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Kappes, Cassiday & Associates

1845 Glendale Avenue, Sparks, Nevada 89431

702-356-7107 - Telex 170049 - Fax 702-356-5609

18 June 1987

Dear Sir:

This letter comprises a bid for one 200 US gpm carbon Adsorption-Desorption-Recovery plant (ADR plant) for a project located in Nevada or nearby states.

The plant price is $178,000.00 FOB Sparks, Nevada. Nevada sales taxes (approximately 6 percent) are not included, and will be charged if applicable.

Installation

Installation costs include shipping to site, complete setup and startup costs. It is assumed that the operator will have two semi-skilled personnel (the future plant operators) available for two weeks during setup to assist with setup, and to learn pre-startup procedures. (If these personnel are not available, the costs shown below will increase $150.00 per man-shift not furnished). Startup assistance and training will be furnished one shift per day for two weeks following setup (Training may be delayed until sufficient process solution is available).

Total cost of Installation, Startup
and Training: $ 26,500.00

Our rate for startup assistance beyond the above "installation" cost, for a KCA engineer familiar with carbon ADR plant practice, is $8,800.00/ month, plus expenses, for time planned and committed on long-term basis, or $440.00 to $580.00 per day for daily services, plus expenses.

Optional extras are discussed in a later section.

General Plant Description

The proposed plant is a complete system for adsorbing gold and silver from cyanide solutions through upflow, fluidized carbon beds, followed by non-pressurized desorption using caustic, caustic-alcohol (preferred method) or caustic-glycol strip systems. The Plant is built conservatively oversize to permit operation in excess of design capacity.

Gold and silver are recovered onto steel wool cathodes. Cathodes are direct-smelted in a 225-pound propane-fired tilting crucible furnace, which is mounted on an external concrete slab (slab and enclosure not furnished).

Piping and pump systems are included for pumping solution from the pregnant pond through the columns and discharging to the barren pond; and for pumping from the barren pond to an adjacent point leading to the heaps. The pumps will be "Tsurumi" submersible pumps mounted in floats or on slides on the pond berm. Pumps will be furnished complete with motor controllers and dual controls inside the plant and at the pond edge.

The plant is designed to permit loading of carbon to 150 ounces gold per ton or better (depending on the system chemistry; similar plants KCA has built achieve loading approaching 1000 ounces per ton), and stripping of two column loads simultaneously. When utilizing a 48-hour strip cycle (and 150 oz/ton loading) the plant is capable of recovering 3700 ounces gold per month. A 24-hour strip cycle can often be attained, which doubles gold production capacity.

Description of Included Items, Alternative 1.

The plant consists of the following items of equipment:

one - column module container consisting of a 45-ft. semi-trailer with high quality floor, ceiling and wall insulation, internal fluorescent lighting, forced air (propane) heating/ventilation, good tires and brakes, external paint with client logo, internal fire extinguishers, one window, one window/escapeway

one - recovery module container consisting of a 28-ft. semi-trailer with high quality floor, ceiling and wall insulation, internal fluorescent lighting, forced air (propane) heating/ventilation, good tires and brakes, external paint

two - steel stairs for plant access when mounted at trailer height above ground

six - adsorption/desorption/acid wash columns, 10 sq. ft. cross-sectional area, 5.5 ft. settled carbon bed depth (8 ft. overall height), arranged as three sets of two columns each; mild steel columns with epoxy lining

one - DSM-type screen and sump for catching and recycling floating carbon

one - pregnant solution pump

one - barren solution pump

one - shedding vortex flowmeter with totalizer

two - floating pond intakes

one - storage tank for strip solution

one - 30 kW electric immersion heater system for heating of strip solution (propane heating may be optionally furnished)

one - sink and countertop

one - work table for cathode manufacture

one - electrolytic cell and variable-output power supply, including cell hood and exhaust fan

one - propane-fired tilting pot furnace with 225-pound crucible

one - main electrical switchbox for external electrical connection (only one connection is required, with other voltages derived internally)

- valves: butterfly and ball valves, steel body with stainless steel working parts

- piping: PVC and CPVC, some steel, depending on duty requirements

- connections: flanged fitting will be used preferentially (if appropriate)

- heat tracing and insulation: all external pipelines

- freeze protection: complete drain capabilities.

Options and Additions

The plant as quoted above is a complete plant capable of producing dore bullion. However, the following options and additions may be desired or required in specific instances. Prices quoted below are installed prices unless otherwise indicated; they do not include Nevada Sales Tax, approximately 6 percent additional.

Concrete Slab Below Trailer Modules. It is normally a good idea to mount the trailer modules on a 6-inch thick concrete slab with a curb and central drain to the pregnant pond.

This slab should have dimensions 32 by 60 feet. KCA will install this slab as part of the plant setup for a cost of $7200.00.

Pressure Strip Option. The quoted plants are designed for non-pressurized stripping of carbon, at an operating temperature of 83°C. Most KCA plants are run using an ethanol strip system, although they can be operated using ethylene glycol or non-organic strips.

We can provide the plants with the capability to strip using 260°F, 25 psi pressurized strip circuit for an additional cost of $120,000.00.

Strip Solution Heating Using Propane. The quoted plants required 30 kW of electric power for heating of strip solution during the preliminary phase of the strip cycle. If this power is not readily available, it is possible to heat the solution using a propane-fired heater. The additional cost to substitute propane heating for electric heating is $5000.00.

Carbon Regeneration Furnace. For an operation using a 200 gpm plant (approximately 1200 tons ore per day), we do not recommend the installation of a carbon regeneration furnace.

Normally, the activated carbon used in the system will maintain its activity sufficient to meet design conditions for slightly in excess of one year. The Sterling, Nevada, plant, which is a similar design to the plant proposed here, has been running since 1981 with average carbon replacement every 14 months. Other KCA plants have similar operating experience.

Once activity is diminished, the carbon can then be sent for custom regeneration in batches, or sold to a large operator. The cost to regenerate carbon will be approximately $12,000.00 per regeneration cycle (probably, $12,000.00 per year).

KCA will furnish an Allis-Chalmers rotary kiln designed for 50 pounds carbon per hour regeneration capacity, complete with screens, tanks, propane hookup, and carbon transfer facilities, all installed and operable, at a cost of $105,000.00 in client's building or $160,000.00 in a concrete-slab, steel-frame building.

Smelting Furnace Slab, Building and Scrubber. The smelting furnace which is furnished with the plant may be operated by placing it on a gravel base or, preferably, a concrete slab measuring 10 x 20 feet. Open air installation and operation is sufficient, although a simple open-walled structure with corrugated steel roof is preferred.

The quoted price for the plant does not include a slab or housing for the furnace. The installed price includes installation of the furnace in client's housing.

If mercury is present in the ore, or if the furnace is to be housed inside a building, it may also be desirable to install a furnace hood, exhaust fan, and wet exhaust scrubber.

KCA will provide a simple slab for a cost of $900.00.

KCA will provide a complete installation including steel pre-fab building, exhaust system and wet scrubber (designed for low-efficiency particulate removal) at an additional cost of $55,000.00.

Mercury Retort. If sufficient mercury is present in the ore, a retort should be installed prior to smelting. KCA can provide a retort sufficient to retort a complete set of electrolytic cell cathodes at a cost of $55,000.00. This retort could be housed within the same building as the smelting furnace at an additional cost for that building of $16,000.00.

Activated Carbon. The quoted prices do not include operating supplies or activated carbon. We currently recommend Pica G210AS carbon, but suitable carbon may be available from Westates Carbon Corporation or Calgon (Calgon type GRC-22 may be acceptable, but lot samples should be examined). Price of carbon is approximately $1.50 per pound.

The plant will require an initial carbon purchase of approximately 12,000 pounds (sufficient for startup plus makeup required during first six months of operation).

Schedule of Construction and Installation.

The plant can be ready for shipping 13 weeks after bid confirmation and receipt of down payment. Shipping will take one week. The plant can be ready to pump and accept process solution within two weeks after arrival on site. The desorption cycle of the plant will be ready one week thereafter.

Operating Manual.

The bid price includes a Basic Operating Manual which includes a valve list, layout drawing, flowsheet and a general discussion of operating procedures.

One section of the operating manual contains all purchase order information and maintenance literature which applies to the commercially-supplied components of the plant.

This Basic Manual has been sufficient for other plants which we have furnished. A Detailed Manual can be prepared, which lists operating plant philosophy and step-by-step operations for all logical operating conditions, at a cost of approximately $8,000.00.

Terms and Conditions.

We will require a payment of 30 percent of the total plant price with bid confirmation ($53,400.00), a 20 percent progress payment six weeks thereafter ($35,600.00), a payment of 25 percent upon shipping ($44,500.00), and the balance of 25 percent ($44,500.00) six weeks thereafter.

With the execution of suitable documents giving KCA the right to repossess on demand and sell the plant, extended payment terms will be provided by KCA at an additional cost calculated at 18 percent annual interest compounded daily on the net balance "overdue" based on the payment schedule outlined above. Late payments will be charged at the same interest rate.

For suitable projects, lease terms are possible.

General Technical Parameters.

1. Other than carbon, basic reagents and supplies needed for plant operation include those shown below. The consumption figures are approximately monthly levels at a full production rate of ten strip cycles per month:

muriatic acid (acid washing of carbon)	100 gal/mo.
steel wool and misc. cathode supplies	$150.00/mo.
ethanol	1000 gal/mo.
sodium hydroxide	800 lbs/mo.
sodium carbonate	800 lbs/mo.
furnace fluxes	400 lbs/mo.
furnace crucibles	2/mo.

2. The electrical loads for the plant consist of the following items (the power shown is installed power).

process pumps, two @ 15 hp:	25 kW
misc pumps	3 kW

strip solution heaters	30 kW
electrolytic cell power supply	12 kW
lights, ventilation, instruments	5 kW
Maximum expected power draw	75 kW
Average expected power draw	30 kW

3. Major categories of spare parts consist of:

rupture discs
pump seals and impellers
spare diode for electrolytic cell rectifier
minor stock of replacement pipe fittings
miscellaneous supplies and hardware items

For a plant operating in Nevada, we recommend a minimum amount of spare parts. Spare parts investment will be less than $3000.00.

4. The plant will be built to withstand operation and periodic shutdowns during freezing weather. One advantage of this type of plant is that it can be shut down and started up relatively easily, several times a shift if necessary, or at nights or on weekends. (Shutdown during freezing conditions requires that correct drain procedures be followed.)

5. The barren solution process pump shall be furnished to provide a pressure at the pump of 100 feet of head at the design flowrate, unless otherwise agreed to in writing.

We appreciate the chance to make this bid, and look forward to working with you on the project.

Sincerely,

Daniel W. Kappes

Kappes, Cassiday & Associates

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1 OCTOBER 1986

SMALL AND LARGE HEAP LEACHES - DIFFERENCES IN APPROACH

by

Randall A. Pyper and Daniel W. Kappes

for presentation at

CIM District 6 Meeting
Victoria, B.C.
3 October 1986

Heap leaching has had a reputation as a bit of an underdog - a low-cost, low-technology approach to properties that couldn't be treated any other way. The recent spate of very large heap leaches, however, has served to dispel that concept.

For example, the Mesquite operation of Consolidated Goldfields in Southern California is a $70 million dollar investment with a production rate of 12,500 tons per day.

The reasons to use heap leaching instead of conventional milling are illustrated by two recent installations in Nevada - Paradise Peak in central Nevada (FMC) and Hog Ranch in the northwest corner of Nevada (Western Goldfields/Ferret Exploration).

As Figure 1 shows, the capital costs of a conventional mill are very significant. Paradise Peak cost $20,000.00 per daily ton of capacity, whereas Hog Ranch - a low-key heap leach installation - cost less than $2,500.00 per daily ton.

At a gold price of $400.00 per ounce and an 11 percent cost of capital, an orebody with a gold content of 0.08 ounces per ton would have to show a very high recovery differential - 40 percent - to justify the mill. This assumes that milling and heap leach operating costs are identical, but of course the mill operating cost is significantly higher than the heap leach.

In the case of Paradise Peak, the mill could be justified because the ore is high grade and contains a large amount of silver which is not readily heap leachable. But for many large orebodies, heap leaching is the choice from a financial standpoint.

Thus the choice of treatment - conventional mill or heap leach - is between two equally valid methods that have both "come of age".

Technology is like the U.S. auto industry, however. As the technology matures, it tends to get larger and to develop a lot of complications.

Has heap leaching grown out of its original role as a "Model T" method for treating small deposits?

The answer should be "no" - Large and small deposits may require technically different approaches, but each should be valid in the appropriate situation.

Why is it that properties show this reverse economy of scale? i.e., why do they get more expensive per ton of throughput as they get larger? There are at least three reasons -

1) Small operations are more closely controlled by people who care - sophistication in design can easily give way to personal "hands-on" involvement in daily operations.

For instance, Little Bald Mountain in northeastern Nevada, near Placer's Bald Mountain operation, was able to successfully run a conveyor stacking system directly on leach pads because the managers are there to supervise each conveyor movement; Hog Ranch requires an 18 inch gravel cover.

2) Small operations can afford to be more labor intensive - they generally have a shorter life and are higher grade than the large ones.

FIGURE 1. CAPITAL COST COMPARISONS - MILL AND HEAP LEACH

	PARADISE PEAK	HOG RANCH
DAILY ORE TONNAGE	4000	4000
TYPE OF MINING	OPEN PIT	OPEN PIT
TYPE OF PROCESSING	CYANIDE MILL	HEAP LEACH
CAPITAL COST	$ 80 MILLION	< $10 MILLION
CAPITAL + FINANCE COST/TON	$ 15.00	$ 1.90
CAPITAL COST, OZ GOLD/TON	0.0375	0.005
PERCENT ADDED RECOVERY NEEDED TO JUSTIFY THE MILL - ORE GRADE 0.08 OZ GOLD/TON	41 %	

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FIGURE 2. CAPITAL COST COMPARISON - LARGE AND SMALL HEAP LEACHES

	MESQUITE	HOG RANCH	LITTLE BALD MTN
DAILY TONNAGE	12,500	4,000	400
CAPITAL COST - (INSTALLATIONS)	$ 50 MM	$ 7.0 MM	$ 0.5 MM
(TOTAL)	$ 70 MM	$ 10.0 MM	$ 1.0 MM
INSTALLATIONS, $/DAILY TON	$ 4,000	$ 1,700	$ 1,000

3) It's easier to modify a small operation to correct or work around initial design difficulties. LBM avoided any agglomeration in the first year even though the ore had a relatively high clay content; by selective mining and careful stacking, production losses were minor. Agglomeration was installed in year two, where it did not affect the initial capital requirement.

Another example of the "reverse economics" of size involves the amount of money spent initially on orebody development prior to the decision to proceed with production.

At what point should a company suspend exploration drilling and begin operations? Should it design operations only for proven tonnage, or for "expected" tonnage?

The small company might stop when it has enough tonnage to breakeven, particularly if it has good chances for developing additional tonnage later on. Usually, the sources of capital available to the small company are very expensive - stock offerings may appear to provide "free" money, but in terms of management time and future flexibility, this is very costly capital. Generally, the sooner a small company can begin developing a cash flow the better.

The large company, on the other hand, has more resources available to it - especially in the critical area of management talent needed to development or "massage" investment capital - its cost of capital is lower. At the same time, it generally places more importance on long-term stability and on financial measures of performance (i.e. return on investment becomes more important in relation to promotional value).

To accomplish its goals, the large company will need to show a decent return on investment and a decent life - and to do this, it will have to prove up at least one and a half times the amount needed for simple recovery of investment. In fact, to meet long-term stability goals, it may find it desirable to develop two or three times as much reserves as it needs.

The goals of a company do and should change. We have dealt with companies that - correctly - placed very small properties into production. Within a year or so, their minimum target for new properties often becomes much larger than the property they just developed. The first property was quite successful in both return on investment, and in allowing the company to out-grow it.

FIGURE 3. APPROACH TO ORE RESERVE DEVELOPMENT

LARGE COMPANY GOAL: TO CREATE LONG-TERM STABILITY
(DRILL FOR PLANNING PURPOSES)

.SMALL COMPANY GOAL: TO DECREASE COST OF FUTURE CAPITAL
(DRILL ONLY ENOUGH TO JUSTIFY DEVELOPMENT)

RECOVERABLE GOLD CONTENT (OZ/TON)	TYPICAL HEAP GRADE (OZ/TON)	BREAK-EVEN TONNAGE	20% DCFROI TONNAGE
0.03	0.045	3,000,000	5,000,000
0.05	0.077	1,000,000	2,000,000
0.06	0.092	460,000	800,000
0.08	0.12	100,000	160,000
0.10	0.15	50,000	85,000
0.15	0.23	20,000	35,000

GOLD PRICE US $ 400/oz

CONCLUSION

Orebodies aren't usually flexible. The small orebodies need the small-company approach. This aspect of the business is generally acknowledged though not always appreciated.

If both approaches are valid, can they both be done by the same company? Probably not.

The concept of a small-company is not very well understood in business. A "small-company approach" is not created merely by lowering the financial goals of the big company. Small companies have more personal involvement by the managers. For a big company to get small successfully, it has to find a way to shift personal involvement from upper management to lower management.

A recent article in one of the business magazines made the point that - fortunately - companies aren't biological organisms. They don't merely grow up, grow old, wither away and die. Good companies may go through many cycles of being large or small, but it's probably difficult to be both sizes at once.

In conclusion, it should be said that most companies which have built expensive "Cadillac" operations - the Mesquite heap leach, the Paradise Peak mill - seem to have made valid decisions. These companies have relatively low costs of capital and very long-term goals of maintaining corporate stability.

The other end of the spectrum - low-cost, rapid development - is equally valid and important. It can be the road to success for small companies with big future plans.

Cerrefeli Gold Bullf Heap

WLK

Write file Gdel – Bulk/Heap

ASARCO

Southwestern Exploration Division

July 2, 1985

F. T. Graybeal
New York Office

Copper Basin, CA
Padre-Madre, CA
Mineral Hill-Planet, AZ

I had asked A.E. Giesecke and M.A. Miller, near the end of their second month out on the Mohave Project, to get together and visit the Copper Basin and Padre-Madre areas in California. Both areas were believed to have the detachment-type structural mode and mineralization which has attracted a large number of drill holes. They were to take the areas apart and relate the mineralization to what they saw, and put them back into a possible usable model for further exploration for a viable deposit.

Mr. Giesecke has submitted reports on Copper Basin (May 30, 1985) and Padre-Madre (Au) (June 11, 1985) and these are being distributed. Mr. Miller is preparing his reports with samples collected.

Mr. Kurtz and I met both of them in ~~Barstow~~ Blythe, CA after they had visited both properties. From their verbal comments, it was suggestive that a critical evaluation was not possible with the time spent.

I attach the Wilkins-Heidrick report from the Mesozoic-Cenozoic Tectonic Evolution volume, noted by A.E. Giesecke, to contain information on the Copper Basin area along with a cross-section. The article is mainly on the Copper Penny property and aptly describes the features which characterize the detachment-type terrain and mineralization features applicable throughout this part of the world, including Padre-Madre.

Note also on pages 198-199 the part of the Mineral Hill-Planet area. This is covered by the AMAX group which wishes to joint venture it after drilling several holes and finding ore grade gold only in the high-angled fracture zones. See my letter to you on AMAX Confidentiality Agreement, Buckskin Project, La Paz County, AZ, dated June 28, 1985.

To emphasize that the detachment terrain is not without difficulties of interpretation, I also submit Rehrig's paper from the same M-C Tectonic volume.

