)	9,900	19,800

ANNUAL SALES IN THOUSANDS OF DOLLARS

COPPER-PERCENT	0.50		1.00	
COPPER-PRICE PER POUND	0.60	0.80	0.60	0.80
HEAP LEACHING	5,940	7,920	11,880	15,840
IN SITU LEACHING	5,940	7,920	11,880	15,840

COPPER ORE - 1,650,000 TONS PER YEAR
CAPITAL COST ESTIMATES (IN MILLIONS OF DOLLARS)

TYPE OF LEACHING	HEAP		HEAP		IN SITU	
RECOVERY METHOD	CEMENTATION		SX-EW		SX-EW	
COPPER PERCENT	0.50	1.00	0.50	1.00	0.50	1.00
MINING	2.0	2.0	2.0	2.0	—	—
CRUSHING	2.2	2.2	2.2	2.2	—	—
ORE HANDLING	0.5	0.5	0.5	0.5	—	—
GRINDING	—	—	—	—	—	—
LEACHING	0.5	0.5	0.5	0.5	0.7	0.7
METAL RECOVERY	1.0	1.2	4.5	7.5	4.5	7.5
SUPPORTING FACILITIES	0.2	0.2	0.2	0.2	0.2	0.2
TOTAL	6.4	6.6	9.9	12.9	5.4	8.4
DOLLARS PER TON DAY *	1,280	1,320	1,980	2,2580	1,080	1,680

* 5,000 TONS PER OPERATING DAY. MINE CAPITAL IS NOT INCLUDED.

NOTES:

(1) COST OF DEVELOPING MINE NOT INCLUDED.

(2) COST OF DEVELOPING A TAILINGS DISPOSAL AREA AND EQUIPMENT FOR HANDLING TAILING NOT INCLUDED.

COPPER ORE - 1,650,000 TONS PER YEAR
SUMMARY OF ALL DIRECT OPERATING COSTS
(COST PER TON ORE TREATED)

TYPE OF LEACHING	HEAP		HEAP		IN SITU	
RECOVERY METHOD	CEMENTATION		SX-EW		SX-EW	
COPPER PERCENT	0.50	1.00	0.50	1.00	0.50	1.00
MINING (note I)	0.900	0.900	0.900	0.900	0.350	0.350
CRUSHING	0.216	0.216	0.216	0.216	—	—
ORE HANDLING	0.159	0.159	0.159	0.159	—	—
GRINDING	—	—	—	—	—	—
LEACHING	0.381	0.554	0.347	0.482	0.190	0.325
METAL RECOVERY	2.055	4.063	0.490	0.771	0.455	0.735
SUPERVISION	0.038	0.038	0.096	0.096	0.053	0.053
ADMINISTRATION	0.202	0.206	0.278	0.333	0.176	0.230
TOTAL	3.951	6.136	2.486	2.957	1.274	1.743
COST/POUND OF COPPER	0.659	0.511	0.415	0.246	0.212	0.145

NOTE I:

STRIPPING RATIO 1:1

ORE AND WASTE COST $0.45 PER TON OF MATERIAL MINED

COPPER ORE - 1,650,000 TONS PER YEAR
CASH FLOW (IN THOUSANDS OF DOLLARS)

0.50 PERCENT COPPER - COPPER AT $ 0.60 PER POUND

TYPE OF LEACHING	HEAP	HEAP	IN SITU
RECOVERY METHOD	CEMENTATION	SX-EW	SX-EW
CAPITAL	6,400	9,900	5,400
NET SALES	5,940	5,940	5,940
COST AND EXPENSES:			
TOTAL OPERATING	6,523	4,104	1,690
MINE AMORTIZATION (1)	50	50	——
DEPRECIATION (2)	640	990	540
DEPLETION (3)	891	891	891
TOTAL	8,104	6,035	3,121
OPERATING INCOME	(2,164)	(95)	2,819
FEDERAL INCOME TAX	——	——	1,353
NET INCOME	(2,164)	(95)	1,466
CASH FLOW	(583)	1,836	2,897
PAY-OUT IN YEARS	——	5.4	1.9
PERCENT RATE OF RETURN (4)	——	14.0	54.5

NOTES:

(1) COST OF DEVELOPING MINE PRIOR TO PRODUCTION $ 500,000. AMORTIZE THIS COST DURING THE LIFE OF THE MINE-TEN YEARS.