James D. Sell

James D. Sell

JDS:mek
Att.

cc: W. L. Kurtz
Mohave File

CHARACTERISTICS OF GOLD AND SILVER HEAP LEACH OPERATIONS

MINE	OWNER	RESERVE (M=million)	CAPITAL COST STARTUP	PRODUCTION RATE	WASTE/ORE, CUTOFF	RECOVERY	OPERATING COST (1983)	GEOLOGY	COMMENTS
GOLD MINING OPERATIONS									
ALLIGATOR RIDGE Nevada	50% Amselco 50% ~~Occidental~~ NERCO	14M tons - 0.12 oz Au	$30M; permitting and construction took 15 months; startup Oct. 1980.	2200 tpd 60,000 oz Au/yr	W/O=3/1; 40-45° pit slope	70-75%	NA	Disseminated gold in zone of strong jasperoid alteration of silty limestone; deeper ore is carbonacous.	Crushing and agglomeration required; carbonaceous ore is stockpiled; more ore in 2 other zones; pad made from compacted clay mined locally; 140 employees.
ANNIE CREEK S. Dakota	Wharf Resources	6.5M tons - 0.043 oz Au 0.16 oz Ag	$4M; startup Summer 1983.	750,000 tpy	Cutoff = 0.02 oz; 45° pit slope	74%	$225-275/oz	Similar to Gilt Edge (?); disseminated gold in pyritic (oxidized) latite and trachyte porphyry in breccia and shear zones.	Wharf holds 50% interest in adjacent Bald Mtn. property (Homestake 50%) with reserves of 9.9 million tons averaging 0.041 oz Au, 0.17 oz Ag.
BOREALIS Nevada	Houston Int. Minerals (Tenneco)	2.1M tons - 0.08 oz Au 0.50 oz Ag	Startup Nov. 1981.	2300 tpd 30,000 oz Au/yr 30,000 oz Ag/yr	NA	NA	NA	Disseminated gold and silver halides in oxidized, silicified and argillized tuff and volcanic breccia.	40 employees.
CORTEZ Nevada	Placer Development	2.2M tons - 0.041 oz Au at Gold Acres 2.8M tons at Cortez.	Startup 1971; part of a milling operation.	NA (not available)	Cutoff = 0.015 oz	50% from clayey Gold Acres ore; 65% from blocky Cortez ore.	NA	Largely Carlin-type with fine-grained disseminated gold in silty limestone, variably carbonaceous.	No crushing, 170,000 ton heaps.
HAVELUCK W. Australia	Whim Creek	1.4M tons - 0.054 oz Au heap leach.	$2M; startup June 1980; 10 mo. for construction.	1000 tpd 14,500 oz Au/yr	W/O=2/1; cutoff = 0.018 oz	70%; 30% in 30 days	$250/oz HL $270/oz CIP	Disseminated gold in strongly weathered and talc-altered mafic volcanic rock.	Began as 100% heap leach; excessive fines required conversion to 50% heap leach, 50% CIP; mining by contractor; total reserve is 4.5M tons at 0.066 oz Au.
NORTHUMBERLAND Nevada	Amoco Minerals	10M tons - 0.052 oz Au	$17M; startup April 1981; 8 mo. for construction.	3000 tpd 30,000 oz Au/yr 30,000 oz Ag/yr	W/O=1.8/1 cutoff = 0.020 oz	60% from jasperoid, 70% from shaley limestone	$402/oz excluding deprec./depl.; haulage is $68/oz.	Carlin-type with disseminated gold in jasperoidized limestone and in decalcified silty limestone.	Crushing required; 10 mile haul pit to heaps; plastic liner under heaps; additional 6M tons averaging 0.034 oz Au at W/O=6.9/1; 110 employees.
ORTIZ New Mexico	Goldfields	6.8M tons - 0.053 oz Au	$14.5M; startup Feb. 1980; 12 mo. for construction.	3000 tpd 50,000 oz Au/yr	W/O=1.2/1; cutoff = 0.02 oz.	78% in 56 days; 60% in 23 days.	$261/oz excluding deprec./depl.	Disseminated gold in fractures and vugs in oxidized breccia pipe cutting quartzite and associated with latite dikes.	Additional 10M tons are below water table; mining done by contractor; crushing required; asphalt-lined heaps are cycled; 48 employees.
ROUND MOUNTAIN Nevada	Smokey Valley Mining Co. (50% Louisiana Land)	Started at 11.6M tons - 0.061 oz Au; now 195M tons - 0.043 oz Au, 0.08 oz Ag	$18M; startup Jan. 1977; 15 mo. for construction.	8000 tpd 75,000 oz Au/yr 30,000 oz Ag/yr	W/O=1/1; cutoff = 0.02 oz; 56° pit slope	67% in 26 days.	$190/oz (est. by FTG based on current ore grade of 0.045 oz Au).	Disseminated gold in iron-stained joints cutting dense weakly altered ash flow tuffs.	First large heap leach gold operation; crushing required; asphalt-lined heaps; heaps are cycled every 28 days; 240 employees.
VICTOR Colorado	Silver State Mining Corp.	3.5M tons - 0.06 oz. Au; potential for 8M tons.	$3M; startup March 1982; construction took 9 mo.	1000 tpd 20,000 oz Au/yr	NA	Close to 90% in 4 days.	$160/oz Au; mining cost is $.20/ton	400 ft. diameter breccia pipe.	Cripple Creek District; 75% of ore is mined by ripping; ore crushed to minus 1/2 in. and agglomerated with 90% of Au in solution by time cement cures; leaching in concrete "vats" covered for year-round leaching; 4-day cycle.
WEST END Idaho	Canadian Superior	3M tons - 0.085 oz Au.	Estimate $10M; startup July 1982; 11 mo. for construction.	3000 tpd	W/O=2/1	75%	NA	Oxidized gold-bearing arsenopyrite in shear zones in argillized schist, quartzite, and cargonate cut by trachyte dikes.	5-month operating season; 2 years to do EIS: 60 employees.
WINDFALL Nevada	~~Idaho Mining Corp.~~ Sold & returned to Idaho	5M tons - 0.035 oz Au 0.01 oz Ag	Startup July 1975.	1500 tpd	W/O - low; cutoff = 0.01 oz Au	NA	NA. Profitable in 1975 when gold was $160/oz (equal to $264/oz. Au in 1983).	Disseminated gold in jasperoidized and sanded dolomite.	No crushing; mining reduces sanded dolomite to 35% minus 200 mesh, ponds instead of sprinklers.
ZORTMAN-LANDUSKY Montana	Wharf Resources Ltd. Pegasus Explorations, Argo Gold Mining Corp., and others.	Zortman is 20M tons - 0.032 oz Au, 0.4 oz Ag; Landusky is 19M tons - 0.027 oz Au; 0.9 oz Ag.	Startup July 1979; construction took less than 12 mo.	12,000 tpd each; 67,000 oz Au and 146,000 oz Ag is total for both.	W/O=0.7/1	70% Au 35% Ag	$250/oz Au exclusive of Ag; equivalent to $4.50/ton ore including adminis., excluding deprec., depl., tax.	Disseminated gold and silver in weakly pyritic (now oxidized) shear zones cutting syenite. Zortman is old gold district, Landusky is old silver district; 8 miles apart.	Mining by contractor; heaps sealed with 2 ft. compacted bentonite; no crushing; 500K-1M ton heaps are permanent; early Zortman grades were 0.06 oz Au; additional Zortman reserves 30-40M tons, possibly 190M tons at Landusky; 65 employees; 1983 production began Feb. 21, 1983.
SILVER MINING OPERATIONS									
CANDELARIA Nevada	60% Occidental; will sell to NERCO.	13M tons - 3.15 oz Ag 0.002 oz Au.	$30M; startup September 1980.	8000 tpd 1,700,000 oz Ag/yr. 9,200 oz Au/yr.	NA	40% in 60 days	NA	Argentiferous Fe and Mn oxides in formerly pyritic chert, sandstone, and shale cut by quartz veins and chalcedonic zones.	Only full-scale heap leach silver mine in world; crushing and agglomerating required; closed Spring 1982 due to low Ag prices; 140 employees.
TOMBSTONE Arizona	Tombstone Exploration Company	Dumps contain 1M tons - 2.26 oz Ag and 0.037 oz Au. Lode averages 1 oz Ag.	Startup of dump material January 1981.	1500 tpd from dumps; 2000 tpd tpd from Contention zone.	NA	Lab tests indicate 75% for Au, 60% Ag both in 16 days.	Lode is profitable at Ag price of $10/oz.	Silver halides, sulfates in quartz-veined and silicified volcanics and siliceous limestone conglomerates, variable manganese.	Lode ore is mined by ripping; dump material crushed to minus 1/2" and agglomerated; leach pad is compacted mill tailings; agglomeration increased recoveries by 5x.
TUSCARORA Nevada	Tuscarora Assoc.	500,000 in dumps 2.5 oz Ag, 0.02 oz Au.	$700,000; startup October 1979; cost includes moving first 200,000 tons.	NA	NA	NA	$6.90/ton of ore.	Silver halides and gold in dumps in an old epithermal quartz-adularia vein camp.	Plastic-lined leach pads; plant treats 1400 tons solution/day; mining by contractor; 6 employees.

FTG - 5/26/83

WK copy

ASARCO

Southwestern Exploration Division

June 8, 1983

To: JHCourtright
FRKoutz
HGKreis
WLKurtz
GJStathis
JRStringham

From: J. D. Sell

Heap Leaching Gold

Attached is a copy of the cover letter by R. L. Brown and the article by F. T. Graybeal on the heap leaching of gold.

Please note the final paragraph on page 2 of the cover letter.

Although field work and a vast number of drill holes with ore is the prerequisite for an operating mine, the ASARCO Library also has a number of publications which have pertinent thoughts on gold. These include:

a) Principal Gold-Producing Districts of the United States, by A. H. Koschmann and M. N. Bergendahl: USGS Prof. Paper 610 (1968).

b) The Geochemistry of Gold and Its Deposits, by R. W. Boyles: Geol. Survey of Canada Bulletin 280 (1979).

c) Precious Metals in the Northern Cordilleran, edited by A. A. Levinson: The Assoc. of Exploration Geochemists (1982).

d) The Geology of Gold in Ontario, edited by A. C. Colvine: Ontario Geological Survey, Miscellaneous Paper 110 (1983).

James D. Sell

James D. Sell

JDS/cg

Attachments

J. D. Sell

New York, May 31, 1983

MEMORANDUM FOR: Exploration Managers

Heap Leaching - Gold

Five years ago most of us considered that heap leaching operations in gold camps were mainly conducted by underfinanced, fly-by-night groups attempting to salvage remnants of surface oxidized surface material and old dumps around some of the various abandoned gold mining localities in the west. The view was that there was no real money to be made from this activity, but that these junior companies might make something promoting their operations on various over-the-counter and regional stock exchanges. Since that time, recognition that such organizations as Selection Trust, Goldfields, Canadian Superior, and Placer Development among the experienced mining companies; and Louisiana Land, Amoco Minerals, and Occidental as examples of very-well financed groups had made investments in heap leach operations, and were deriving from them handsome profits and cash flows, obviously necessitate a re-evaluation of that view.

Mr. Graybeal has put together a compilation of data concerning the heap leach operations and it, together with his observations concerning the state of the "heap leaching" art, is in my opinion a valuable piece of background information which I am herewith distributing to you.

The advent of heap leaching as a viable metallurgical recovery technique quite obviously has added a new class of mineral deposit - the class of deposits containing pervasively distributed free gold grading say from .04 gold to those deposits grading say .15 Au (which might be more economic under conventional milling techniques - to our list of legitimate exploration targets. There are of course, limitations - not all .04 gold deposits are heap leachable, one reason being that in some of these deposits the gold is caught up in pyrite or in some other sulfide, and the development of this technique obviously does not constitute a basis for the assumption that all areas in which there is presumptive very low grade mineralization of these grades should be drilled. Nevertheless, new technology has opened up a new group of mineral deposits which can earn attractive rates of return, and mining operations on these deposits are in operation and are now continuing to be constructed in the Western United States and in Australia, and operations of this type are either under construction or being contemplated in the Philippines, and in South Africa.

RECEIVED
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S. W. U. S. EXPL. DIV.

In summary I think you will find the information Mr. Graybeal has compiled very useful to you, and I ask that you circulate it among your staff and make sure they understand both the possibilities and the limitations inherent in heap leaching.

R. L. Brown

Attachment

cc w/attachment

RLHennebach
RdeJOsborne
TCOsborne

New York, May 25, 1983

MEMORANDUM FOR: Mr. R. L. Brown

Review of Gold and Silver
Heap Leach Operations

Heap leaching is a simple process in which a cyanide solution is percolated through heaps of gold-bearing rock dissolving gold which is usually recovered on activated carbon. Geological factors of importance include distribution and grain size of gold in the ore, supergene oxidation which liberates sulfide-bound gold and reduces consumption of cyanide, and physical nature of the ore which governs permeability in the leach heaps.

The first significant operation to recover gold by heap leaching began at Cortez in 1971 as an addition to a conventional milling operation. The first significant independent heap leach operation began at Windfall in 1975 and there are presently 12 independent heap leach gold mines. I estimate that 320,000 oz. Au, or roughly 20% of the total USA gold production, was recovered by heap leaching in 1982 equivalent to about half of the primary open pit gold produced in the USA.

The attached fold-out chart summarizes information on 15 significant precious metal heap leach operations. Data are incomplete, change almost daily, and come from many sources of variable quality. Twelve of the 15 deposits are gold mines of which 6 are in Nevada and only 1 is outside the USA. Seven of the 15 are run by junior mining companies. Five additional operations for which only incomplete data are available are described briefly below. The following discussion will focus on the gold operations.

Reserve data in Table 1 suggest that at a gold price of $400/oz. a minimum exploration target size would be about 1 million tons averaging 0.08 oz. Au. Minimum ore grade for a large and low cost operation with recoverable gold and silver might be about 0.040 oz. Au equivalent. The mineralization in heap leach ores is usually disseminated with assay boundaries so reserves may change substantially with the price of gold. Exceptional care is required to measure the gold content of heap leach material which is near the lower limit of acceptable fire assay precision. Increased confidence in reserve estimates is obtained by running duplicate assays, assaying large samples (10-20 lbs.), drilling large diameter holes, and drilling a large number of holes.

TABLE 1, Summary of Heap Leach Reserves

Mine	Tons (millions)	oz. Au	oz. Ag
Haveluck (1)	2	0.054	-
Borealis	2	0.08	0.5
West End	3	0.085	-
Windfall	5	0.035	-
Alligator (2)	5	0.12	-
Annie Creek	7	0.043	0.16
Ortiz	7-16	0.053	-
Northumberland	10-16	0.052	-
Round Mountain	12-230	0.043	-
Zortman-Landusky	40-190	0.030	0.7

(1) Additional carbon-in-pulp ore was included in overall reserve which justified development.

(2) Highest grade heap leach operation; considering that carbonaceous ore must be stockpiled and additional reserves are present, this might have been better suited to conventional milling.

Major capital cost variables are the need for a crushing plant and whether mining is done by contractor. Capital costs in 1983 dollars for heap leach operations including Candelaria (Ag) ranged from $2300-6600/daily ton of ore for 6 of the operations where good data are available (see Table 2). Alligator Ridge was $15,000/daily ton and may be in error. Capital cost for the Zortman-Landusky complex are not known accurately, but if estimates of $1 million (1983 dollars) are close the cost would be $200/daily ton. This remarkably low cost resulted from contract mining of ore which did not require crushing. Table 2 shows that heap leach capital costs are substantially less than those for more conventional milling operations due to a relatively simple flowsheet. Capital costs per ounce of recoverable gold for heap leaching are equivalent to or less than costs for conventional milling as shown on Table 3.

TABLE 2. Capital Cost/Daily Ton of Ore Mined (1983 dollars)

Heap Leach		Conventional Milling	
Haveluck	- $2,300	Golden Sunlight	-$10,000
Santa Clara[1]	- 2,700	Pinson	- 22,000
Annie Creek	- 2,900	Mercur	- 27,000
Round Mountain	- 3,500	Jerritt Canyon	- 44,000
Candelaria	- 4,300	McLaughlin	- 66,000
Ortiz	- 6,300		
Northumberland	- 6,600	Average	-$34,000
Alligator Ridge	- 15,000		
Average	$ 5,400		

(1) Not on fold out table; see text

TABLE 3. Capital Cost/oz. of Recoverable Gold in Reserve (1983 dollars)

Heap Leach		Conventional Milling	
Haveluck[1]	- $16/oz.	Golden Sunlight	- $43/oz.
Annie Creek	- 18	Jerritt Canyon	- 49
Santa Clara	- 23	Pinson	- 63
Northumberland	- 58	McLaughlin	- 69
Round Mountain	- 59	Mercur	- 99
Ortiz	- 67		
Alligator Ridge	- 83		

(1) Heap leach and CIP reserve and plant.

Capital cost per ounce of recoverable gold in the original reserve for the Zortman-Landusky complex would be less than $1.00.

Construction intervals vary from 8-15 months. Six of the operations began production in the interval 1979-1981. Mining rates (ore) for 8 operations cluster between 1000-3000 tpd. The two largest operations are Round Mountain at 8000 tpd and Zortman-Landusky at 25,000 tpd. Round Mountain produces more gold than Zortman due to a longer operating season and higher grades. Although cold weather is a problem for plumbing systems and reduces the rate of dissolution of gold Zortman-Landusky, which is located in northern Montana, resumed mining on February 21, 1983, indicating cold weather is not a complete deterrent. The Victor Mine leaches heaps in enclosed vats at Cripple Creek and Round Mountain recently installed a heater to keep circulating fluids warmer. Because of the low ore grades stripping ratios are low and pit slopes are kept steep where possible. Cutoff grades are about 0.02 oz. Au.

Gold recoveries are in the range 60-70% for a 30-day interval. The most important and unpredictable variable in heap leaching is the movement of solution through the heaps. In order to obtain any recovery a cyanide solution must be brought in contact with the gold. Where the gold occurs on natural fractures crushing may not be required. If crushing is required or if the ore is unusually soft, excessive fines may reduce permeability and cut the flow of solution below that required to maintain production. Fines have been a problem at Alligator Ridge, Haveluck, and Candelaria. Agglomeration techniques have been developed by the U.S. Bureau of Mines to handle fines, but they increase capital and operating costs. Proper bedding of material on the leach heaps has been extensively researched to avoid compacting the ore with haulage equipment and to reduce sorting of fine and coarse material as individual loads of ore are added to the heaps. Careful selection of samples for metallurgical work is necessary as a single orebody may have several distinct ore types. Recognition of different ore types is difficult because most low grade deposits are explored using air hammer drilling methods which pulverize the sample, but careful logging can provide useful data. Most companies construct one or several 1000-5000 ton test heaps before final operating procedures are selected. The importance of accurate measurements of ore grades and recoveries cannot be overstated; for a 3000 tpd operation with $270/oz Au operating costs at a gold price of $430/oz., a change in recovered gold of six thousandths of an ounce per ton of ore (0.006 oz. Au) will change the annual net profit before taxes by $1 million.

Operating costs per ounce of gold produced seem to cluster at $250-270/oz. The Round Mountain estimate of $190/oz. Au is the 1983 GNP adjusted equivalent of estimates made in 1977 by Copper Range and Mountain States using a head grade of 0.045 oz. Au and is probably too low. Costs at Northumberland are unusually high at $402/oz., even when $68/oz. haulage costs are deducted.

Heap leach operations for which less data are available include:

1) Carlin Mining Co. Heap leaching is now in progress at Carlin. Major reserves at Maggie Creek and Gold Quarry of at least 150 million tons averaging 0.036 oz. Au will be agglomerated and heap leached. The Bootstrap dumps were previously leached as they lay after mining without prior sealing of the base. By 1984 it is estimated that 30% of the gold production at the Carlin complex (Carlin, Maggie, Gold Quarry, Rain, Blue Star) will be by heap leaching.

2) Picacho District, Calif. Chemgold, Inc. (Glamis Gold, Inc.) is leaching 300,000 tons and producing 600 oz. Au/mo. Agglomeration techniques are being studied. Recent drilling expanded reserve to 10 million tons averaging 0.05 oz. Au. Mineralization rumored to occur in mid-Tertiary detachment fault zones.

3. Santa Clara, Costa Rica. Located 40 miles from San Jose and owned 60% by United Hearne and 40% by Canadian Barranca. Reserves are 4.1 million tons averaging 0.063 oz. Au and recent drilling indicates substantial increases are probable. Came on stream September 1981 at 1500 tpd for $4 million. Operating costs are estimated at $6.00/ton.

4) Darwin Mine, Calif. Anaconda is drum agglomerating and heap leaching flotation tails averaging 1.6 oz. Ag and 0.01 oz. Au. Leach pads are compacted tailings.

5) Grubstake Mine, Montana. Resources America, Inc. is drum agglomerating and heap leaching 20,000 tons of dump material averaging 0.07-0.15 oz. Au. The Merrill-Crowe recovery plant is highly automated

Serious feasibility studies or pilot heap leach test work is ongoing at:

1) Pinson mine, Nev. A test pad is being built to study heap leach characteristics of 5 million tons of low grade material involving two ore types.

2) Bell Mtn. mine, Fairview Dist., Nev. American Pyramid is developing a combined milling and heap leach operation.

3) Dayton mill tailings, Comstock Lode, Nev. Tailings average 2.5 oz. Ag and are 65% minus 200 mesh. A 40-ton experimental heap was agglomerated and yielded 78% recovery in Ag in 5 days.

4) Packard mine, Nev. This property is located 3 miles southwest of the Rochester deposit. Reserve is estimated at 2 million tons averaging 1 oz. Ag. An initial unagglomerated 30,000 ton test was unsuccessful, but subsequent 300 ton pilot heaps with ore crushed to 80% minus 9/16 in. and agglomerated gave 65% Ag recovery in 6 days. Follow-up work on a 1000 ton heap gave similar results and work is continuing.

5) Goldfield, Nev. Initial 8000 ton test heap failed due to low permeability. Column leach tests on agglomerated rock crushed to minus 1 in. gave 80% recovery of gold in 14 days. Existing heaps will be recrushed and agglomerated as a pilot test.

6) El Plomo, Colo. E&B Explorations (German money) is evaluating 1 million tons of 0.057 oz. Au. Large test heap program was completed in 1982.

7) Gold Stripe, Plumas County, Calif. Test production by NCA Minerals will begin June 1983 from 10,000 ton test pad. Deposit has reserves 1.2 million tons averaging 0.063 oz. Au.

8) Zaca Mine, Alpine County, Calif. California Silver has operating permits for a 3000 tpd/heap leach mine. Reserves are 4.1 million tons of 0.036 oz. Au and 0.61 oz. Ag at 0.01 oz. Au cutoff. Column leach test of combined oxide and sulfide ore crushed to minus 3/8" recovered 76% Au and 38% Ag in 19 days.

9) German Gulch, Montana. Reserve is 11 million tons of 0.048 oz. Au. Heap leach recoveries on two 4500 ton heaps are reportedly 85%.

10) La Ronge Dist., Saskatchewan. Waddy Lake Resources has conducted tests said to be successful on low grade rock containing coarse gold.

11) Hedley, British Columbia. Banbury Gold Mines has applied for permit to begin heap leach production on gold-silver property.

12) Telfer mine, W. Australia. Newmont has completed a 1000 ton test heap leach pad and will crush ore to minus 1" and agglomerate. They say there is considerable heap leach ore, my guess is at least 5 million tons.

13) Masbate mine, Philippines Is. Atlas is building an asphalt heap leach test pad to evaluate characteristics of an eventual 4 million ton stockpile grading 0.030-0.045 oz. Au.

14) Gabiet Mine, Sudan. Robertson Research (U.K.) in complex ownership package is conducting pilot heap leach tests on mine tailings. Ultimate production might be 3000/oz. Au/yr.

15) Witwatersrand S. Africa. Feasibilities are reportedly underway to evaluate heap leaching on old dumps and tailings.

Heap leach operations are attractive because capital costs are low, technology is simple, and development intervals are rapid. These features permit the development of deposits otherwise too small or too low grade to justify a conventional milling operation. A substantial body of literature is available on the technology, but less information is available on existing operations, particularly operating costs. Operating costs per ounce of gold recovered may exceed costs at open pit milling operations, but are less than costs at the Homestake mine (1982 costs about $400/oz.). Although heap leach operating costs/oz. Au may be somewhat more than conventional milling operations, the capital investment may be somewhat less, and heap leaching therefore represents a valid and competitive mining investment. A 3000 tpd heap leach plant mining 0.07 oz. Au with 65% recovery at a gold price of $430/oz. and operating cost of $270/oz. would net $8 million before tax, clearly of sufficient size to be of interest to Asarco.

The many low grade gold prospects in various stages of exploration now known in the western US substantially outnumber known prospects of higher grade mineralization which might become conventional milling operations. This ratio may also apply to future discoveries and a significant effort is currently underway in low grade environments where the odds for discovery would appear to be more attractive.

F. T. Graybeal

F. T. Graybeal

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ASARCO

MINERAL
NOV 28 1983
BENEFICIATION DEPT

Exploration Department
Frederick T. Graybeal
Chief Geologist

JRS-~~JMS~~ → WLK File Gold Bulk/Heap

November 18, 1983

Mr. D. R. Crowell, Director
Mineral Beneficiation Dept.
Tucson Office

Western Australia
Heap Leach Operations

Dear Mr. Crowell:

Attached is a memorandum with tables prepared by Mr. Williamson which summarize certain aspects of heap leach operations in Western Australia. This information is for your general background although it might be of interest to both Mr. Kurtz and Mr. Anzalone.

Very truly yours,

F. T. Graybeal

F. T. Graybeal

Attachment

cc: HCWilliamson

D. E. C.
NOV 28 1983

F.T. Graybeal,
Chief Geologist
NEW YORK OFFICE

F.T.G.

NOV 18 1983

NOV 16 1983

EXPLORATION DEPT.

Au Heap Leach and CIP Plants
Western Australia

Dear Mr. Graybeal,

Attached please find a copy of an article by Mr. Geldard, Manager of the Whim Creek operation at Meekatharra, from the AIMM Bulletin of July-August 1983 (No. 476-477).

I have used Mr. Geldard's figures in his table, plus other data I have obtained from the Annual Reports of various companies together with press reports and notes on discussions with various company people, to build up Tables 1 to 3. I have used the latest company published results in Tables 1 to 3 if Geldard's figures are different from reported statistics.

For comparison, statistics for the conventional Telfer (Newmont 70%, BHP 30%) and Kalgoorlie operations of Kalgoorlie Mining Associates, N. Kalgurli and CSR's Paringa operations, are included. These data will supplement your data on heap leach plants dated May 25, 1983, and may be useful background for our discussion with Western Alluvial Directors.

The recently published annual report of WMC confirms that its 100% owned subsidiary, Great Boulder Holdings Ltd., is operating at close to 2,000 tonnes/day at its Kambalda plant. The head grade improved from 2.44 g/t to 4.09 g/t for the year ending June 30, 1983. The open cut reserves at Victory are nearly exhausted and ore reserves decreased slightly to 4.1 Mt with open cut ore of 0.3 Mt at 6 g/t on the Great Boulder leases on the Golden Mile, and 0.2 Mt at 5.0 g/t at the Sand King about 75 km northwest of Kalgoorlie.

Also at WMC's 100% owned Lancefield operation (near the Windarra Ni mine) throughput has improved to 700 tons/day at an average head grade of 3.66 g/t. Of this, about 52% came from the Lancefield open cut but this ore source is nearing exhaustion. Lancefield reserves stand at 0.4 Mt grading 7.0 g/t.

WMC has a 19.6% interest in Kalgoorlie Lake View Pty Ltd (KLV) which, in turn, has a 52% interest in Kalgoorlie Mining Associates (KMA). Homestake has a 48% interest in KMA. KMA are the operators of gold mines at Mt Charlotte and on the Golden Mile.

WMC reports that at 30 June the Cassidy Shaft on the Mt Charlotte orebody operated by KMA had reached a vertical depth of 2800 ft. The planned depth of the Cassidy Shaft is 3500 ft. It will be completed and equipped by 1987 at a cost of A$36 M (1983 $). This shaft is reported by the Chairman of KMA to give access to about 40 M tons of ore grading 0.147 oz/ton below the Flanagan Fault. It is designed to produce 1.5 Mt per annum (present rate 0.9 Mt/annum). Mt Charlotte ore is non refractory and KMA is believe to be considering a new CPI plant to treat the ore.

The annual reports of the various companies contain much useful statistical and background data but I think they are summarized in the table. However, if you need further information, please advise.

Yours sincerely,

H.C. Williamson

Encs.

pa BE SER HF TM

ASARCO (AUSTRALIA) PTY. LTD. P.O. Box 6010, Hay Street East, PERTH, Western Australia 6000
Street East, PERTH. Telex: AA93955. Cables: Asarco. Telephone: (09) 325 9533

1 1A.H 7 before 2 September, 1983.

TABLE 1

SUMMARY OF RECENT ORE DISCOVERIES

Plant	Owner	Mine	Tons Million	Head Grade oz/t.	Comment
Callion	Centamin		0.045	0.11	Small heap leach.
Golden Mile	N. Kalgurli	Oxide Ore	0.23	0.154	
Big Bell	A.C.M.	Golden Crown	0.35	0.67	Shaft sinking starts Jan.84
Laverton	W.M.C.	Lancefield	0.40	0.23	1982/83
Marvel Loch	Jingellic	Nevoria	0.6	0.131	Start up July 1984.
Sir Samuel	Spargos	Bellevue	0.75	0.45	
Lake Grace	Otter	Griffin	0.8	0.12	Pilot Plant 125 tons/day.
Leonora	C.R.A.	Tower Hill	1.0	?	Start up Dec. 1983.
Horseshoe	Nolex 45 % Eastern 35.6% Samantha 19.4%	Horseshoe Lights	1.0	0.19	
Reedy's	Metana	Rand	1.6	0.131	Start up July 1984.
Gwalia	Sons of Gwalia	Gwalia	1.7	0.121	Start up ?
Porphyry	Pioneer 75% Poseidon 25%	Porphyry	2.4	0.18	Start up June 1984.
Kalgoorlie	Occidental 55% Black Hill 45%	3 Lodes	3.0	0.131	Press Report.
Leonora	Esso 55% Carr Boyd 40% Aztec 5%	Harbour Lights	3.0	0.163	Estimates.
Kalgoorlie	C.S.R. 80% Hampton 20%	Paringa	4.0	0.19	Purchase + construction costs.
Meekatharra	Whim Creek	Paddy's Flat	4.3	0.082	
Kambalda	W.M.C.	Victory Great Boulder Sand King Total Kambalda	4.61	0.17	
Paddington	Pancontinental	No. 1 No. 2	7.8	0.105	
Big Bell	A.C.M.	Big Bell	18.0	0.10	
Kalgoorlie	K.M.A.	Mt Charlotte	40.0	0.147	

TABLE 1A

SUMMARY OF TAILINGS RETREATMENT PLANTS

Plant at	Owner	Mine	Tons Million	Recovered Grade oz/t
Mt Morgan	Peko/Anglo	Mt Morgan, Q.	40.0	0.035
Wiluna	Western Alluvials/ Nolex	Wiluna	6.5	0.015
Leonora	Sons of Gwalia	Gwalia	6.0	0.033
Sir Samuel	Spargos	Lawlers	0.86	0.054

TABLE 2

CAPITAL COST/DAILY TON OF ORE MILLED
(A$1983) US$1.0 = A$0.9

Plant	Capital $M	Throughput t p d	Capital Cost/ Daily Ton A$	Comments
Mt Morgan	17.0	10000	1700	Actual cost of plant.
	28.0	10000	2800	Incl. purchase price + mine development.
Australis	1.0	500	2000	Presently being expanded.
Haveluck	1.8	640	2812	Presently being expanded.
Kambalda	6.0	1900	3157	
Reedy's	3.0(1)		5000	Cost of plant.
	5.0(2)	600	8300	Start upJune 1984.
N. Kalgurli	4.0	750	5300	
Lancefield	4.0	700	5714	
Horseshoe	9.0(3)	1000	9000	Throughput increasing.
	14.0(4)		14000	" "
Paddington	40.0(5)	2500	16000	Press Report
Porphyry	10.5(1)	600	17500	Start up June 1984.
Telfer	47.0(6)	1300	36000	Isolated open cut conventional plant.
Paringa	27.6	600	46000	Incl. purchase price + mine development.
Big Bell	190.0(7)	3000	63300	Press report includes underground development.
Tower Hill	N.A.			Start up Dec. 1983.

(1) Estimated treatment plant and service costs only.

(2) Includes exploration and acquisition costs.

(3) Based on purchase price of Zamu 45% interest by Nolex.

(4) Based on Nolex offer of $5.2M for AMI's 35.6% interest.

(5) Paddington's final feasibility being prepared prepared by Davey McKee will be ready in December 1983.

(6) Start up capital cost in 1977 was $27M.

(7) Estimate

It must be noted that the Kambalda and Lancefield CIP plants were built alongside existing nickel concentrators; hence their capital costs were considerably below the cost of new independent plant constructed away from services.

TABLE 3

CAPITAL COST/OZ OF RECOVERABLE GOLD IN RESERVES

Plant	Reserves oz of Au	Capital Cost $/oz	Comment
Kambalda	780,000	7.7	
Reedy's	500,000	6 10	Plant cost only Total cost
Metana	208,000	14.4	
Porphyry	432,000	24	
Telfer*	1,064,000	25.4	
Paringa	735,000	37	
Lancefield	92,000	43	Plant only
Horseshoe	190,000	47 73	Jan. 1983 Purchase Oct. 1983 Purchase
Paddington	830,000	48	
Big Bell	2,800,000	68	
N. Kalgurli	35,500	113	
Meekatharra	328,000	?	Plant expansion.

*Reserves 7 capital $27M at start up. Capital cost 1983 A$ and using present day ore reserves.

Alkali Gold Bibliography

LIST OF SELECTED REFERENCES PERTAINING TO EPITHERMAL PRECIOUS-METAL DEPOSITS, THE EVOLUTION OF MINERALIZATION IN THE SAN JUAN MOUNTAINS, AND THE MINERAL ALUNITE

Albers, J.P., and Cornwall, H.R., 1968, Revised interpretation of the stratigraphy and structure of the Goldfield district, Esmeralda and Nye Counties, Nevada (abs): Geol. Soc. America Spec. Paper 101, p. 285.

_____, and Kleinhampl, F.J., 1970, Spatial relation of mineral deposits to Tertiary volcanic centers in Nevada, in Geological Survey research 1970: U.S. Geol. Survey Prof. Paper 700-C, p. C1-C10.

Andrusenko, N.I., and Shchepot'yev, Yu. M., 1974, Temperatures and stages in the formation of central Kamchatka subvolcanic Au-Ag deposits: Geochemistry Int., vol. 11, p. 130-137.

Ashley, R.P., 1974, Goldfield mining district, in Guidebook to the geology of four Tertiary volcanic centers in central Nevada: Nevada Bur. Mines and Geology Rpt. No. 19, p. 49-66.

_____, and Albers, J.P., 1969, Elements associated with gold in the oxidized zone at Goldfield, Nevada (abs): Geol. Soc. America Spec. Paper 121, p. 480.

_____, and Albers, J.P., 1973, Distribution of gold and other ore-related elements near ore bodies in the oxidized zone at Goldfield, Nevada: U.S. Geol. Survey Open-File Rpt., 126 p.

_____, and Keith, W.J., 1973, Occurrences of alunite in hydrothermally altered rocks and ores at Goldfield, Nevada (abs): Geol. Soc. America Abs. with Programs, vol.5, vol. 1, p. 6.

_____, and Silberman, M.L., 1976, Direct dating of mineralization at Goldfield, Nevada, by potassium-argon and fission-track methods: Econ. Geol., vol. 71, p. 904-924.

Barton, M.D., 1980, The Ag-Au-S system (abs): Econ. Geol., vol. 75, p. 303-316.

Barton, P.B., Jr., Bethke, P.M., and Roedder, E., 1977, Environment of ore deposition in the Creede mining district, San Juan Mountains, Colorado: Part III. Progress toward interpretation of the chemistry of the ore-forming fluid for the OH vein: Econ. Geol., vol. 72, p. 1-24.

Bejnar, W., 1957, Lithologic control of ore deposits in the southwestern San Juan Mountains, in New Mexico Geol. Soc. Guidebook, 8th Field Conf., Southwestern San Juan Mountains, Colorado, 1957, p. 162-173.

Bethke, P.M., and Barton, P.B., Jr., 1971, History of filling of the OH vein, Creede, Colorado (abs): Econ. Geol., vol. 66, p. 1265.

_____, Barton, P.B., Jr., Lanphere, M.A., and Steven, T.A., 1976, Environment of ore deposition in the Creede mining district, San Juan Mountains, Colorado: II. Age of mineralization: Econ. Geol., vol. 71, p. 1006-1011.

_____, Barton, P.B. Jr., and Rye, R.O., 1973, Hydrogen, oxygen, and sulfur isotopic compositions of ore fluids in the Creede district, Mineral County, Colorado (abs): Econ. Geol. vol. 68, p. 1205.

Boyle, R.W., 1968, The geochemistry of silver and its deposits: Canada Geol. Survey Bull. 160, 264 p.

_____, 1979, The geochemistry of gold and its deposits: Geol. Survey of Canada, Bull. 280, 584 p.

Brannock, W.W., Fix, P.F., Gianella, V.P., and White, D.E., 1948, Preliminary geochemical results at Steamboat Springs, Nevada: Trans. American Geophysical Union, vol. 29, no. 2, p. 211-226.

Brophy, G.P., Scott, E.S., and Snellgrove, R.A., 1962, Sulfate studies II, solid solution between alunite and jarosite: Am. Mineralogist, vol. 47, p. 112-126.

Browne, P.R.L., 1969, Sulfide mineralization in a Broadlands geothermal drill hole, Taupo volcanic zone, New Zealand: Econ. Geol., vol. 64, p. 156-159.

_____, 1971, Mineralization in the Broadlands geothermal field, Taupo volcanic zone, New Zealand, in Proceedings of the 1970 IMA-IAGOD Mtgs: Soc. Mining Geol. Japan, Spec. Issue 2, p. 64-75.

Buchanan, L.J., 1979, The Las Torres mine, Guanajuato, Mexico, ore controls of a fossil geothermal system: Ph.D. Thesis, Colo. School Mines, 138 p.

_____, 1980, Ore controls of vertically stacked deposits, Guanajuato, Mexico: SME-AIME Meeting, Feb. 1980, Las Vegas, Nevada.

Bundy, W.M., 1958, Wall rock alteration in the Cochiti mining district, New Mexico: New Mexico Bur. Mines and Minerals Res. Bull. 59, p. 1-71.

_____, and Murray, H.H., - , Argillization in the Cochiti mining district, New Mexico: New Mexico Inst. of Mining and Tech., Reprint Series, No. 4, p. 342-368.

Burbank, W.S., 1932, Geology and ore deposits of the Bonanza mining district: U.S. Geol. Survey Prof. Paper 169, 166 p.

_____, 1933, Manganese minerals of the Sunnyside veins, Eureka Gulch, Colorado: Am. Mineralogist, vol. 18, no. 12, p. 513-527.

_____, 1940, Structural control of ore deposition in the Uncompahgre district, Ouray County, Colorado, with suggestions for prospecting: U.S. Geol. Survey Bull. 906-E, p. 189-265.

_____, 1950, Problems of wall rock alteration in shallow volcanic environments: Colo. School Mines Quart., vol. 45, p. 286-316.

_____, 1951, The Sunnyside, Ross Basin, and Bonita fault systems and their associated ore deposits, San Juan County, Colorado: Colo. Sci. Soc. Proc., vol. 15, no. 7, p. 285-304.

_____, 1960, Pre-ore propylitization, Silverton caldera, Colorado: U.S. Geol. Survey Prof. Paper 400-B, p. B12-B13.

Callaghan, Eugene, 1938, Preliminary report on the alunite deposits of the Marysvale region, Utah: U.S. Geol. Survey Bull. 886-D, p. 91-134.

Casadevall, T., and Ohmoto, H., 1977, Sunnyside mine, Eureka mining district, San Juan County, Colorado: Geochemistry of gold and base metal ore deposition in a volcanic environment: Econ. Geol. vol. 72, p. 1285-1320.

Coats, R.R., 1940, Propylitization and related types of alteration on the Comstock Lode, Nevada: Econ. Geol. vol. 35, no. 1, p. 1-16.

Dreier, J.E., 1976, The geochemical environment of ore deposition in the Pachuca-Real del Monte district, Hidalgo, Mexico: Unpub. Ph.D. thesis, Univ. of Arizona, 115 p.

Einaudi, M.T., 1977, Environment of ore deposition at Cerro de Pasco, Peru: Econ. Geol. vol. 72, p. 893-924.

Emmons, W.H., and Larson, E.S., Jr., 1913, A preliminary report on the geology and ore deposits of Creede, Colorado: U.S. Geol.Survey Bull. 530, p. 42-65.

_____, 1923, Geology and ore deposits of the Creede district, Colorado: U.S. Geol. Survey Bull. 718, 198 p.

Ewers, G.W., and Keays, R.R., 1977, Volatile and precious metal zoning in the Broadlands geothermal field, New Zealand: Econ. Geol., vol. 72, p. 1337-1354.

Fernandez, H.E., and Damasco, F.V., 1979, Gold deposition in the Baguio gold district and its relationship to regional geology: Econ. Geol., vol. 74, p. 1852-1868.

_____, Damasco, F.V., and Sangalang, L.A., 1979, Gold ore shoot development in the Antamok mines, Philippines: Econ. Geol., vol. 74, p. 606-627.

Field, C.W., 1966, Sulfur isotopic method for discriminating between sulfates of hypogene and supergene origin: Econ. Geol., vol. 61, p. 1428-1435.

Forester, R.W., and Taylor, H.P., 1972, Oxygen and hydrogen isotope data on the interaction of meteoric ground waters with a gabbro-diorite stock, San Juan Mountains, Colorado: Intern. Geol. Cong., 24th, Montreal 1972, Sec. 10, Geochemistry, p. 254-263.

Graton, L.C., and Bowditch, S.I., 1936, Alkaline and acid solutions in hypogene zoning at Cerro de Pasco: Econ. Geol., vol. 31, p. 651-698.

Gross, W.H., 1975, New ore discovery and source of silver-gold veins, Guanajuato, Mexico: Econ. Geol., vol. 70, no. 7, p. 1175-1189.

Harvey, R.D., and V italiano. C.J., 1964, Wall rock alteration in the Goldfield district, Nevada: Jour. Geology, vol. 72, p. 564-579.

Hattori, K., 1975, Geochemistry of ore deposition at the Yatani lead-zinc and gold-silver deposit, Japan: Econ. Geol., vol. 70, p. 677-693.

Hemley, J.J., Hostetler, P.B., Gude, A.J., and Mountjoy, W.T., 1969, Some stability relations of alunite: Econ. Geol., vol. 64, p. 599-612.

_____, and Jones, W.R., 1964, Chemical aspects of hydrothermal alteration with emphasis on hydrogen metasomatism: Econ. Geol., vol. 59, p. 538-569.

Hollister, V.F., and Entwistle, L.P., 1977, The Milluachaqui epithermal silver district of northern Peru: Min. Dep., vol. 12, p. 235-238.

Howland, A.L., 1940, Specularite-alunite mineralization at Hickeys Pond, Newfoundland: Am. Mineralogist, vol. 25, p. 34-45.

Izett, G.A., 1973, Late Tertiary sedimentation and deformation in northern Colorado and adjoining areas (abs): Geol. Soc. America Abstracts with Programs, vol. 5, no. 6, p. 487-488.

Jensen, M.L., Ashley, R.P., and Albers, J.P., 1971, Primary and secondary sulfates at Goldfield, Nevada: Econ. Geol., vol. 66, p. 618-626.

Kamilli, R.J., and Ohmoto, H., 1977, Paragenesis, zoning, fluid inclusion, and isotopic studies of the Finlandia vein, Colqui district, central Peru: Econ. Geol., vol. 72, no. 6, p. 950-982.