(2) DEPRECIATION - STRAIGHT-LINE TEN YEARS.

(3) DEPLETION - 15% SALES

(4) CASHFLOW DISCOUNTED TO PRESENT WORTH.

COPPER ORE - 1,650,000 TONS PER YEAR
CASH FLOW (IN THOUSANDS OF DOLLARS)

0.50 PERCENT COPPER - COPPER AT $ 0.80 PER POUND

TYPE OF LEACHING	HEAP	HEAP	IN SITU
RECOVERY METHOD	CEMENTATION	SX-EW	SX-EW
CAPITAL	6,400	9,900	5,400
NET SALES	7,920	7,920	7,920
COST AND EXPENSES:			
TOTAL OPERATING	6,523	4,104	1,690
MINE AMORTIZATION (1)	50	50	———
DEPRECIATION (2)	640	990	540
DEPLETION (3)	1,188	1,188	1,188
TOTAL	8,401	6,332	3,418
OPERATING INCOME	‹481›	1,588	4,502
FEDERAL INCOME TAX	———	762	2,161
NET INCOME	‹481›	826	2,341
CASH FLOW	1,397	3,054	4,069
PAY-OUT IN YEARS	4.6	3.2	1.3
PERCENT RATE OF RETURN (4)	18.3	29.4	75.0

NOTES:

(1) COST OF DEVELOPING MINE PRIOR TO PRODUCTION $ 500,000. AMORTIZE THIS COST DURING THE LIFE OF THE MINE - TEN YEARS.

(2) DEPRECIATION - STRAIGHT-LINE TEN YEARS.

(3) DEPLETION - 15% SALES

(4) CASHFLOW DISCOUNTED TO PRESENT WORTH.

COPPER ORE-1,650,000 TONS PER YEAR CASH FLOW (IN THOUSANDS OF DOLLARS)

1.00 PERCENT COPPER - COPPER AT $ 0.60 PER POUND

TYPE OF LEACHING	HEAP	HEAP	IN SITU
RECOVERY METHOD	CEMENTATION	SX-EW	SX-EW
CAPITAL	6,600	12,900	8,400
NET SALES	11,880	11,880	11,880
COST AND EXPENSES:			
TOTAL OPERATING	10,126	4,830	2,464
MINE AMORTIZATION (1)	50	50	——
DEPRECIATION (2)	660	1,290	840
DEPLETION (3)	1,782	1,782	1,782
TOTAL	12,618	8,002	5,086
OPERATING INCOME	(738)	3,878	6,794
FEDERAL INCOME TAX	——	1,861	3,261
NET INCOME	(738)	2,017	3,533
CASH FLOW	1,754	5,139	6,155
PAY-OUT IN YEARS	3.8	2.5	1.4
PERCENT RATE OF RETURN (4)	26.4	39.1	77.0

NOTES:

(1) COST OF DEVELOPING MINE PRIOR TO PRODUCTION $ 500,000. AMORTIZE THIS COST DURING THE LIFE OF THE MINE-TEN YEARS.

(2) DEPRECIATION - STRAIGHT-LINE TEN YEARS.

(3) DEPLETION - 15% SALES

(4) CASHFLOW DISCOUNTED TO PRESENT WORTH.