Knight, J.E., 1977, A thermochemical study of alunite, enargite, luzonite and tennantite deposits: Econ. Geol., vol. 72, p. 1321-1336.

Lang, Barbu, 1979, The base metals-gold hydrothermal ore deposits of Baia Mare, Romania: Econ. Geol., vol. 74, p. 1336-1351.

Learned, R.E., 1966, The solubility of quartz, quartz-cinnabar, and cinnabar-stibnite in sodium sulfide solutions and their implications for ore genesis: Ph.D. thesis, Univ. of Calif., Riverside, 161 p.

Lipman, P.W., and Mehnert, H.H., 1969, Structural history of the eastern San Juan Mountains and the San Luis Valley, Colorado, in Abstracts for 1968: Geol. Soc. America Spec. Paper 121, p. 525-526.

_____, 1975, Late Cenozoic basaltic volcanism and development of the Rio Grande depression in the southern Rocky Mountains, in Curtis, B.F., ed., Cenozoic history of the Southern Rocky Mountains: Geol. Soc. America Memoir 144.

_____, Steven, T.A., and Mehnert, H.H., 1970, Volcanic history of the San Juan Mountains, Colorado, as indicated by potassium-argon dating: Geol. Soc. America Bull., vol. 81, p. 2329-2351.

_____, Steven, T.A., Luedke, R.G., and Burbank, W.S., 1973, Revised volcanic history of the San Juan Uncompahgre, Silverton, and Lake City calderas in the western San Juan Mountains, Colorado: U.S. Geol. Survey Journ. Research, vol. 1, no. 6, p. 627-642.

_____, Fisher, F.S., Mehnert, H.H., Naeser, C.W., Luedke, R.G., and Steven, T.A., 1976, Multiple ages of mid-tertiary mineralization and alteration in the western San Juan Mountains, Colorado: Econ. Geol., vol. 71, p. 571-588.

Locke, Augustus, 1912a, The ore deposits of Goldfield - I: Eng. and Mining Jour., vol. 94, no. 17, p. 797-802.

_____, 1912b, The ore deposits of Goldfield - II: Eng. and Mining Jour., vol. 94, no. 18, p. 843-849.

Lodder, W., Gold-alunite deposits and zonal wall rock alteration near Rodalquilar, S.E., Spain, 1966: Geol. Inst., Mededelingen No. 318, Univ. van Amsterdam, 93 p.

Mayor, J.N., and Fisher, F.S., 1972, Middle tertiary replacement ore bodies and associated veins in the northwest San Juan Mountains, Colorado: Econ. Geol., vol. 67, p. 214-230.

McKee, E.H., 1976, Origin of the McDermitt caldera in Nevada and Oregon and related mercury deposits: AIME preprint 76-S-83, presented at the 1976 AIME annual meeting, Las Vegas, Nevada, 12 p.

_____, and Greene, R.C., 1975, Chronology of volcanism, tectonism, and mineralization of the McDermitt caldera, Nevada-Oregon: Geol. Soc. America Abs. with Prog., vol. 7, no. 5, p. 629-630.

Mehnert, H.H., Lipman, P.W., and Steven, T.A., 1973a, Age of mineralization at Summitville, Colorado, as indicated by K-Ar dating of alunite: Econ. Geol., vol. 68, p. 399-401.

Nash, J.T., 1975, Fluid inclusion studies of vein, pipe and replacement deposits, northwestern San Juan Mountains, Colorado: Econ. Geol., vol. 70, p. 1448-1462.

Newhouse, W.H., (ed.), 1969, Ore deposits as related to structural features: Hafner Pub. Co., New York, 280 p.

Nolan, T.B., 1935, The underground geology of the Tonopah mining district, Nevada: Nevada Bur. Mines Bull. No. 23.

Norton, D.L., 1964, Geological and geochemical investigations of stibnite deposits: Ph.D. thesis, Univ. of Calif., Riverside, 61 p.

O'Neil, J.R., Silberman, M.L., Fabbi, B.P., and Chesterman, C.W., 1973, Stable isotope and chemical relations during mineralization in the Bodie mining district, Mono County, California: Econ. Geol., vol. 68, p. 765-784.

_____, and Silberman, M.L., 1974, Stable isotope relations in epithermal Au-Ag deposits: Econ. Geol., vol. 69, p. 902-909.

Osborne, D.H., 1963, Arsenic as a direct and indirect geochemical guide to epithermal precious metal deposits: M.S. thesis, Univ. of Nevada, Reno. 69 p.

Petersen, U., Noble, D.C., Arenas, M.J., and Goodell, P.C., 1977, Geology of the Julcani mining district, Peru: Econ. Geol., vol. 72, p. 931-949.

Raab, W.J., and Dickson, F.W., 1967, Solubility of stibnite, realgar and orpiment in alkaline solutions below 100°C. and 1 bar (abs): Geol. Soc. America Annual Mtg. Program, p. 180.

Ransome, F.L., 1909, Geology and ore deposits of Goldfield, Nevada; U.S. Geol. Survey Prof. Paper 66, 258 p.

_____, 1910a, Geology and ore deposits of the Goldfield district, Nevada, Part I-Geology: Econ. Geol., vol. 5, no. 4, p. 301-311.

_____, 1910b, Geology and ore deposits of the Goldfield district, Nevada. Part II-Mines and mining: Econ. Geol., vol. 5, no. 5, p. 438-470.

Raymahashay, B.C., 1968, A geochemical study of rock alteration by hot springs in the Paint Pot Hill area, Yellowstone Park: Geochim. et Cosmochim. Acta, vol. 32, p. 439-522.

Roedder, E., 1967, Fluid inclusions as samples of ore fluids, in Barnes, H.L. ed., Geochemistry of hydrothermal ore deposits: New York, Wiley-Interscience, p. 465-514.

Roper, M.W., 1976, Hot springs mercury deposition at McDermitt mine, Humboldt County, Nevada, AIME preprint 76-S-28, paper presented at Annual Meeting, Las Vegas, Nevada, Feb. 22-26, 14 p.

Rye, R.O., and Sawkins, F.J., 1974, Fluid inclusion and stable isotope studies on the Casapalca Ag-Pb-Zn-Cu deposit, Central Andes, Peru: Econ. Geol., vol. 69, no. 4, p. 181-205.

Rytuba, J.J., 1976, Geology and ore deposits of the McDermitt caldera, Nevada-Oregon: U.S. Geol. Survey Open File Report 76-535, 9 p.

Sawkins, F.J., O'Neil, J.R., and Thompson, J.M., 1979, Fluid inclusion and geochemical studies of gold vein deposits, Baguio district, Philippines: Econ. Geol., vol. 74, no. 6, p. 1420-1434.

Searls, Fred, Jr., 1948, A contribution to the published information on the geology and ore deposits of Goldfield, Nevada: Nevada Univ. Bull., vol. 42, no. 5 (Geol. and Mining Ser. 48), 24 p.

Silberman, M.L., Chesterman, C.W., Kleinhampl, F.J., and Gray, C.H., Jr., 1972, K-Ar ages of volcanic rocks and gold-bearing quartz-adularia veins in the Bodie mining district, Mono County, California: Econ. Geol., vol. 67, p. 597-604.

_____, and White, D.E., 1975, Limits on the duration of hydrothermal activity at Steamboat Springs, Nevada, by K-Ar ages of spatially assoicated altered and unaltered volcanic rocks (abs): Geol. Soc. America Abstracts with Programs, vol. 7, no. 7, p. 1272.

Slack, J.F., 1976, Hypogene zoning and multistage vein mineralization in the Lake City area, western San Juan Mountains, Colorado: Unpub. Ph.D. thesis, Stanford, 327 p.

Smith, D.M., Jr., 1978, The significance of silver-gold ratios at the Tayoltita mine, Durango, Mexico: SME of AIME preprint 78-L-53, 11 p.

_____, 1978, Buckskin mining district, Humboldt County, Nevada: ASARCO private report, 17 p.

_____, Albinson, T.D., and Sawkins, F.J., 1979, Geologic and fluid inclusion studies of the Tayoltita silver-gold vein deposit, Durango, Mexico: MS submitted to Econ. Geol., 41 p.

Spurr, J.E., 1905, The ores of Goldfield, Nevada, in Contributions to economic geology, 1904: U.S. Geol. Survey Bull. 260, p. 132-139.

Steiner, A., 1963, Hydrothermal rock alteration at Wairakie, New Zealand: Econ. Geol., vol. 48, p. 1-13.

Steven, T.A., 1969, Possible relation of mineralization to thermal springs in the Creede district, San Juan Mountains, Colorado: A discussion: Econ. Geol. vol., 64, p. 696-698.

_____, 1975, Middle tertiary volcanic field in the southern Rocky Mountains: Geol. Soc. America Mem. 144, p. 75-94.

_____, and Eaton, G.P., 1975, Environment of ore deposition in the Creede mining district, San Juan Mountains, Colorado: I, Geologic, hydrologic and geophysical setting: Econ. Geol., vol. 70, p. 1023-1037.

_____,and Epis, R.C., 1968, Oligocene volcanism in south-central Colorado, in Epis, R.C., ed., Cenozoic volcanism in the southern Rocky Mountains: Colorado School Mines Quart., vol. 63, p. 241-258.

_____, and Friedman, I., 1968, The source of travertine in the Creede Formation, San Juan Mountains, Colorado: U.S. Geol. Survey Prof. Paper 600-B, p. B29-B36.

_____, and Lipman, P.W., 1976, Calderas of the San Juan volcanic field, southwestern Colorado: U.S. Geol. Survey Prof. Paper 958, 35 p.

_____, Luedke, R.G., and Lipman, P.W., 1974, Relation of mineralization to calderas in the San Juan volcanic field, southwestern Colorado: U.S. Geol. Survey Journ. Research, vol. 2. p. 405-409.

_____, Mehnert, H.H., and Obradovich, J.D., 1967, Age of volcanic activity in the San Juan Mountains, Colorado, in Geological Survey Research 1967: U.S. Geol. Survey Prof. Paper 575-D, p. D47-D55.

_____, and Ratté, J.C., 1960, Geology and ore deposits of the Summitville district, San Juan Mountains, Colorado: U.S. Geol. Survey Prof. Paper 343, 70 p.

_____, and Ratté, J.C., 1960, Relation of mineralization to caldera subsidence in the Creede district, San Juan Mountains,Colorado: U.S. Geol. Survey Prof. Paper 400, p. B14-B17.

_____, and Ratté, J.C., 1965, Geology and structural control of ore deposition in the Creede district, San Juan Mountains, Colorado: U.S. Geol. Survey Prof. Paper 487, 90 p.

_____, Stacey, J.S., Delevaux, M.H., Doe, B.R., Lipman, P.W., and Fisher, F.S., 1979, Genesis of ore deposits in the San Juan volcanic field, southwestern Colorado - Lead isotope evidence: Econ. Geol., vol. 74, no. 1.

Sumi, Kiyoshi, 1968, Hydrothermal rock alteration of Masakawa geothermal area, Northeast Japan: Geol. Survey of Japan Report No. 225.

Taylor, H.P., Jr., 1973, O^{18}/O^{16} evidence for meteoric-hydrothermal alteration and ore deposition in the Tonopah, Comstock Lode, and Goldfield mining districts, Nevada: Econ. Geol., vol. 68, p. 747-764.

Taylor, R.B., 1975, Upper tertiary tectonism and deposits in south-central Colorado, in Curtis, B.F., ed., Cenozoic history of the southern Rocky Mountains: Geol. Soc. America Memoir 144.

Thompson, G.A., and White, D.E., 1964, Regional geology of the Steamboat Springs area, Washoe County, Nevada: U.S. Geol. Survey Prof. Paper 458-A, p. A1-A52.

Tolman, C.F., and Ambrose, J.W., 1934, The rich ores of Goldfield, Nevada: Econ. Geol., vol. 29, p. 255-279.

Turneaure, F.S., 1960, A comparative study of major ore deposits of central Bolivia: Econ. Geol., vol. 55, p. 217-254,574-606.

Vikre, P.G., 1979, Geology and mineralization of the Buckskin-National district, Humboldt County, Nevada: ASARCO private report, 43 p.

_____, 1980, Fluid inclusions in silver-antimony-arsenic minerals from precious metal vein deposits: Econ. Geol., vol. 75, no.2, p. 338-339.

Wallace, A.B., 1978, Geochemistry of polymetallic veins and associated wall rock alteration, Pyramid district, Nevada: SME of AIME Annual Meeting, Denver, preprint 78-L-3, 7 p.

Wandke, A., 1930, Ore deposition in open fissures formed by solution pressure: Am. Inst. Mining Metall. Eng. Tech. Pub. 342, 15 p.

Weissberg, B.G., 1969, Gold-silver ore-grade precipitates from New Zealand thermal waters, Econ. Geol., vol. 64, p. 95-108.

White, D.E., 1955, Thermal springs and epithermal ore deposits, in Bateman, A.M. , ed., Fiftieth Anniversary volume, 1905-1955: Econ. Geol., p. 99-154.

_____, 1974, Diverse origins of hydrothermal ore fluids: Econ. Geol., vol. 69, p. 954-973.

_____, Muffler, L.J.P., and Truesdell, A.H., 1971, Vapor-dominated hydrothermal systems compared with hot-water systems: Econ. Geol., vol. 66, p. 75-97.

_____, and Roberson, C.E., 1963, Sulphur Bank, California, a major hot-spring quicksilver deposit, in Petrologic Studies—A volume in honor of A.F. Buddington: New York Geol. Soc. America, p. 397-428.

_____, Thompson, G.A., and Sandberg, C.H., 1964, Rocks, structure and geologic history of Steamboat Springs thermal area, Washoe County, Nevada: U.S. Geol. Survey Prof. Paper 458-B, 63 p.

Whitebread, D.H., 1976, Alteration and geochemistry of tertiary volcanic rocks in parts of the Virginia City quadrangle, Nevada: U.S. Geol. Survey Prof. Paper 936, 43 p.

Wilson, H.D.B., 1944, Geochemical studies of the epithermal deposits at Goldfield, Nevada: Econ. Geol., vol. 39, p. 37-55.

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NO.127(1984)
JULY 2, 1984

George Cross News Letter

"Reliable Reporting"

WESTERN CANADIAN INVESTMENTS

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NO.127(1984)
JULY 2, 1984

BAR RESOURCES LIMITED (BSL-V)

INCUMBENT DIRECTORS CONFIRMED - DIRECTORS OPTIONS CONFIRMED - MERGER TO BE FURTHER STUDIED - The proxies at the Bar Resources Limited meeting on June 28,1984 were finally counted in the early evening. After a number of disputed ballots were disallowed it was determined that incumbent directors were reelected by a substantial margin: W. Clarke Campbell, Steven Kent, Evan A. Koblanski, D.W.J. Specht and Robert A. White. The meeting approved options to directors and employees covering 245,000 shares at $1.25 each exerciseable until 28Sep88.

Mr. Specht stated that the acquisition of Eros Resources Inc. by Bar has been suspended pending a further evaluation of the properties of Eros which will be submitted to a further shareholders meeting prior to completion (See GCNL 126(84), Page One, 29Jun84 for some detail)

PEGASUS GOLD LTD. (PGU-V,T) MONTORO GOLD INC. (MNT-V)

MINE PROGRESS REVIEWED - AMALGAMATION PROPOSAL MAILED - Ore reserves at the Zortman/Landusky heap leach mines of Pegasus Gold Ltd. located in the Little Rocky Mountains of Montana have recently been recalculated as 28,000,000 tons grading 0.021 oz.gold per ton and 0.25 oz.silver per ton using a 0.006 oz.gold/t cut-off grade. The reserves include only oxide ore. Research is underway on gold recovery from the sulphide ores on the properties. This work has indicated that total recovery is about 10% less from the sulphide ore than from the oxide ore. As a result it is suggested that a higher cut-off grade will have to be used in calculating sulphide ore reserves. Research work is continuing.

For 1985 operations, construction has started on two pads, each to have capacity of 6,000,000 tons. It is hoped that some ore will be dumped on these pads before the end of the 1984 season.

To 26Jun84, there have been a total of 3,345,000 tons of ore grading 0.023 oz.gold/t, 0.126 oz.silver/t with a stripping ratio of 0.45 waste to each ton of ore added to the heap leach. Gold recovery to date has been 28,000 oz., some 3,000 oz. ahead of this date last year, plus 60,057 oz.silver, the same as last year. Costs are $250 per oz. of recovered gold, about $3.00 per ounce less than in 1983.

The meeting was told that the 25May84 proposed amalgamation of Pegasus and Montoro is proceeding with the formal offer being mailed 29Jun84. The amalgamated company will be named New Pegasus and the share exchange, one for one for Pegasus and one share of New Pegasus for each 2 2/3 shares of Montoro.

Montoro holds the Beal property near Butte, Montana which is expected to have a final feasibility report by the end of 1984. Montoro also holds 50% of the Florida Canyon property, Pershing county, Nevada which has reserves of 20,000,000 tons grading 0.022 oz.gold/t, plus a new zone with higher grade.

SULPETRO LIMITED (SUL-T,Alberta)

6 MONTHS ENDED 30 APRIL	1984	1983
Gross Revenue	$111,300,000	$106,400,000
Oil/Gas Sales, Net	69,700,000	68,000,000
Cash Flow	10,600,000	11,900,000
Net Loss	$ 19,000,000	$ 13,500,000
Loss Per Share	$1.26	90¢

In presenting comparative results for their fiscal year's first half ended 30Apr84, management of Sulpetro Limited attribute the higher revenues mainly to increased North Sea net oil sales. Management expect the loss for the full fiscal year will not exceed that reported in 1983, even though reduced capitalization of interest and increased amortization of foreign exchange resulted in $4,500,000 of additional expense in the first half of fiscal 1984.

Cash flow in the first fiscal half declined due to a decrease in recoveries of current taxes. Additional funds in the amount of $29,300,000 were generated from property sales and were used mainly for Canadian exploration and development projects with potential for early cash flow. Capital expenditures were $29,000,000, up from $24,400,000 in the first half of the previous fiscal year. Long-term debt was reduced by $21,300,000 during the first half of fiscal 1984.

ASARCO

File Gold Bulk/Heap X

Exploration Department
Frederick T. Graybeal
Chief Geologist

January 12, 1984

Mr. David F. Skidmore
Assistant to General Manager
Southwestern Mining Department
P. O. Box 5747
Tucson, Arizona 85703

Wiluna Project, W. A.

Dear Mr. Skidmore:

The following comments cover certain aspects of the mineralization at Wiluna which might be of importance during your evaluation. The mineralization that we have drilled adjacent to both the East Lode and West Lode pits is underlain by stopes along the vein structures. We have a fair amount of information on the location and size of these stopes, but as the mine was a major underground producer (9,000,000 tons), some of these stopes may be large and a troublesome factor during open pit mining.

The majority of the mineralization is linear, following shear zones, and some of the ore zones may be 10-30' wide. They are often of high grade, but because of their narrow width, they may suffer significant dilution during open pit mining which must be recognized during your reserve calculations.

Some of the ore grade gold intersections are of relatively uniform grade. Others show significant grade variations with local zones in excess of 0.5 oz.Au, surrounded by lower grade envelopes. The vein structures which contain the higher grade central zones now appear to contribute a significant portion of the total grade of the deposit and I am inclined to worry that in a heap leach operation that the smaller volumes of higher grade rock might not be washed as consistently and thoroughly as the low grade rock and significantly reduced recoveries over those indicated by recent column leach work might result. Therefore, although a great deal of talk and metallurgical work has centered around the possibilities of a heap leach operation at Wiluna, we (I) have in no way ruled out the possibility of a CIP or agitation leach flow sheet.

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JAN 17 1984
EXPLORATION DEPARTMENT

Some of the mineralization, and we don't know how much, contains relatively abundant sulphate and chloride minerals. These minerals appear to be water soluble, but I understand that soluble salts in solution might interfere with carbon recovery of gold and, therefore, the distribution of sulphate and chloride in the ore may be of significance. I don't think we would want to reject this material to waste, because the sulphate or chloride minerals contain a certain amount of water-soluble gold. Dr. El Tawil at Central Research is currently studying the mineralogy of these soluble salts. You may want to discuss with Mr. Crowell the extent which we should run water soluble salt assays of composite drill samples.

Mr. Williamson can provide information on the distribution of ground water which would be available for a mining operation in the Wiluna region. My understanding is that both fresh and saline aquifers are present. I have also been told that Western Alluvials permit for their tailings operation is quite specific for that operation and could not be transferred to another operation without revising the terms with the W. A. government.

I haven't discussed any aspects of the geology at Wiluna which you can obtain from Mr. Williamson and El-Raghy. The only recent finding which differs in a minor way from information you obtained in Denver and which is currently in the files that I will be sending you, is that the oxide-sulfide boundary is locally shallower (100-125' rather than 140-150') than we had originally thought. It is likely that it may also be locally deeper than we originally thought.

Background information on the project is contained in the information package which accompanies this letter. Please return all of this information to the New York office with the exception of the environmental studies which relate to the Lake Way uranium deposit. I think that it would be useful to circulate this information, which is almost entirely geological in nature, to both Mr. Kurtz and Mr. Anzalone, so that they could provide additional comments before you leave for Perth. In addition to this information, we have several feet of file data which is also available in Perth, and which might better be reviewed there. Mr. Crowell has a column leach test report by Mr. Apland, which you will also find useful.

Our plans for 1984 involve drilling the entire East and West lode structures along east-west sections 50 meters apart with drill hole spacing along each section of roughly 30 meters. Any ore grade intersections on a particular section will be followed up by drilling interspaced drill holes at a 15 meter spacing and by drilling the intervening sections (25 meters spacing). This work will be starting sometime in March. In addition, we have some drill core which was obtained for metallurgical purposes which will be sent to Mr. Crowell early in February for further column leach testing. Included in our 1984 budget for Wiluna is a $250,000 expenditure for a pilot heap leach test which might begin around September. Plans for this test might be deferred if alternate metallurgical flow sheets look more attractive, but in any case, we must spend a minimum of $2,040,000 prior to August 31, 1985 to vest with a 51% interest in the property. Failure to reach that expenditure level would result in loss of the property. Following your review of the accompanying data, you might discuss with Mr. Williamson the type of information required to begin reserve studies and the form in which you would like that information presented.

Very truly yours,

F. T. Graybeal

F. T. Graybeal

encl.

cc: R. J. Kupsch
W. L. Kurtz
S. A. Anzalone
D. R. Crowell
R. A. Raihl
H. C. Williamson

ASARCO

Mineral Beneficiation Department

xc: WLKurtz, 9/28/83 ab

WLK file Gold Bolt/Heap

August 10, 1983

Mr. R. J. Kupsch, Vice President
Mining Department - New York Office

Dear Mr. Kupsch:

Enclosed are individual reports on the Nevada precious metal operations which we recently toured. These are written by Mr. Raihl and myself. We have on file here a large number of photographs of these operations taken by Mr. Skidmore and myself which you might want to review on your next trip to Tucson or we could send the photo album to New York if you prefer.

I feel that the entire group obtained a considerable amount of information on the tour some of which has already been useful during the planning of the Rochester heap leach test work.

D. E. Crowell

DEC/ab
Attachments

cc: TEScartaccini
DEHolt
MEl Tawil
File/DF G6b

R. J. C.
SEP 21 1983

A. R. R.
SEP 20 1983
TUCSON

SEP 9 1983
D.C.D.

Nevada Heap Leach Operations

General comments will be made concerning the heap leach operations that were visited in Nevada.

One item of note is that the extremely rough cost estimate made for the Florida Canyon Project early in 1981 was not too much different than the capital costs of the present operating plants. The Florida Canyon estimate not including mining equipment or premine costs was as follows:

2000 TPD Plant	$10,300,000	$5000/ton day
8000 TPD Plant	$25,900,000	$3250/ton day

The total cost at Candelaria was 40 million for a 50 - 60,000 TPW plant (7,000 - 8,600 TPD). Probably 10 million of this cost was for mine equipment and premine development. Alligator Ridge cost about 25 million not including the mine equipment for a 2500 TPD plant.

After the recent visit of the Nevada operations, many of the design aspects used for the Florida Canyon study would be changed. However with these changes, especially with the leach pad sealing system, the overall costs would not be greatly modified.

Dump Leach Solution Distribution

The operations have fairly well standardized on "Wobbler" sprayheads obtained from Conley Company, Salt Lake City. Sprayheads are manufactured by Senninger Irrigation, Inc., Orlando, Florida. These heads are on approximately 40 ft. centers, and flow to each head is controlled at 8 gpm with a Dole flow control. Solution flow rate is therefore 0.006 gpm per square foot.

"Yellow Mite" plastic pipe has become standard for distributing solutions. It is assumed that this is a HDPE pipe.

Ditch and Pond Liners

HDPE sheeting with matting is becoming standard. This type of lining is much better than Hypalon.

Leach Pad Seal

The leach pad seal for most of the Nevada operations consists of 18 inches of compacted clay and silt obtained from local dry lake beds. This seal though satisfactory in Nevada may not be satisfactory in other states.

Some but not all operations place a 12 inch layer of minus 2 inch plus 3/4 inch gravel over the clay layer to help with drainage.

Also some operations place a perforated piping system on about 15 ft. centers between the clay and the gravel to help with solution drainage. The gravel and the perforated pipe help reduce the hydrostatic head on the clay seal.

Height of Leach Dumps

More and more operations are going to multiple lifts on the leach dumps with projected ultimate heights of 60 ft. Good possibility that higher dumps will be planned in the future.

Building of Individual Lifts on Dumps

There is no standard method of building ore lifts as there appeared to be several years ago. Height of lifts has been lessened from 20 to 25 ft. to 8 to 15 ft. in general. Apparently methods of building lifts presently is dependent on how finely crushed, fines content, mode of operation, etc. Lift building is undergoing a considerable amount of experimentation.

Leach Pad Slopes

The leach pads we saw are being built on alluvial fans and utilize a slope in two directions of between 3-1/2 and 5 percent. The area is cleared of all organic material and then the slopes are evened out with high spots cut off and low spots filled in. I would assume that the fill is compacted. After this ground preparation the clay seal is placed. It must be remembered that once the clay seal is placed, it must be kept continually moist.

Agglomeration

All the heap leach operations visited that crushed in two or more stages agglomerated the ore with cyanide solution, and the pH regulator was normally NaOH. Only the Borealis operation used cement in their agglomeration procedure and only Alligator Ridge used lime. The reason for their use of lime is probably due to the large amounts required, between 5 and 15 lbs. per ton.

Agglomeration methods varied from using a drum agglomeration at Alligator Ridge to using several belt conveyor transfers to just dumping the moistened ore onto a stockpile.

Winter Operation

Several operations installed submersed burner boilers to heat the solutions going to the leach dumps so that winter operation was possible. Comments concerning this are as follows:

1. Submersed burner boilers, though thermally efficient, destroy all the cyanide and acidify the solution. For this application standard indirect fired boilers are best.

2. It may only work during mild winters because the leach solutions freeze in the dumps and not in the pipe lines.

3. One Nevada operator stated that winter operation was not economically justifiable.

Laboratory and Pilot Plant Testing

There is not much concensus of opinion concerning column leach testing of ore. Most agreed; however, that actual optimum leach time in a dump was considerably longer than the optimum time as determined in leach columns. Tests columns varied from the size of a 5 gallon can to columns 4 to 5 ft. in diameter by 12 to 15 ft. high. In general columns were 8 to 10 inches in diameter by 4 to 6 ft. high. Some of the literature talks of modeling the leach column test results to determine what an actual dump will do; however, no detailed information is available. The procedure at some plants for column leach tests is to run only fresh solution over the ore instead of circulating solution.

Most of the heap leach operations have conducted a small heap pilot plant. Most of these test heaps have been 10,000 tons but some have been as little as 5,000 tons. Consensus of opinion is that if a pilot program is conducted using a test heap, the heap should contain a minimum of 10,000 tons. If the test heap is to be 15 ft. high, the minimum pad dimensions should be 150 ft. by 150 ft.

STERLING MINE - BEATTY, NEVADA

Heap Leach - 6 inch ore
Open Pit, future underground
5000 tons per month
Feed Grade - 0.25 opt gold
Recovery - unknown
Carbon Absorption
Capital Costs - unknown

BOREALIS PROPERTY

Heap Leach - 80% minus 1 inch
Open Pit
1500 TPD, 450,000 TPY
Feed Grade - 0.15 opt Au, 0.4 opt Ag
Recovery - 75-80% Au, 15-30% Ag
Merrill Crowe
Capital Costs - unknown

CANDELARIA

Heap Leach - minus 1-1/2 inch
Open Pit
10,000 TPD, 50-60,000 TPW
Feed Grade - ± 3.0 opt Ag
Recovery - less than 50%
Merrill Crowe
Capital Costs - 40 million total incl. mining equipment

PINSON MINING COMPANY

Heap Leach - Mine Run
Open Pit
2500 TPD
Feed Grade - 0.01-0.04 opt Au, estimated avg. 0.025 opt
Recovery - unknown
Carbon Absorption
Capital Costs - unknown

Note: Heap leach is in addition to existing fine grind agitation leach that has an ore cut off grade of 0.04 opt Au.

ALLIGATOR RIDGE

Heap Leach - minus 3/4 inch
Open Pit
2500 TPD, 750,000 TPY
Feed Grade - 0.1 opt, NaCN soluable gold
Recovery - 65-70% to date
Carbon Absorption
Capital Costs - 24 million, not incl. mine equipment

Silver Peak Mine - Sunshine Mining Company

The Silver Peak or 16 to 1 operation of the Sunshine Mining Company was visited in the p.m. of June 8th, 1983. Mr. Allen Young gave us a tour of the milling operation.

The operation is fine grind silver leach plant. Ore reserves contain 6 to 7 opt silver and 0.04 opt gold. Leach recoveries (metal in feed and doré bars) is about 88 percent for the silver and about 95 percent for the gold. Reserves are 1.2 million tons.

The plant was designed by Industrial Engineers, Salt Lake City. Surface plant including crusher, mill, office, shop, warehouse, changehouse, tailing dam and water supply cost about $11.0 million. Construction period was about 10 months and went into operation in 1982. Total plant cost including premine development and some employee housing facilities was about $24.0 million.

Desired mill capacity was 450 TPD so was designed for 550 TPD. With minor modifications the plant is presently operating at about 625 TPD and may be capable of close to 750 TPD. They are planning to treat up to 100 TPD of custom ore. Presently are negotiating with Eagle Pitcher on some gold ore from California. Mill operates 24 hours per day, 7 days per week.

Mine - Underground basically single vein up to 22 ft. wide. The underground ore is hoisted and passes over a grizzly about 22 x 22 inches. Oversize is broken with a rock breaker. The ore is then hauled with 25 - 30 ton trucks approximately 3 miles to the crushing plant. The ore is dumped into an ± 50 ton ore hopper and is fed with a Nico hydrostroke feeder to a 30 x 42 inch Kue Ken jaw crusher. Ore is crushed in closed circuit to about minus 1/2 inch with a 4-1/4 ft. Nordberg cone crusher. The screen is an AC horizontal which is located over the fine ore bin.

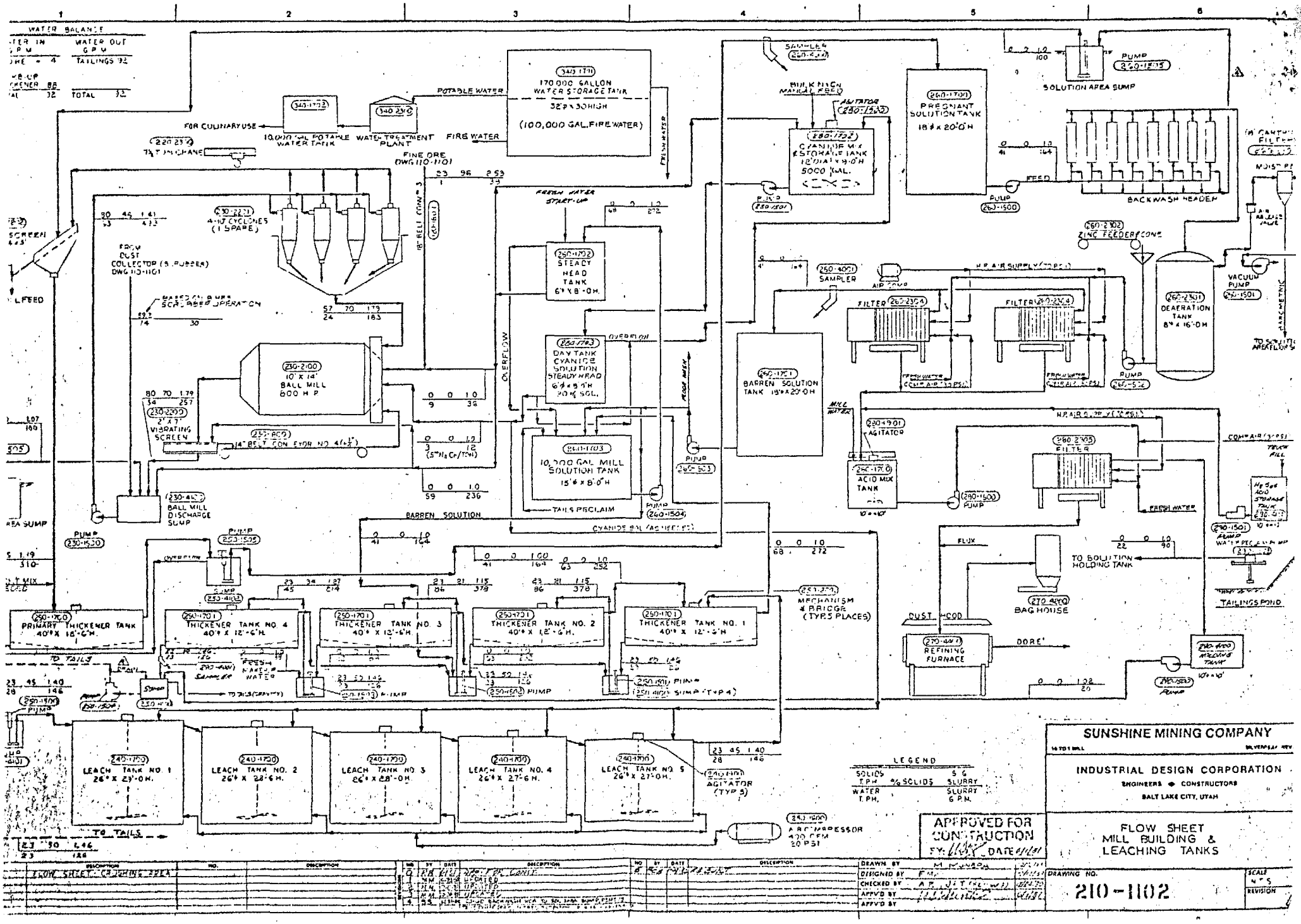
The ore is ground to 80 percent minus 200 mesh in a single stage 10 x 14 ft., 800 hp ball mill. The ball mill has a trommel on the discharge and the oversize is fed back into the feed end using conveyors. The cyclone overflow is screened to remove any tramp. Preliminary work index was determined to be about 18.0 but ore is considerably softer than initially thought. This is the reason for the above design capacity. Ball consumption is 3.4 lb/ton.

The cyclone overflow is thickened and then leached in 5 agitators in series. Approximately 80 scfm of air is blown into each 26 x 28 ft. agitator. The pulp is washed with a four stage CCD circuit. Approximately 0.36 lb/ton of flocculant is used. Cyanide concentration in the 1st leach tank is 16 lb/ton and consumption is about 7.5 lb/ton. The pH is controlled at about 11.2 with approximately 2.4 lb/ton of lime.

Silver and gold is recovered using the standard Merrill Crowe process. Cartridge filters were initially used to polish the pregnant solution but they were not satisfactory. This part of the circuit had to be changed to U.S. leaf filters. Perrin plate and frame filters recover the precipitate. A precipitate acid leach circuit was installed but its not being used. Precipitate is smelted.

The total employment at the operation is 85 men, 70 hourly and 15 staff. There are a total of 20 hourly men in the mill for a 7 day per week operation.

The plant was designed by Industrial Design Corporation in Salt Lake City which did a relatively good job.


WATER BALANCE
WATER OUT GPM
TAILINGS 92
TOTAL 32
170,000 GALLON WATER STORAGE TANK
32'Ø x 30' HIGH
(100,000 GAL. FIRE WATER)
POTABLE WATER
FOR CULINARY USE
10,000 GAL. POTABLE WATER TANK
WATER TREATMENT PLANT
FIRE WATER
FINE ORE DWG 110-1101
4-10' CYCLONES (1 SPARE)
FROM DUST COLLECTOR (8 RUBBER) DWG 110-1101
BASED ON 8 HRS SCALPER OPERATION
10' x 14' BALL MILL 800 H.P.
3' x 7' VIBRATING SCREEN
14" BELT CONVEYOR NO 4
BALL MILL DISCHARGE SUMP
PUMP
STEADY HEAD TANK 6'Ø x 8'-0" H.
DAY TANK CYANIDE SOLUTION STEADY HEAD 6'Ø x 8'H 20% SOL.
10,000 GAL. MILL SOLUTION TANK 15'Ø x 8'-0" H
TAILS PIC LAIM
BARREN SOLUTION
CYANIDE SOL (AS NEEDED)
OVERFLOW
FRESH WATER START-UP
CYANIDE MIX & STORAGE TANK 12' DIA x 8'-0" H 5000 GAL.
MILK OF LIME MANUAL FEED
AGITATOR
PREGNANT SOLUTION TANK 18'Ø x 20'-0" H
SOLUTION AREA SUMP
BACKWASH HEADER
FEED
ZINC FEEDER CONE
DEAERATION TANK 8'Ø x 16'-0" H
VACUUM PUMP
SAMPLER
FILTER
H.P. AIR SUPPLY (70 PSI)
BARREN SOLUTION TANK 15'Ø x 20'-0" H
ACID MIX TANK
FILTER
FRESH WATER
FLUX
TO SOLUTION HOLDING TANK
BAG HOUSE
DUST HOOD
REFINING FURNACE
DORE'
TAILINGS POND
MECHANISM & BRIDGE (TYP 5 PLACES)
PRIMARY THICKENER TANK 40'Ø x 18'-6" H.
THICKENER TANK NO. 4 40'Ø x 12'-6" H.
THICKENER TANK NO. 3 40'Ø x 12'-6" H.
THICKENER TANK NO. 2 40'Ø x 12'-6" H.
THICKENER TANK NO. 1 40'Ø x 12'-6" H.
FRESH MAKEUP WATER
TO TAILS
LEACH TANK NO. 1 26'Ø x 23'-0" H.
LEACH TANK NO. 2 26'Ø x 23'-6" H.
LEACH TANK NO. 3 26'Ø x 23'-0" H.
LEACH TANK NO. 4 26'Ø x 27'-6" H.
LEACH TANK NO. 5 26'Ø x 27'-0" H.
AGITATOR (TYP 5)
AIR COMPRESSOR 400 CFM 20 PSI
LEGEND
SOLIDS T.P.H.
% SOLIDS
S.G. SLURRY
WATER T.P.H.
SLURRY G.P.M.
SUNSHINE MINING COMPANY
INDUSTRIAL DESIGN CORPORATION
ENGINEERS ◆ CONSTRUCTORS
SALT LAKE CITY, UTAH
APPROVED FOR CONSTRUCTION
FLOW SHEET MILL BUILDING & LEACHING TANKS
DRAWING NO. 210-1102
SCALE NOTES
REVISION
DRAWN BY
DESIGNED BY
CHECKED BY
APPVD BY
APPVD BY
FLOW SHEET CRUSHING AREA
DESCRIPTION
NO.
DATE
BY

Sterling Mine - Saga Exploration

The Sterling Mine was visited on the morning of June 8. The trip was set up by Greg Austin of Saga Exploration. The metallurgical end of the operation was designed by D. W. Kappes of Miller-Kappes, a Sparks, Nevada consulting group. A very detailed paper on the Sterling Property by Messrs. Austin, Kappes, and Parrish is attached so I will only give some very general comments here.

1. The capacity of the plant was given as 5,000 tons per month but this varies greatly and can be as high as 10,000 tpm. Since the ore heaps are left in place there is no particular timed cycle or schedule that must be met. Heaps continue to be leached until they are judged uneconomical.

2. The ore grade was given as .25 opt Au with a 30:1 Au:Ag ratio. Ore from the mine is passed over a stationary 4-inch grizzly and only grizzly undersize goes to the leach pads. The plus 4-inch material is stockpiled and they plan to crush this material and leach it someday.

3. Recovery was given as 80% or better in about 90 days of leaching. This is a very high figure for minus 4-inch ore but then the .25 opt heads are far higher than other heap leach operations.

4. The pads are constructed of 30 mil pvc laid over a thin gravel sub-base with the pvc then covered with 6 inches of sand and 12 inches of minus 3/4 inch gravel. 4-inch diameter perforated pvc pipe is laid in the gravel on 10 foot centers. The side and bottom berms of the pads are very simply constructed by rolling the pvc liner up over a row of empty cyanide drums laid end to end. Sunlight will eventually destroy the exposed pvc but probably not until the leaching process is well completed. Solution ponds are constructed of hypalon for sunlight resistance.

5. Low grade pregnant solution from older heaps is recycled to newly placed heaps in order to build higher grade solutions for the gold recovery circuit. Sterling is the only operation that does this to my knowledge. It is normally not considered good practice to return a pregnant solution to a heap leach but this may be feasible at Sterling with their very amenable ore. Also Sterling has a very small carbon absorption plant and solution recycle would be one way of increasing their gold recovery capacity.

6. Mr. Austin commented that even with the "ideal" Sterling ore, the most important single factor in the operation was careful control during heap construction to avoid segregation. Heaps are constructed in three 5-foot lifts with trucks dumping on top of the lift and bulldozers pushing the ore over the leading edge of the lift. After a lift is completed, it is ripped before the next lift is started.