COPPER ORE-1,650,000 TONS PER YEAR
CASH FLOW(IN THOUSANDS OF DOLLARS)

1.00 PERCENT COPPER-COPPER AT $ 0.80 PER POUND

TYPE OF LEACHING	HEAP	HEAP	IN SITU
RECOVERY METHOD	CEMENTATION	SX-EW	SX-EW
CAPITAL	6,600	12,900	8,400
NET SALES	15,840	15,840	15,840
COST AND EXPENSES:			
TOTAL OPERATING	10,126	4,880	2,464
MINE AMORTIZATION (1)	50	50	———
DEPRECIATION (2)	660	1,290	840
DEPLETION (3)	2,376	2,376	2,376
TOTAL	13,232	8,596	5,680
OPERATING INCOME	2,628	7,244	10,160
FEDERAL INCOME TAX	1,261	3,477	4,877
NET INCOME	1,367	3,767	5,283
CASH FLOW	4,453	7,483	8,499
PAY-OUT IN YEARS	1.5	1.7	1.0
PERCENT RATE OF RETURN (4)	67.5	57.9	———

NOTES:

(1) COST OF DEVELOPING MINE PRIOR TO PRODUCTION $ 500,000. AMORTIZE THIS COST DURING THE LIFE OF THE MINE-TEN YEARS.

(2) DEPRECIATION-STRAIGHT-LINE TEN YEARS.

(3) DEPLETION-15% SALES

(4) CASHFLOW DISCOUNTED TO PRESENT WORTH.

extraction-electrowinning (SX-EW). Mining rate is to be at 5,000 tons per day, 330 days per year for 1,650,000 tons per year.

CONCLUSIONS

I. The most important results of this test program will be: the determination of the expected percent of contained leachable copper that will be recovered; the length of time that it will take to accomplish this leaching; and the maximum fragment size that can be leached. Without this information it will be very hard to make a justifiable evaluation of the probable success of a leaching operation, either in place or on leach pads.

II. That portion of the foregoing estimate which considers the mining-leaching costs results in an approximate operating cost of $56,000 or $1.12 per pound of copper.

This assumed cost, for the mining-leaching portion of test TZ-1, is high. However, if this test does indicate that the ore may be amenable for the recovery of copper, by a mining-leaching process, without use of excessive explosive, acid or iron consumption, it should be possible to plan a profitable leaching project, provided that sufficient ore can be developed to insure a profitable operation.

III. As an alternative to one large 10,000 ton leach dump, it might be well to consider splitting the rubblized blasted rock into several smaller leach dumps, depending upon the maximum fragment size derived from the blast. These smaller test dumps would be run-of-mine or crushed blasted rock and would not be screened. They might be as follows:

Run-of-mine (original blast) hopefully -6 inches

Run-of-mine crushed to -4 inches

Run-of-mine crushed to -2 inches

Whether the above alternative is to be considered may well depend upon the results of the rubblizing blast.

IV. Another alternative to the 10,000 ton leach dump might be to plan a rubblizing blast that could be leached in place, by collecting the leach solutions from a decline or adit, depending upon the topography, driven below the rubblized blast zone.

V. In the preface of this office report, I mention that I have not seen Coralta Resources' property and that I am working from plausable, but scanty, information furnished by Frank Lewis. I hope the data I have gathered will adequately present workable procedures to be followed toward gaining the information necessary to determine whether or not Coralta Resources does have an economically exploitable leach project.