RECOVERY OF GOLD FROM THE STERLING MINE

NYE COUNTY, NEVADA

I. S. Parrish

Michener & Booth, Inc.
Golden, Colorado

G. G. Austin

SAGA Exploration Company
Winnemucca, Nevada

D. W. Kappes

Miller-Kappes Company
Sparks, Nevada

R. J. C.
MAR - 4 1982

Introduction

The Sterling gold mine is typical of the many small heap leach operations active in Nevada. The ore is composed of micron-sized gold particles unevenly disseminated in iron-stained sedimentary rocks overlying a major thrust plate. The gold is recovered from a heap leached by a cyanide solution which is then passed through activated carbon. The carbon is later stripped by a caustic medium. The Sterling mine merits study as its history illustrates that small tonnage, moderate grade underground mines can be profitably worked.

Location

As illustrated on Figure 1, the Sterling mine is located in western Nye County some 10 miles (16 km) southeast of Beatty, Nevada. Physiographically it is on the eastern flank of Bare Mountain adjacent to Crater Flat and the Nevada Test Site. Topography of the mountain is rugged with relief of as much as 2,000 feet (600 m). The adjacent valley is gravel filled, broad and flat. The ore body is within the mountain, allowing access via the deeply eroded canyons. The main recovery plant is located on the relatively flat main valley floor.

The climate is arid and summer temperatures exceed 110°F (45°C).

History

Beatty has been an active mining area since the turn of the century. Most production has come from the Bullfrog district west of Beatty. Prospectors of the time combed the Bare Mountain area and the so-called Panama shaft on the Sterling property dates from the 1910-1920 period. Production here was negligible. Some ore was found in siliceous fillings of limestone caverns and in narrow quartz veins in dolomite. Gold has also been reported from other mines on Bare Mountain, such as the Gold Ace and the Telluride.

Of more economic importance was the fluorspar production from the Diamond Queen, or Goldspar pit, adjacent to the Sterling mine. Most of the production here occurred between 1958 and 1967.

In early 1978, John Livermore and Peter Galli of the Cordex Syndicate (Cordilleran Exploration) staked the Stirling claims to cover the Panama shaft and all the land extending from it north to the Goldspar group. After initial mapping and geochemistry, they drilled holes numbered 1 through 37, some of which encountered gold mineralization. The claims were optioned by one of the present authors, Greg Austin, for SAGA Explorations. Austin drilled 14 more holes establishing an apparent ore trend and drove a short adit to expose the ore. In 1979, Dan Kappes began to study the possibility of heap leaching the Stirling ore. Derry, Michener & Booth, Inc. and E & B Explorations, Inc. of Vancouver joined with SAGA in January of 1980 to provide financing and technical assistance via the Sterling Mines Joint Venture for development of the possible ore zone.

The adit was rehabilitated and underground advance began in April of 1980. Ore was stacked on a pilot pad and leaching began on July 1, 1980. The first gold was recovered in September of 1980. During the Fall of 1981, payback of all capital expenses was achieved. Drilling has extended the ore zone so that several years of additional production and profits are anticipated.

The success of the Sterling Joint Venture is, in part, due to the division of responsibilities amongst qualified professionals. The mining and every day operation of the Sterling is the responsibility of SAGA Explorations of Winnemucca, Nevada. They maintain a work crew of about 30 people under the direction of Jim Bosley, Mine Manager. The geology and exploration is directed by Derry, Michener & Booth, Inc. of Golden, Colorado. DMB has one full-time geologist at the site and provides additional geologists on an as-needed basis. DMB also provides long range engineering input. Miller-Kappes, Inc. of Reno, Nevada designed and carries responsibility for the recovery system. A representative spends about one week a month on the property and is on call at all times. E & B Explorations, Inc. of Vancouver, in addition to providing the bulk of the financing, lends overall direction to the Joint Venture.

Geology

General

Bare Mountain is underlain by sedimentary rocks that range in age from Late Precambrian to Upper Mississippian. Paleozoic and later Cretaceous deformations have folded and faulted the sequence so that stratigraphic correlation is difficult. In addition, Tertiary volcanic activity and associated faulting and subsidence complicate the geology.

The Bare Mountain area lies with the Walker "Lane" or lineament which is 30 miles (48 km) wide and strikes northwest across western Nevada (Figure 2).

The geology of the area is best described by Cornwall and Kleinhampl (1961) on the 15 minute Bare Mountain Quadrangle map and in the Nevada Bureau of Mines & Geology Bulletin No. 77, on Geology and Mineral Deposits of Southern Nye County (Cornwall, 1972).

Local

The rocks in the area of the Sterling mine are a series of light colored (gray, tan, orange) dolomites, siltstones and quartzites. The ore occurs principally in red siltstones of the Wood Canyon formation of lower Cambrian age overlying gray dolomite of the Bonanza King formation. Earlier maps indicate the host unit to be the Johnnie formation, however, the most recent studies (Cornwall, 1972) show the unit as Wood Canyon formation.

It should be noted that very small, but important gold concentrations are found in the Bonanza King dolomite and also that narrow dolomitic beds are present in the Wood Canyon

formation. Also, although ore is predominantly found in red siltstones, gold has been mined from tan, gray, green and even white siltstones. The ore lies above the contact at a thickness of up to 35 feet (10.5 m). Locally, gold mineralization extends as much as 15 feet (5 m) below the contact into the dolomite of the footwall. In many instances a massive quartzite bed is present a meter or so above the ore.

The Wood Canyon/Bonanza King contact is thought to be a thrust contact because of the stratigraphic relation of the younger Bonanza King unit beneath the older Wood Canyon. The thrust relation is not at all certain as the stratigraphic correlation is tenuous and although in some places the contact itself appears sheared and broken, in others it is normal and undisturbed.

In the vicinity of the mine the rocks are folded into a syncline. This fold can be traced in the mine workings and from drill hole data. It has a northeast trend and plunges gently to the south.

Ore Geology

Although several ore pockets have been located, only two have been worked. One, the main zone, has been the focus of activity, the second, near-surface enrichment, has only received cursory attention up to this time.

The main zone is a shoestring body (Figure 3) that ranges from 50 feet (15.5 m) to 150 feet (46 m) in width, up to 35 feet (10.5 m) in height and is known to extend at least 2,000 feet (600 m) in length. In longitudinal section, the zone plunges gently (10°) to the southwest, but rolls and pitches so that the development decline is flat in places and locally rises. In cross section the ore body is roughly canoe or keel shaped being thickest near or at its middle and thinning to both sides (Figure 4). The top of the body is generally flat. Note, however, that the cross section in any one area may be tabular, tilted or have abrupt faulted(?) edges.

The grade of the ore ranges up to several ounces over five feet (1.5 m), but averages 0.25 oz. Au/ton. Silver content is negligible with a gold:silver ratio of about 30:1. There is little or no copper, lead or zinc. A very minor amount of stibnite and galena have been found. The alteration minerals of volumetric consequence are the iron secondaries (hematite, limonite and goethite), the clays and sericite. Pyrite is also present in small amounts and it is assumed that pyrite is the source of the iron.

The gold is present as flakes and particles to 4 microns in size, found with hematite and rimming calcite grains.

Although little work has been done on gold found outside of the main zone, the occurrences suggest similar controls and habits.

Operation

SABA Explorations operates the Sterling mine. Jim Bosley is Mine Manager. He has a crew of about 20 people including 3 miners, 1 surveyor, 1 foreman, 2 assayers, 2 truck drivers, 1 clerk, 5 laborers, 2 samplers, 2 shift bosses and 3 mechanics.

The surface plant, excluding the recovery plant which will be discussed in detail later on, consists of an office trailer, an assay trailer, warehouse and machine shop trailers, a Deutz 50 KVA generator and a 600 cfm Compressor. Rolling stock, excluding pickups, includes a 22 ton dump truck, a 7,200 gallon (27,250 liter) water tanker, two 4 yd (3 m^3) front-end loaders (a Cat and a Terex), a Case 1 1/8 crawler loader and a small 8 foot (2.4 m) Basic grader.

Water is a major concern. The Sterling at present buys its water from the town of Beatty. An average of 9,000 gal/day (34,065 liters/day) in the winter and 15,000 gal/day (56,775 liters/day) in the summer is trucked to the mine site. About 20% is stored in a tank above the workings for use underground and the remainder is used at the recovery plant.

The mine is worked on a 2-shift basis. Underground equipment includes a Secoma hydraulic drill ATHE 12, 2 Elmac 5250 loaders, 3 Elmac trucks (two 010s and one D5), pneumatic drills, etc. The Secoma is an entirely hydraulic unit. Elmac units were chosen because of their parts standardization (interchangeable engines, transmissions and drive-trains). Split-set rock bolts installed on 4 foot (1.2 m) centers have proven to be the most suitable for ground support especially when augmented by wire screening. The rock bolts are installed using jack-legs. Ventilation is provided through two vent raises. Two fans, one 6 foot (1.8 m) diameter 50 hp and one 36 inch (.9 m) diameter 50 hp, have been installed.

The general mining method has been modified room and pillar with an occasional shrinkage stope. The mine was developed via an 8 foot (2.4 m) x 10 foot (3 m) decline. This drive was positioned so as to lie in the lower central part of the ore body. As drill data was sparse the drive has not always been ideally located, nonetheless, it is essentially a central footwall drift. From this drift crosscuts were driven to the edges of the ore body at approximately 75 foot (23 m) intervals. The crosscuts leave the main drift at nearly right angles and were driven so as to keep the footwall ore contact in the lowest part of the face.

Direction and control of the headings was supplied by visual and sample data, by previous drill hole data and in some cases, by longhole information. Samples were taken of every face. Vertical rib samples were also taken of both walls at 5 foot (1.5 m) intervals.

The samples were analyzed on site at a company owned and operated atomic absorption unit.

The ore zone plunges steeply south from a point midway along its length. At this point the central drift was halted and east and west fringe drifts were driven at the edges of the ore body. These drifts connected the crosscuts and divided the body into "rooms". Stoping of these "rooms" by slashing has begun. The first stope recovered a higher grade section between crosscuts 3 and 4 west

(Figure 5). A 25 hp Joy slusher was required to bring the ore down to the central haulage drift.

A second stope was started between crosscuts 8 and 11 east in low grade ore (Figure 6). Slashing here has shown that the wall rocks can be supported by rock bolts and screening and that 60% of the ore can be retrieved on a first pass with rubber tired LHD equipment. Further stoping is underway with present plans calling for making the western fringe drift the main haulage. Mining will proceed from south to north and from east to west. Future work will include driving below the ore pillars and eventual recovery of the pillars through finger raises.

The broken and sheared siltstone ore drills and breaks easily. The Secoma drill can complete 1.5 rounds per shift, with about 28 holes in each face. Break is about 9 feet (2.7 m) per round. The broken ore is transferred into an Elmac truck for haulage to surface. During the development phase, rate of production was 100 tons/day. This will be increased to 250 t.p.d. during the stoping phase.

On surface the ore is crushed to 3" (7.6 cm) size in a 24 x 36" (61 cm x 91.4 cm) jaw crusher prior to being trucked to the pads for leaching.

Heap Leach Design

Initial test work on fine-crushed ore indicated that there would be no percolation or chemical problems. The ore was competent enough that the heaps could be driven on by trucks and endloaders while being constructed, with only minor effects on solution flow. Very soft unsilicified ore yields 90 percent recovery in two weeks of laboratory leach testing, while the bulk of the ore is moderately silicified and yields 70-75 percent of its fire-assayable gold in six weeks of field leaching, when crushed to two inches (5 cm).

To confirm the lab test data, and to provide early reassurance to the investors, a mini-production heap of the first 3,000 tons of ore was stacked ten feet (3 m) high in two five foot (1.5 m) lifts, and leached using a temporary carbon column recovery circuit. Recoveries from this heap verified lab tests and heap size has gradually been increased to the present 15 foot (4.6 m) to 17 foot (1.5 m) height.

The current production heaps are located on a large alluvial fan about three-fourths (1.2 km) of a mile below the mine. To prepare a leach area, vegetation is removed and contours are smoothed by cutting and minor filling using a bulldozer. A thin layer of gravel is spread to cover rough areas of the prepared surface, and then a 30 mil thick PVC pad is installed by the mine crew.

This pad is covered with six inches (15 cm) of gravel screened to 3/4 inch (2 cm), which is made in an on-site screening plant from alluvial fan material. To insure good drainage, 4 inch (10 cm) diameter perforated polyethylene pipe is installed parallel to drainage, on 8 foot (2.4 m) centers within the gravel cover. This pipe is manufactured in 200 foot (61 m) long rolls, making installation very simple.

Heap Construction

The ore heaps are built on the pads in modules of about 12,000 tons each, and are permanently left on the pads. New heaps abut directly against older heaps so as to cover the old heap sideslopes. Eventually, the leaching area will appear to be one large continuous heap.

The ore is stacked on the 15 foot (4.6 m) high heaps in three five foot (1.5 m) high layers, to minimize segregation. A ramp of waste rock is built behind the heap up to the height of the layer to be constructed. Trucks drive up onto the top of the layer and dump their loads and a bulldozer pushes the material over the leading edge of the heap. When a layer is completed it is thoroughly ripped, and thereafter no truck traffic is allowed upon it. Once the third layer is completed, leaching is begun.

Heap Leach Design

Solutions are sprayed onto the heaps ten to sixteen hours per day (during daysnift and evenings) at a rate of 70 gpm (265 liters/m), or approximately 0.003 gallons per minute per square foot (.0209 ml/s per square meter). Because of the high assay grade of the ore and the hot, sunny climate, care is taken to insure good sideslope irrigation. Solution is applied using Bagdad wigglers with 3/16 inch (5 mm) I.D. tygon tubing, spaced on 9 foot (2.7 m) centers. Senninger wobbler No. 8 sprinklers, spaced on 27 foot (8.2 m) centers, are now being tried successfully on the top surfaces, though Bagdad wigglers will be maintained on the sideslopes. Addition of 500 ml per day of Barochem S-35 to the barren pond has almost eliminated scaling of the sprinkler heads.

The leach solution contains 0.5 to 1.0 grams sodium cyanide per liter. Sodium hydroxide is available for pH control, but the rock appears naturally alkaline and it is seldom added.

Water for the operation is hauled 23 miles (37 km) in a 7,200 gallon (27,250 liter) tank truck from the town of Beatty. Evaporation rate on hot summer days is 30 percent of solution sprayed, or about 10,000 gallons (37,850 liters). An additional 5,000 gallons (18,925 liters) per day is required to saturate new ore. The water is high in fluorides and chlorides. These are not affecting the leaching or carbon adsorption behavior, but they are the apparent cause of very high corrosion rates on anodes within the stripping plant.

Pregnant solutions exiting a new heap run 60 mg per liter (ppm) gold or higher, and remain above 3 ppm during most of the time it takes to reach 70 percent recovery. Each of the 12,000 ton heap modules drains to a separate pond and countercurrent leaching of older heaps is planned to begin in late 1981. This will permit very high recovery while maintaining low flowrates through the recovery plant.

Gold Recovery

The gold recovery plant encompasses several unique features, including the following:

- upflow, non-fluidized columns
- permanent residence of the carbon in the columns throughout adsorption, desorption, and acid washing cycles
- plastic (CPVC) pipe construction in the gold stripping circuit
- priming of all centrifugal pumps with a Moyno positive displacement pump

The complete plant is housed in a 40 foot (12.1 m) semi-trailer. Included within the trailer are four 36 inch (.9 m) diameter insulated carbon columns, a power supply and electrolytic cell, two pregnant solution pumps (one for normal operation, and a second for occasional fluidization of the carbon), a totally enclosed acid wash system, and a sink with countertop for handling the electrolytic cell cathodes. Immediately adjacent to the trailer is an 800 gallon (3,028 liter) insulated tank for strip solution storage, heated with 21 kilowatts of electric immersion heaters. The cost of the system was $50,000.00.

Column Operation

Each of the four columns is 36 inches (.9 m) in diameter, 7 1/2 feet (2.3 m) high, and contains 1,000 pounds (455 kg) of 6 x 12 mesh Calgon type PCB carbon. The four columns are arranged in two parallel sets of two columns each. Sixty to eighty gpm (273-363 liters/m) flows upward through the first column in the set at the rate of 8-12 gpm per square foot (335-505 liters/min per square m). A special fluidization pump is used to raise the flowrate to 17 gpm per square foot (867 liters/m per square meter) once every few days. Inflow solutions commonly contain in excess of 20 ppm gold and exit the second column to the barren pond with levels of 0.2 - 0.5 ppm gold.

Gold Stripping

Once gold levels in barren solution climb above 0.5 ppm a column set is taken off-line by switching valves and the alternate set is placed on-line. Gold is then stripped from the carbon in the off-line set using hot alkaline alcohol solution (83°C, 20 percent ethanol, 1 percent NaOH) at a flowrate of 8 gallons per minute (30 liters/m). Addition of sodium carbonate to the strip solution appears to greatly speed up the process and to significantly reduce corrosion. It is also important to calculate alcohol additions on the assumption that the carbon will absorb 12 percent of its weight in alcohol. The strip cycle is about 12 hours, and is preceded by a 2-hour carbon preheating period in which solution is pumped through the carbon then back into the heating tank. Strip solutions sometimes attain peak gold contents exceeding 5,000 ppm.

The electrolytic cell is a rectangular 30 gallon (114 liter) polypropylene tank containing eight stainless steel anodes on 3.5 inch (9 cm) centers and seven steel wool cathodes measuring 18 x 18 inches (46 x 46 cm). Solution flows up and then down between alternate anodes, and the cathodes are made to provide 1/4 inch (0.6 cm) of free solution flowspace between the cathode and the anode.

The cell is run at 400 - 600 amps, and up to 9 volts (with no anode corrosion, voltage is usually about 4). Current efficiency for the first several hours of the strip is 40 - 70 percent, and gold recovery per pass is 75 percent.

Regardless of the current density, anode corrosion is severe. This is attributed to buildup of chlorides and fluorides on the carbon from the field heap solutions, and could possibly be alleviated by a deionized water wash prior to stripping. Mild steel anodes are unusable after one strip cycle. type 316 stainless anodes have to be cleaned after each cycle and replaced every four cycles. A titanium anode is being tried.

Melting

The wet cathodes from the electrolytic cell are immediately transferred to a Denver No. 4 tilting hearth furnace. The iron is fluxed off using sodium nitrate. The product of this single stage melt is a dore button assaying 950 fine or better.

Acid Washing

Following stripping, the carbon in the columns is fluidized with heap solution, then acid-washed in a closed-pumping cycle with dilute HCL until the pH stabilizes below 2.0. The acid wash solution is then neutralized with NaOH prior to being discharged to the barren pond. The columns are again fluidized until the effluent is clear, and then they are ready to be placed back on line. After eight months of continuous operation during which the carbon has never been removed from the columns, they are showing only minor signs of channeling and activity loss.

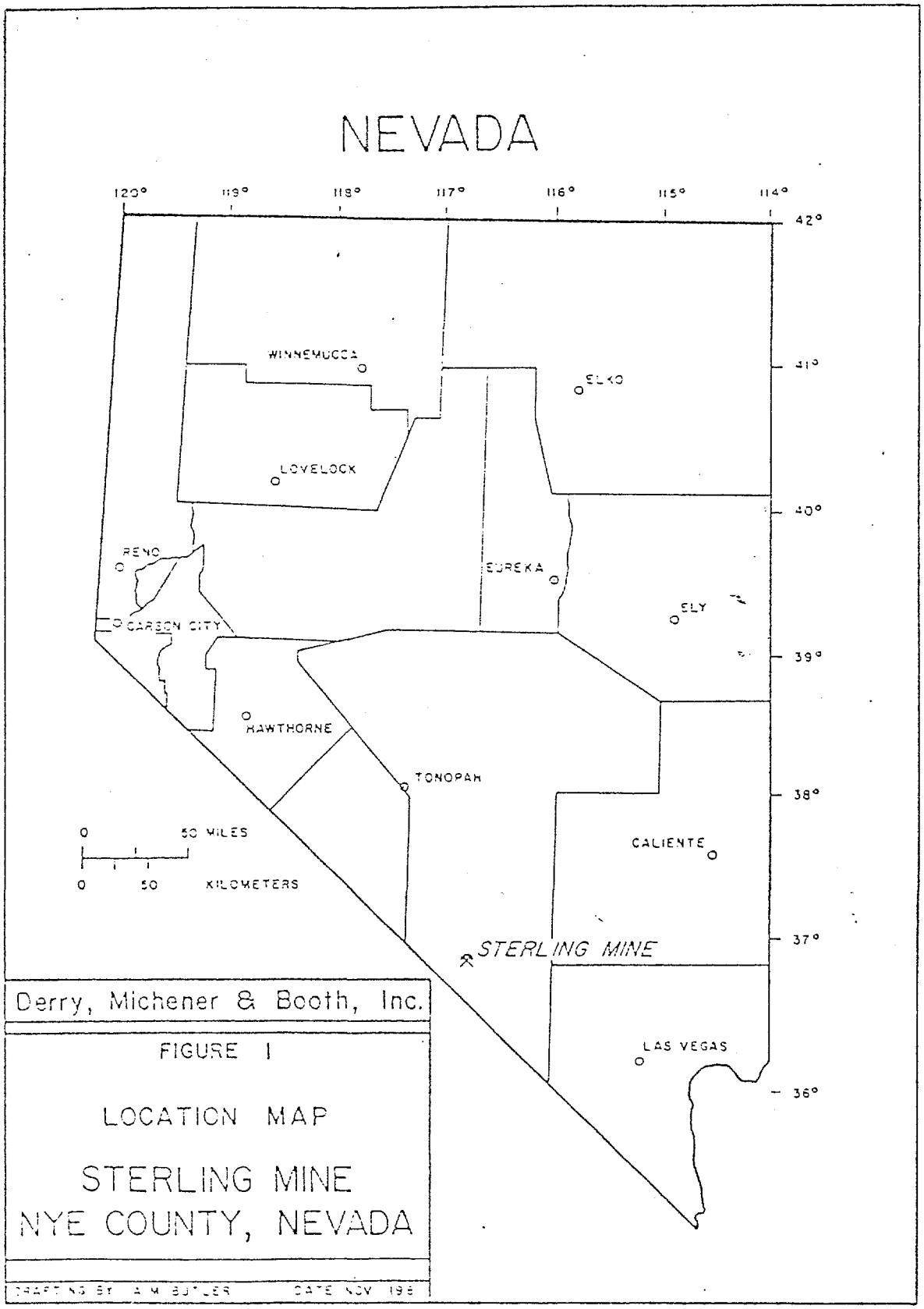
Conclusion

The Sterling ore body was not exposed on surface. It is too small and linear to be amenable to open cast mining. Yet, as a result of prudent exploration, efficient underground development and the application of low cost heap leaching technology, the ore body has begun to yield a profit to its owners. The Sterling story is a testimony to the fact that small scale mining is alive and well in these United States.