McLaren Forbes

January 30, 1979 J. McLaren Forbes

J. McLAREN FORBES
Consulting Geologist

2275 MUELLER DRIVE
RENO, NEVADA 89509

TELEPHONE: AREA CODE 702 - 826 1545

PROFESSIONAL RECORD

1934-36	Various jobs as a mucker, assayer, geologist, surveyor, flotation and cyanide plant operator and other mill work.
1936-40	Philippine Islands, as follows:
36-38	Mining Geologist and Mine Superintendent, Opisso & Co.
38-39	Geologist and Mill Superintendent, Paracale Gumaus Mine.
39-40	Geologist for International Engineering Co., reconnaissance in Northern Luzon, and mine geologist at North Camarines.
1941-45	Chief Geologist and Chief Engineer, New Idria Quicksilver Mining Co. Developed detailed isometric block diagrams showing observed and projected geology of the mineralized area.
1945-46	Geologist, U.S. Forest Service Flood Control Project. Examined dam and road locations, made watershed erosion surveys.
1946-46	Geologist, Exploration Department, Potash Co. of America.
1946-49	Geologist for the United States Smelting, Refining and Mining Co. at Vanadium, New Mexico. Detailed mapping in their vein and limestone lead-zinc mines. Kept up geology maps, sections, ore reserves; laid out diamond drilling and exploration.
1949-51	Chief Geologist at Frontino Gold Mines, Ltd., Colombia, South America. Started first detailed mine geology at this vein gold mine.
1952-58	Chief Geologist, Consolidated Coppermines Corp., Kimberly, Nevada. Experience in the geology, exploration and drilling of porphyry copper ore bodies, their reserve calculations, open pit design and mining. Examined more than 200 mining properties in the western U.S.A.
Fall of 1958	Examination of Cerro de Pasco properties in Peru for Consolidated Coppermines.
1958-1965	Consulting Mining Geologist, long term jobs, as follows:
59-61	Consulting Geologist for Cerro Corporation, McCune Pit Project, Cerro de Pasco, Peru. Assembled basic data for their decision to design and develop an open pit at this major producer of lead, zinc, silver, and copper.
62-63	Consulting Geologist for Placer Developments, Pamex Mining Co., Philippines, (now Marcopper Mining Corp.). In charge of development and exploration of this porphyry copper. Staff and labor force, 200 to 300. Eight diamond drills, underground and surface, up to eight development headings.
	Six months for Placer Development, Ltd., on their American Exploration Company, Plumas County, California, copper prospects.
64-65	Three-quarters of my time for Kern County Land Co., working on a western states silver project.
65 to Present:	Consulted for various clients, including the following: Straus Exploration Co. (formerly Guggenheim Exploration Co.). Work in Chile, Brazil, and the western United States.

Kerr-McGee Corp.
Examined, evaluated and reported on silver-lead-zinc prospect in northern Peru.

Standard Oil of Indiana.
One of the team selected by Chapman Wood and Griswold to examine Cerro Corporation's Peruvian Properties, at the time of the proposed merger of Cerro Corp. with Standard of Indiana. I evaluated the ore reserves of Cerro de Pasco's mines in Peru.

Involved in the geological studies and mining of Cerro Corporation's Big Mike mine in Nevada.

Consultant for Bethlehem Steel Corporation; supervised development at a porphyry copper property in Sonora, Mexico.

Have had some placer evaluation experience.

EDUCATION

1928-34	University of Arizona, B.S., 1933. Majored in geology, with courses in mining and ore dressing. Graduate work 1933-34.
1940-41	University of California at Los Angeles, graudate work.
1956	U.S.G.S. two-week Geochemical Exploration course at Denver.
1962	Computer Short Course and Symposium, University of Arizona.
1975	November 10-14: Short Course - Inplace Leaching and Solution Mining, Mackay School of Mines, University of Nevada.
1976	October 25-30: Short Course - Placer Exploration and Mining, Mackay School of Mines, University of Nevada,

Have a working knowledge of Spanish.

REGISTRATION

Professional Engineer and Land Surveyor, New Mexico.
Registered Geologist, California.

PROFESSIONAL SOCIETIES

American Institute of Mining and Metallurgical Engineers.
Society of Economic Geologists.
Geological Society of America.
Geological Society of Nevada.
Nevada Exploration Geologists

PUBLICATIONS

The Isometrograph as Developed and Used at The New Idria Quicksilver Mine, Calif. Jrnl. Mines & Geology, July 1943.

Practical Diamond Drilling for the Geologist and Engineer, Transactions A.I.M.E., Vol. 163.