References

Cornwall, H. R., 1972, Geology and mineral deposits of southern Nye County, Nevada: Nevada Bur. Mines and Geology Bull. 77.

Cornwall, H. R., and Kleinhampl, F. J., 1961, Geology of the Bare Mountain quadrangle, Nevada: U. S. Geol. Survey Geol. Quad. Map GQ-157.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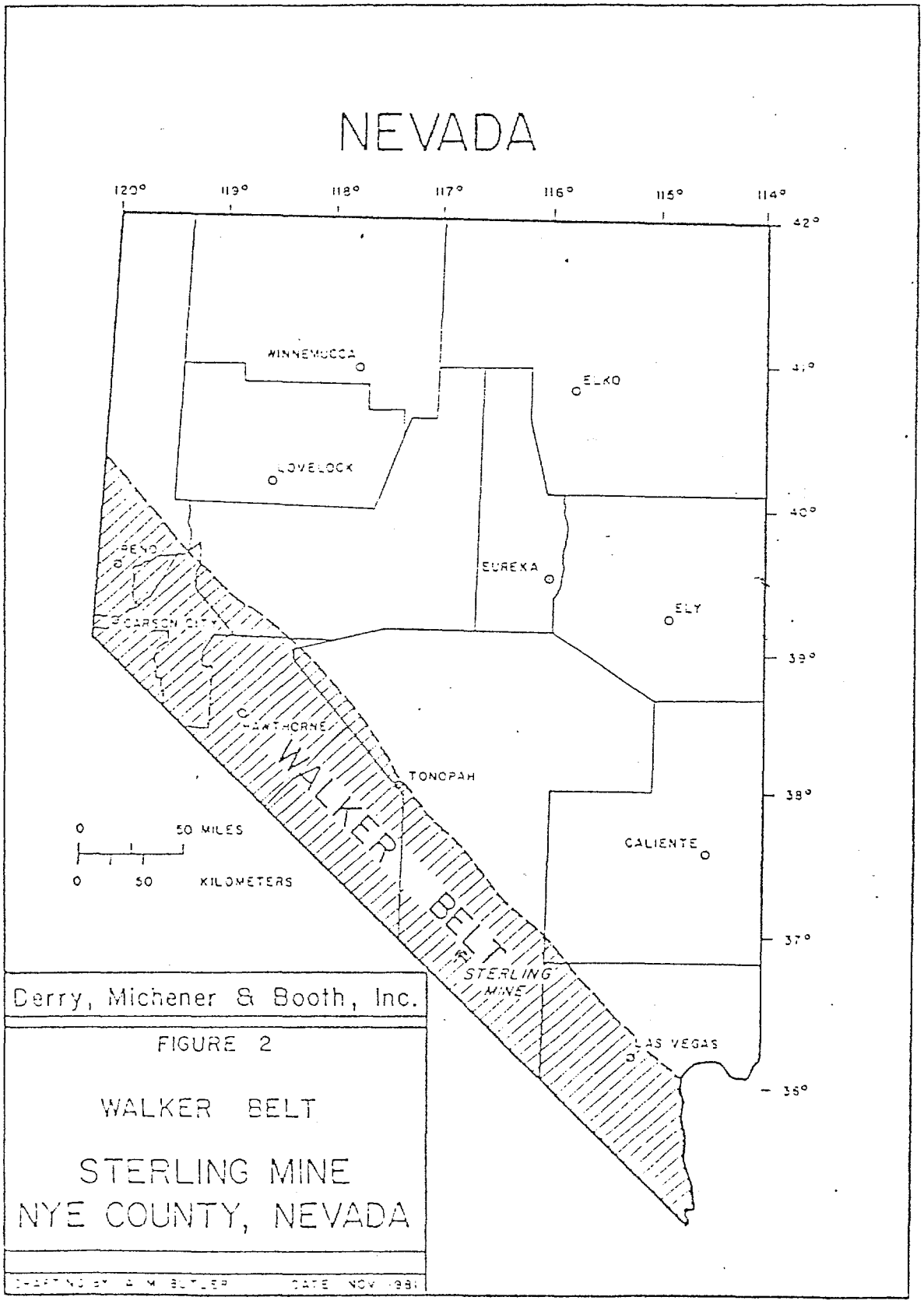
37°
36°
WINNEMUCCA
ELKO
LOVELOCK
RENO
EUREKA
ELY
CARSON CITY
HAWTHORNE
TONOPAH
CALIENTE
STERLING MINE
LAS VEGAS
0 50 MILES
0 50 KILOMETERS
Derry, Michener & Booth, Inc.
FIGURE 1
LOCATION MAP
STERLING MINE
NYE COUNTY, NEVADA
DRAFTING BY A.M. BUTLER DATE NOV 198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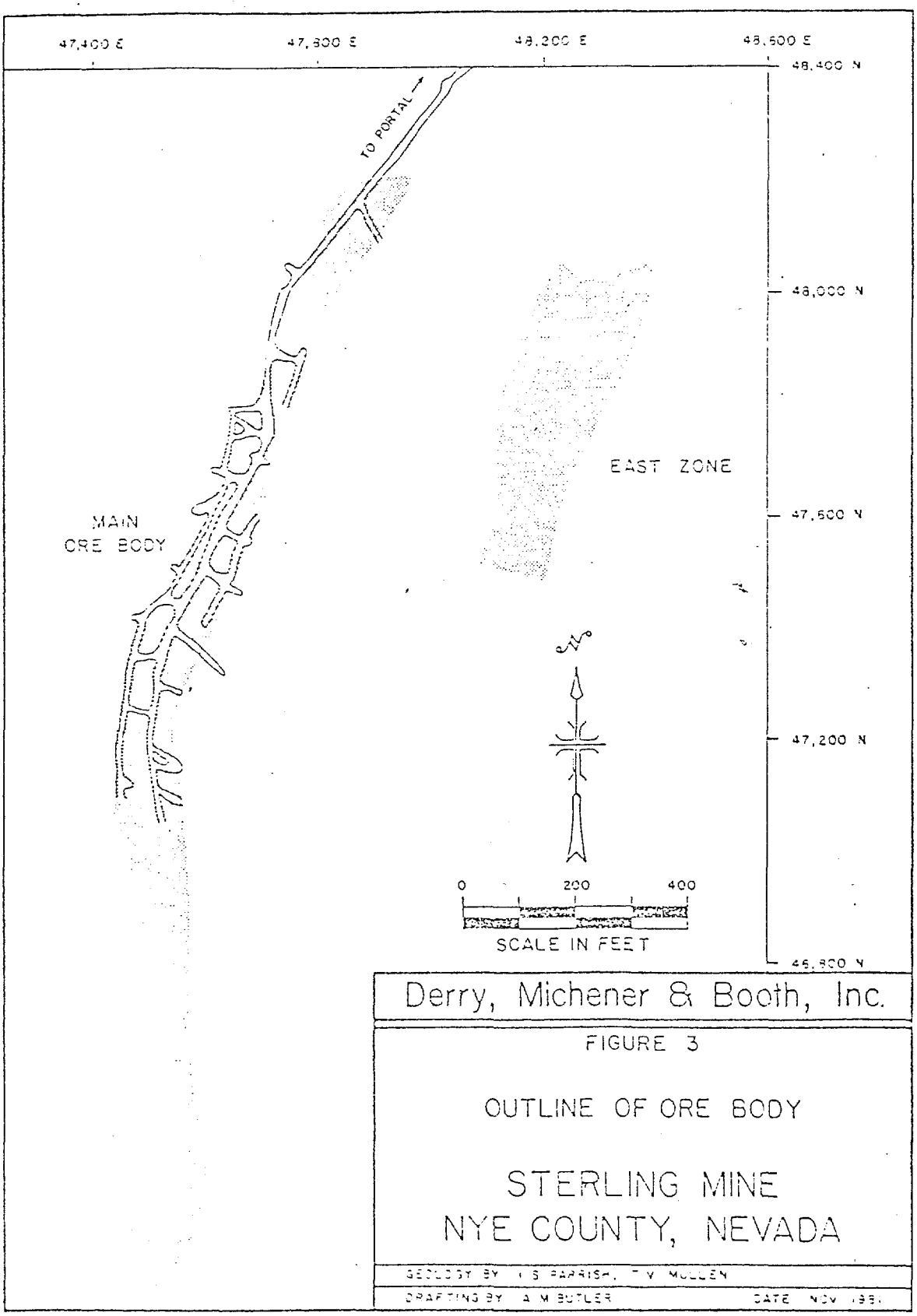
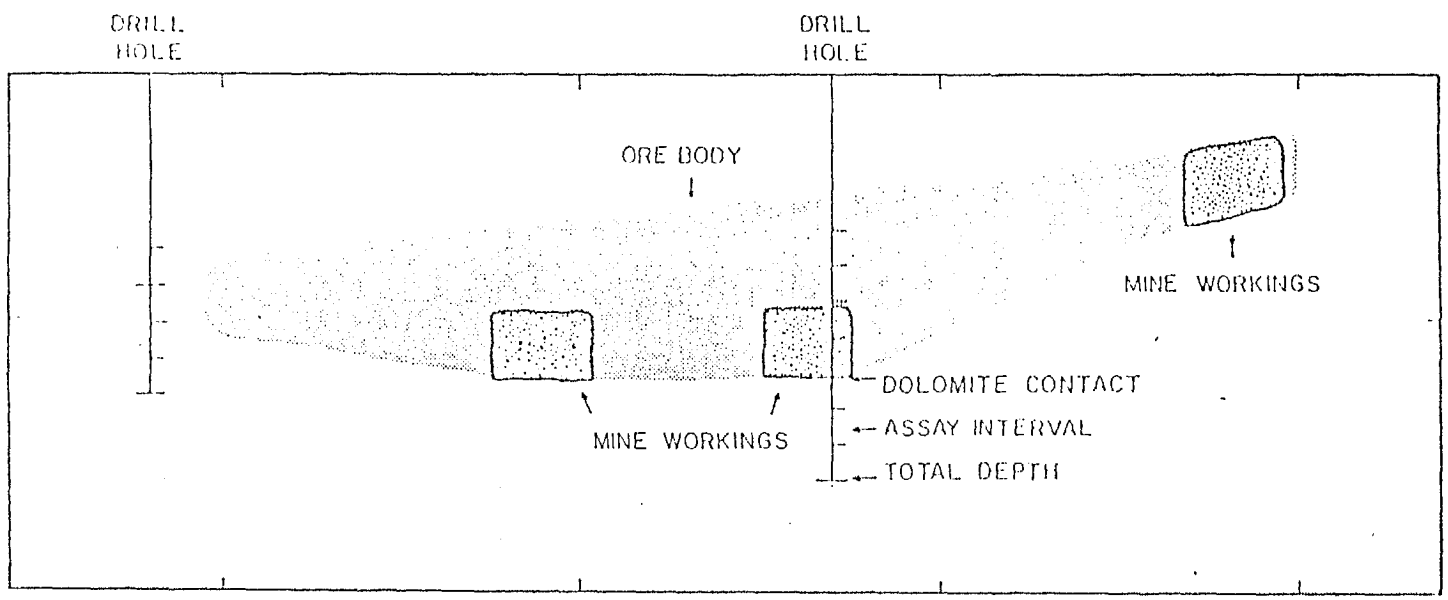
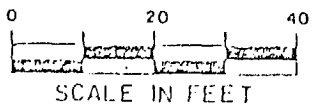
ELKO
LOVELOCK
RENO
CARSON CITY
EUREKA
ELY
HAWTHORNE
WALKER BELT
TONOPAH
CALIENTE
STERLING MINE
LAS VEGAS
0 50 MILES
0 50 KILOMETERS
Derry, Michener & Booth, Inc.
FIGURE 2
WALKER BELT
STERLING MINE
NYE COUNTY, NEVADA


47,400 E
47,800 E
48,200 E
48,600 E
48,400 N
48,000 N
47,600 N
47,200 N
46,800 N
TO PORTAL
EAST ZONE
MAIN
ORE BODY
0
200
400
SCALE IN FEET
Derry, Michener & Booth, Inc.
FIGURE 3
OUTLINE OF ORE BODY
STERLING MINE
NYE COUNTY, NEVADA
GEOLOGY BY
DRAFTING BY A M BUTLER
DATE NOV





Derry, Michener & Booth, Inc.

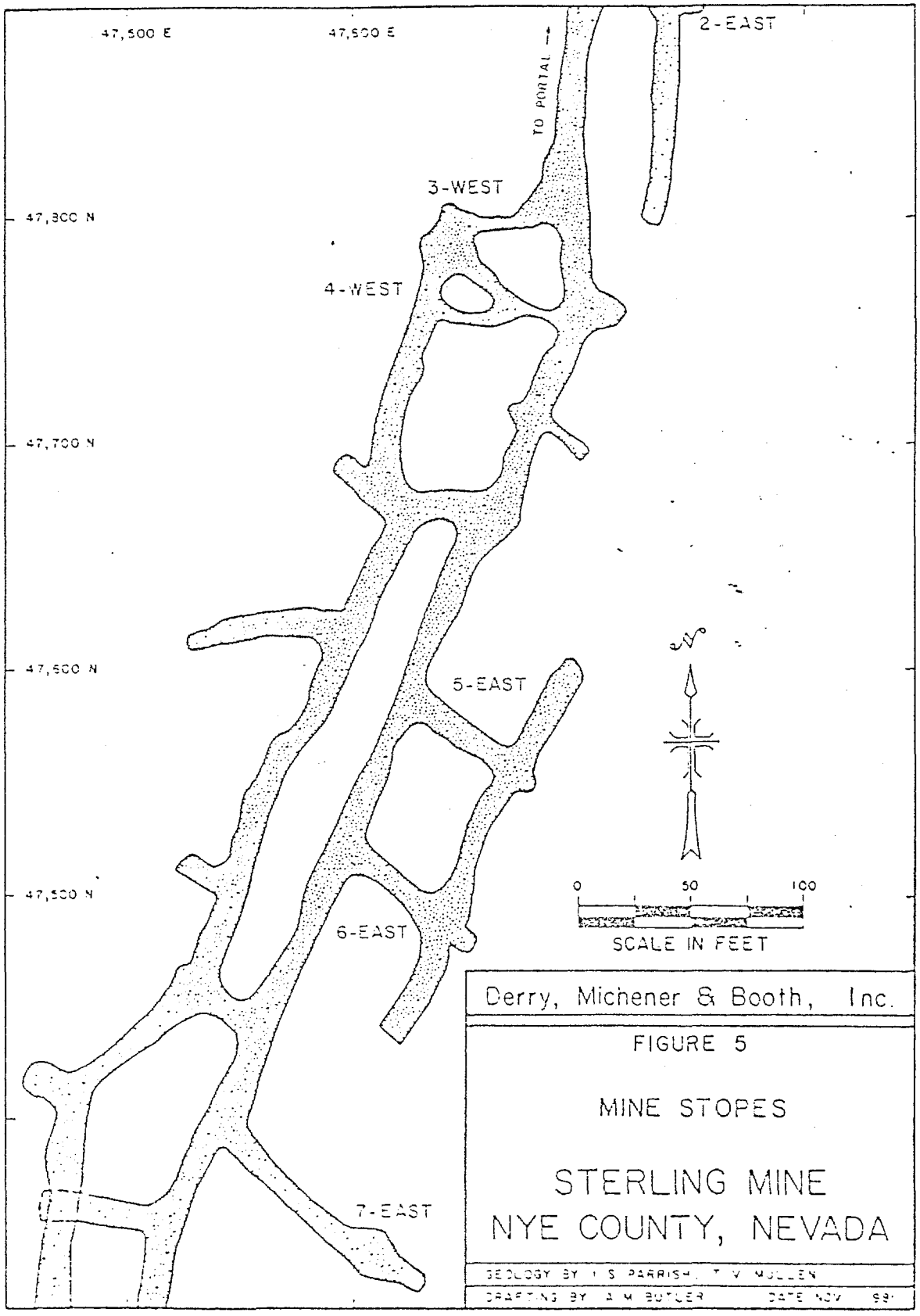
FIGURE 4

IDEALIZED SECTION THROUGH ORE BODY

STERLING MINE
NYE COUNTY, NEVADA

GEOLOGY BY: L.S. PARRISH, T.V. MULLEN

DRAFTING BY: A.M. BUTLER DATE: NOV 1981


47,500 E
47,600 E
TO PORTAL
2-EAST
3-WEST
4-WEST
5-EAST
6-EAST
7-EAST
47,800 N
47,700 N
47,600 N
47,500 N
0
50
100
SCALE IN FEET
Derry, Michener & Booth, Inc.
FIGURE 5
MINE STOPES
STERLING MINE
NYE COUNTY, NEVADA
GEOLOGY BY
DRAFTING BY A M BUTLER
DATE NOV

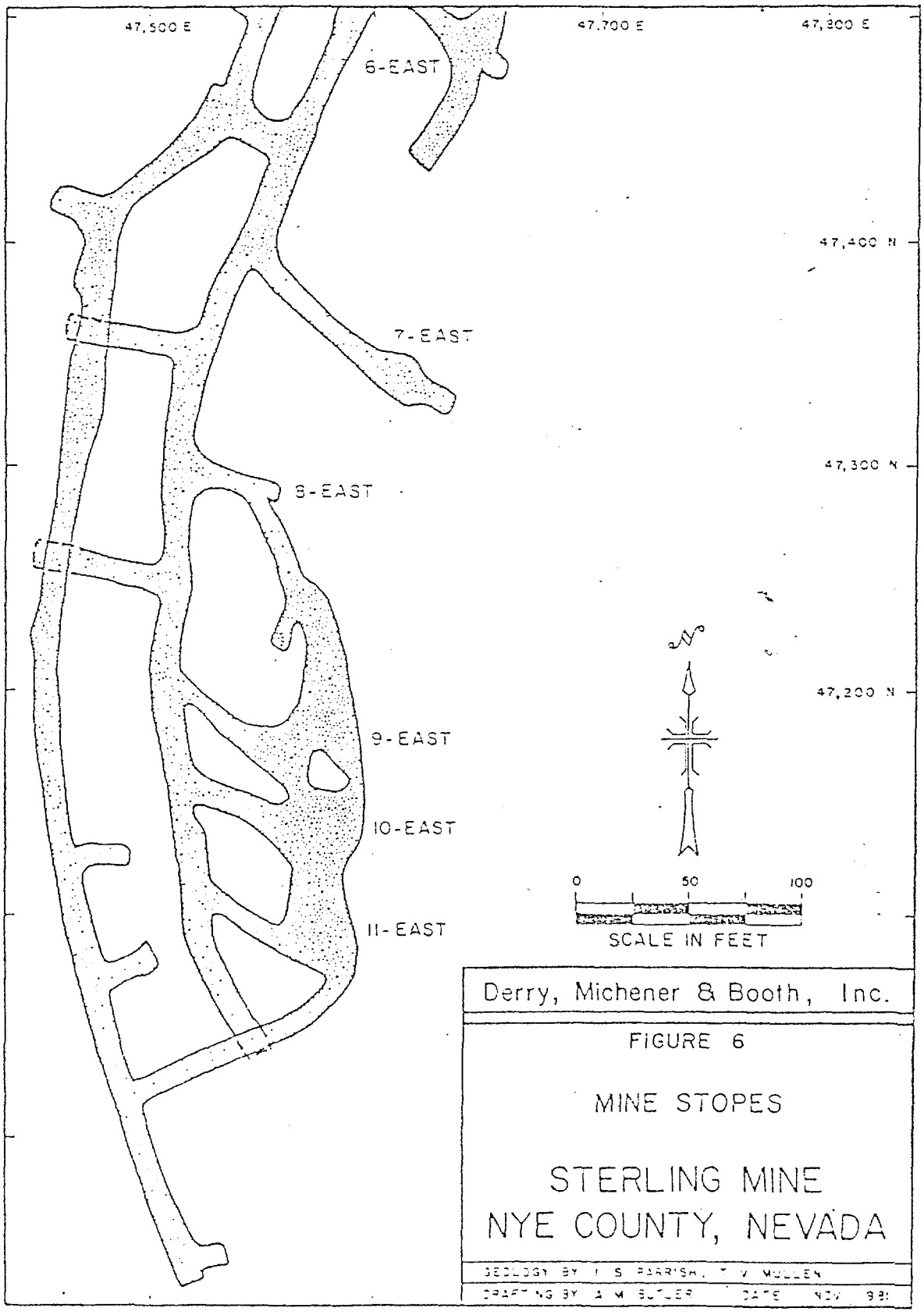

47,500 E
47,700 E
47,300 E
6-EAST
47,400 N
7-EAST
47,300 N
8-EAST
47,200 N
9-EAST
10-EAST
11-EAST
0
50
100
SCALE IN FEET
Derry, Michener & Booth, Inc.
FIGURE 6
MINE STOPES
STERLING MINE
NYE COUNTY, NEVADA

Plate Captions

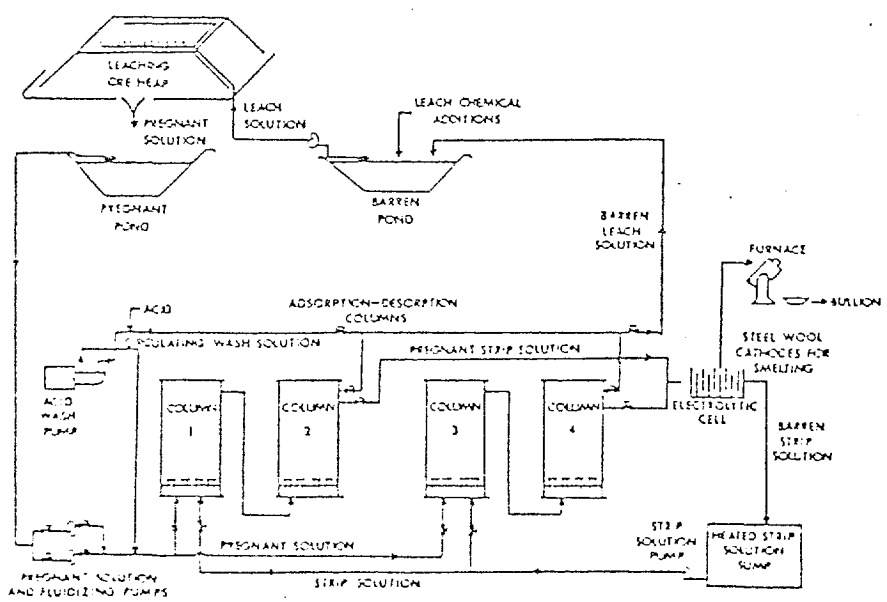
PLATE 1: The heap leach system is designed for quick-leaching, high-grade gold ores at the rate of 300 tons ore per day. Each 12,000 ton heap drains to a separate pregnant pond, which allows countercurrent leaching to extract the last traces of gold from the ore.

PLATE 2: Heaps of 12,000 tons each are built, 15 feet high, atop gravel-covered PVC pads. Lots of land is available on the alluvial fan below the mine.

PLATE 3: Heaps are built in three five-foot lifts. The heap in the background will remain under leach for at least two months. The first layer of the adjacent heap is under construction in the foreground.

PLATE 4: Individual pregnant ponds are below the heaps. From there, solution is pumped through the recovery trailer, and is then discharged to the barren pond in the right foreground. Solid caustic soda and cyanide are manually added to the barren pond before solution is recycled to the heaps.

PLATE 5: The recovery trailer contains four recovery columns holding 1000 pounds carbon each. All construction was done in Reno, then the system was transported 700 miles to the project site. Key features include insulated adsorption/stripping columns and a rectifier/electrolytic cell system for in-place recovery of gold.



STERLING MINE RECOVERY SYSTEM PLATE 5

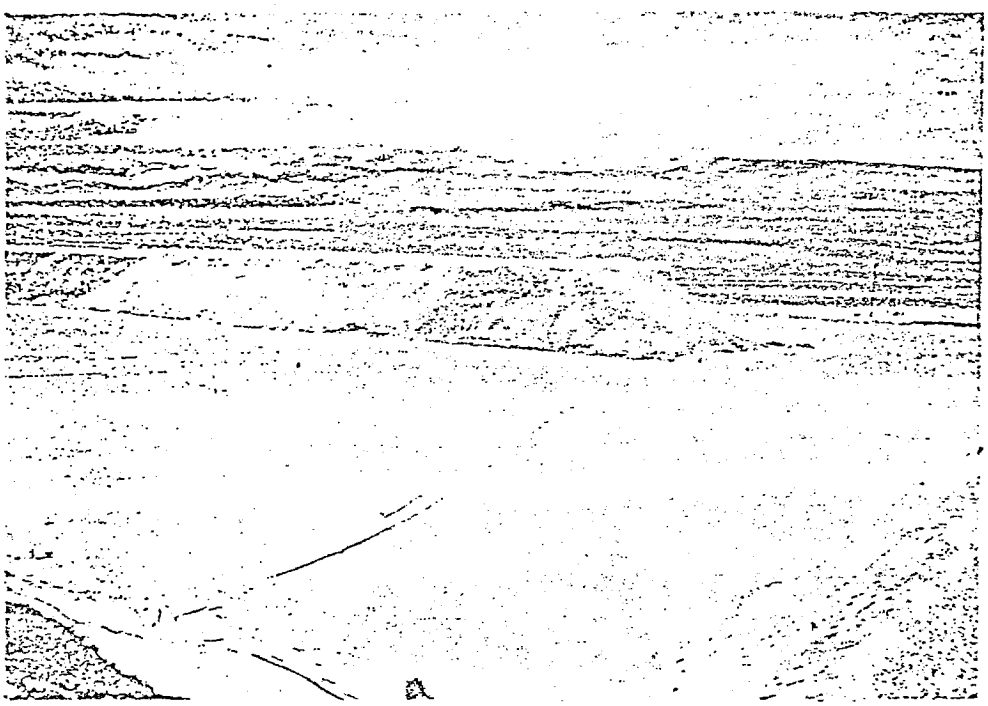


PLATE 2

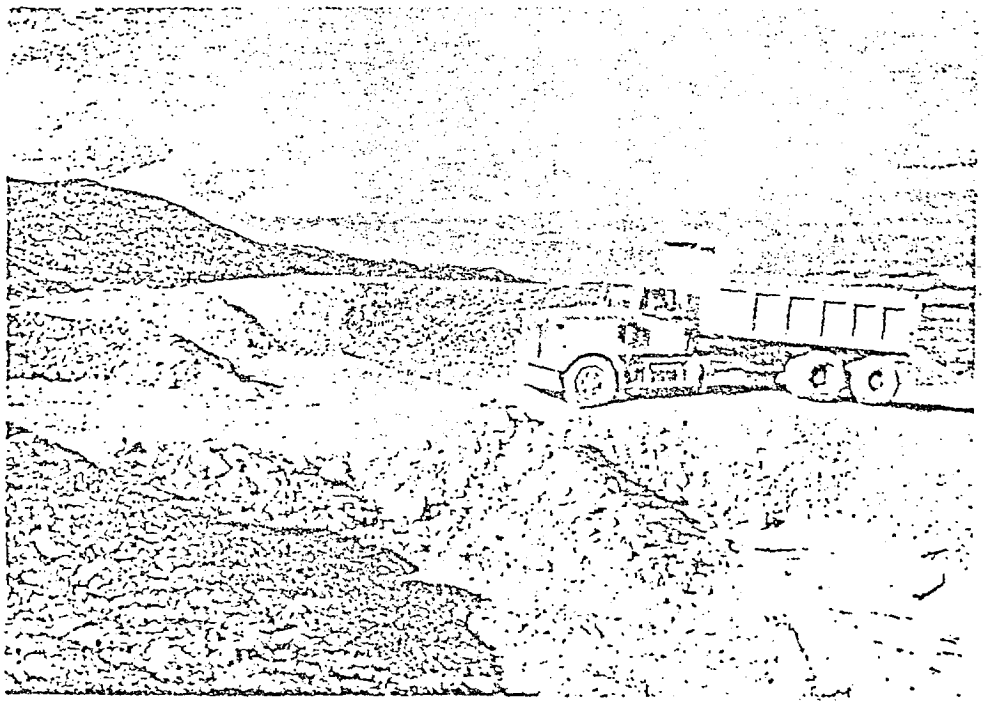


PLATE 3

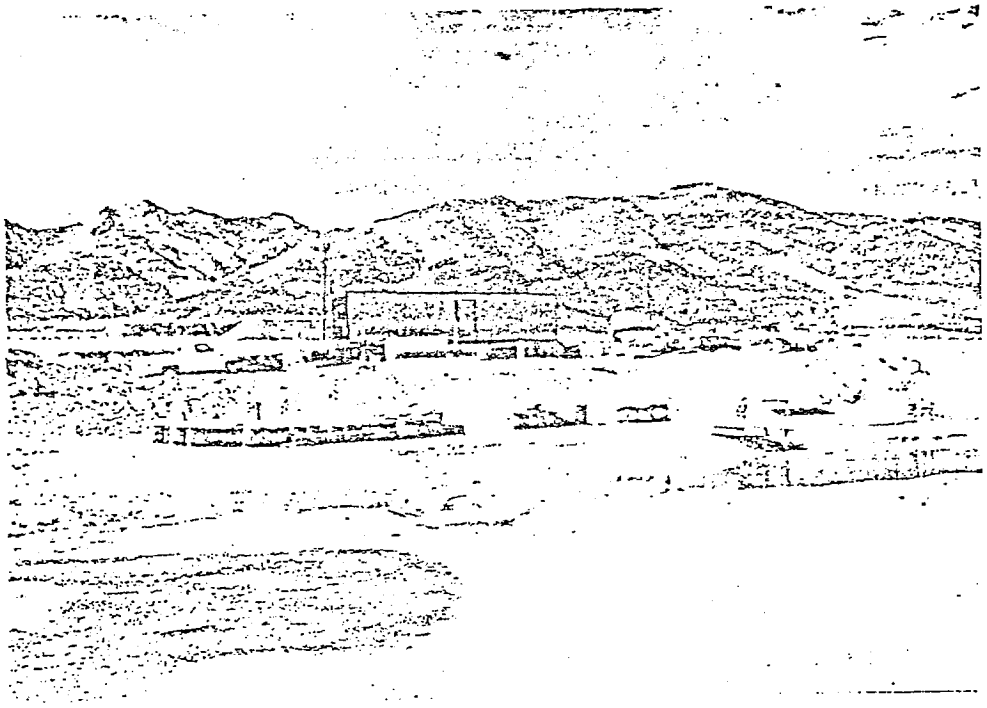


PLATE 4

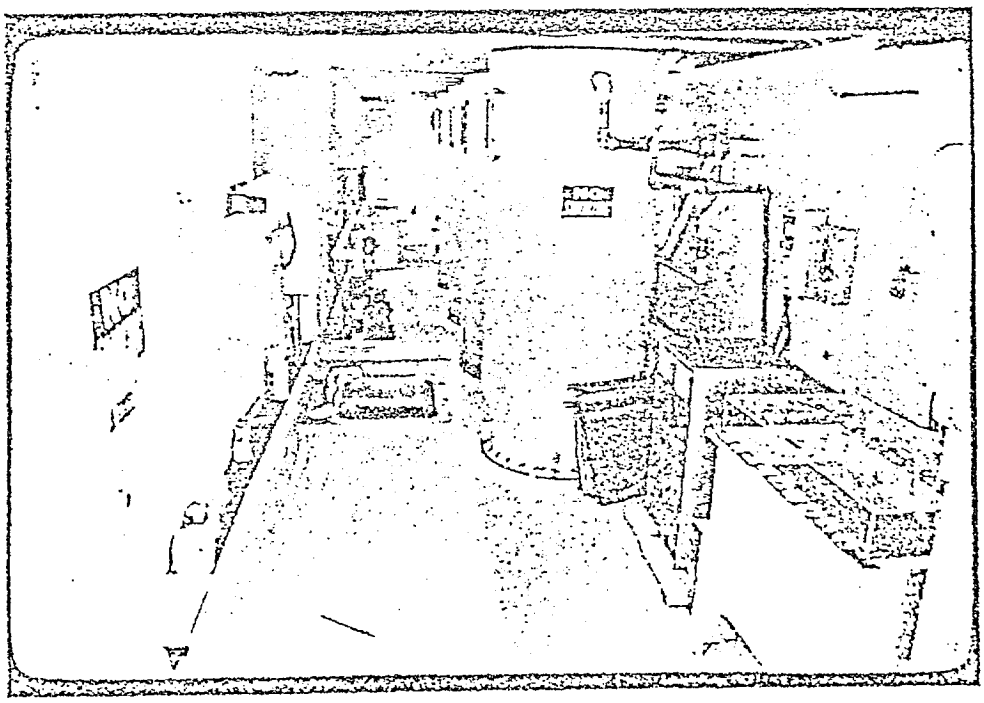


PLATE 5

Borealis Plant

The Borealis Plant of Houston International Minerals Co. was visited on the afternoon of June 9. This plant is located southwest of Hawthorne on the Lucky Boy pass road. The plant superintendent is Mike Hall and we were taken on a tour of the operations by Eric Daniels, operations superintendent.

Borealis heap leaches 2,000 tpd of gold ore from 3 small open pits. The heads from the main Borealis pit run .15 opt Au and 0.2 opt Ag and ore from the Jaime's Ridge pit which they were running during our visit was somewhat higher in grade.

Ore is crushed to 80% minus one inch in an open circuit "gravel plant" type portable crusher operating five days per week. Ore is placed in 20,000 ton pads, leached for 30 days and then removed to a tailing dump. The total cycle takes 40 days and consists of: load - 3.5 days, leach - 30 days, drain & dry - 3 days, unload - 3.5 days.

Borealis agglomerates the ore with 10 lb/ton of cement and a 20% NaCN solution. Agglomeration is done at four conveyor transfer points and ore is then stacked with a radial stacker. The stack holds 20,000 tons and the ore "cures" in the stack for at least a week before being transferred to the leach pad. Overall gold recovery is 75% and 67.5% is recovered in the first week of leaching. They feel that most of the gold is dissolved during the curing phase. Silver recovery is only 15-30%.

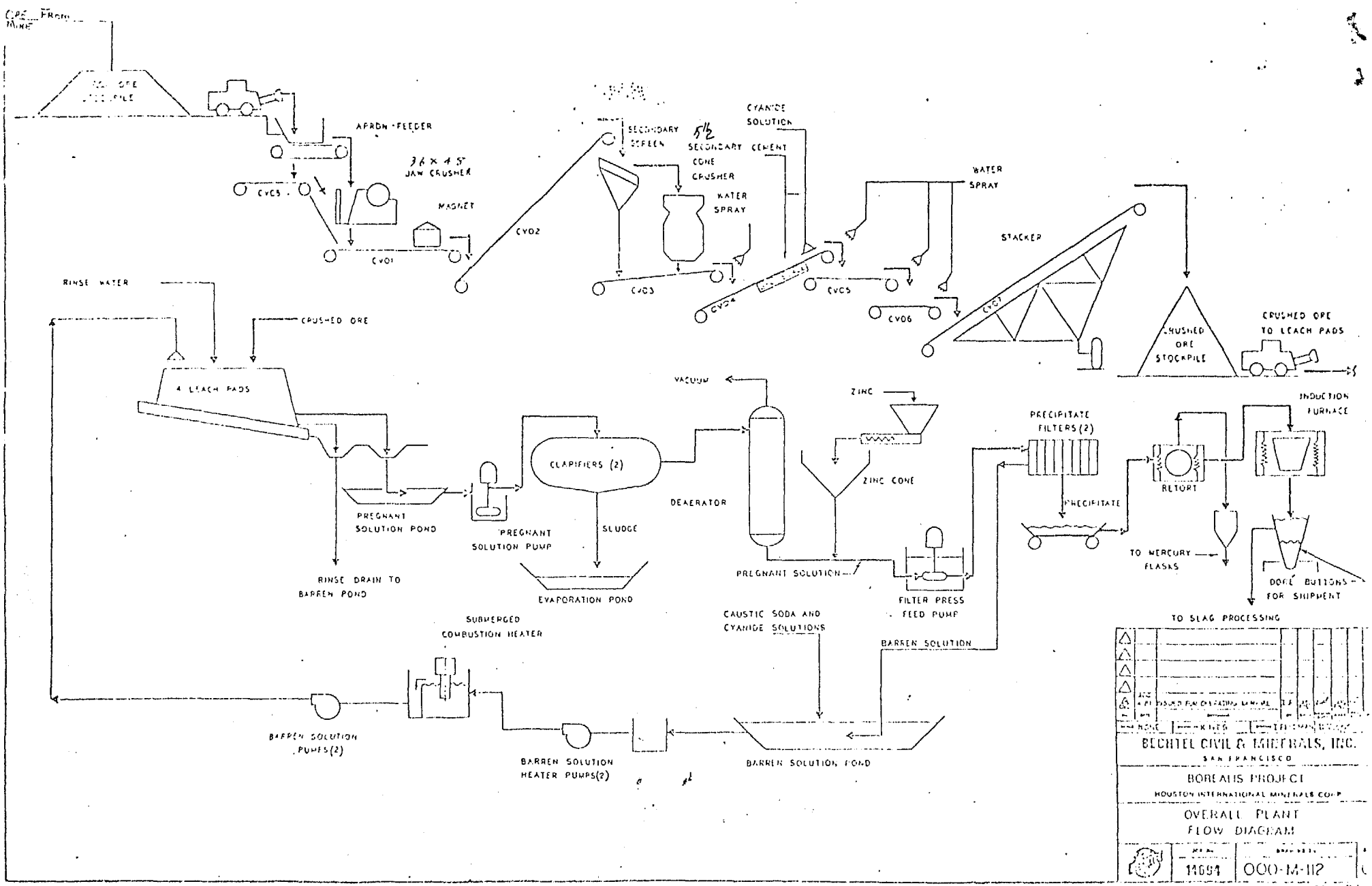
There are four leach pads holding 20,000 tons each. Pad construction consists of a plastic membrane on the bottom covered with 2 ft. of clay, 7 inches rubberized asphalt and 18 inches of gravel. The initial asphalt decomposed because it was the "fast cure" type using sterates which reacted with the caustic in the leach solution. The pads are sloped 4% in two directions with berms between pads to control solution flow. When a pad is excavated, earth ramps are built over the berms to allow front end loaders and trucks to work on two sides of the heap to speed up excavation and ore drying.

Solution ponds are constructed of 80 mil high density poly ethylene (HDPE) laid over two feet of clay.

Gold is recovered in a conventional Merrill-Crowe system. Soluble oxygen is reduced from 8 ppm to less than one ppm and solids are reduced to less than two ppm before precipitation. A "light" slurry of diatomaceous earth is fed into the filter press to aid filtration.

All assaying is done by fire assaying (Chiddy method for solutions) except low grade silver solutions.

The Borealis ore contains mercury and this is removed in two retorts. Conditions are 24 hours at 1,400 degrees F with a 3 cfm draft to each retort.


ORE FROM MINE
ORE STOCKPILE
APRON FEEDER
36 x 45" JAW CRUSHER
MAGNET
CV01
CV02
CV03
CV04
CV05
CV06
CV07
SECONDARY SCREEN
5½ SECONDARY CONE CRUSHER
WATER SPRAY
CYANIDE SOLUTION
CEMENT
STACKER
CRUSHED ORE STOCKPILE
CRUSHED ORE TO LEACH PADS
RINSE WATER
CRUSHED ORE
4 LEACH PADS
PREGNANT SOLUTION POND
RINSE DRAIN TO BARREN POND
PREGNANT SOLUTION PUMP
CLARIFIERS (2)
SLUDGE
EVAPORATION POND
VACUUM
DEAERATOR
ZINC
ZINC CONE
PREGNANT SOLUTION
FILTER PRESS FEED PUMP
PRECIPITATE FILTERS (2)
PRECIPITATE
RETORT
INDUCTION FURNACE
TO MERCURY FLASKS
DORE BUTTONS FOR SHIPMENT
TO SLAG PROCESSING
CAUSTIC SODA AND CYANIDE SOLUTIONS
BARREN SOLUTION
BARREN SOLUTION POND
BARREN SOLUTION HEATER PUMPS(2)
SUBMERGED COMBUSTION HEATER
BARREN SOLUTION PUMPS(2)
BECHTEL CIVIL & MINERALS, INC.
SAN FRANCISCO
BOREALIS PROJECT
HOUSTON INTERNATIONAL MINERALS CORP.
OVERALL PLANT FLOW DIAGRAM
14094
000-M-112

Nevada Molybdenum Operation - Anaconda Company

The Nevada molybdenum operation was toured on June 9, 1983. We talked with Mr. Tom Parker, Plant Manager and Mr. Paul Valenti, the Mill Superintendent. Primary purpose of the visit was to review the difficulties the plant is having with the SAG and ball mills supplied by Koppers Company.

The plant was engineered and constructed by Bechtel Corporation at a total cost of about $240 million dollars. Engineering was started in 1979, construction began in 1980 and was completed in 18 months. This cost included housing in Tonopah, 18 miles of paved road, etc. The plant has design capacity of 20,000 MTPDC (22040 STPD). Molybdenum orebody reserves are about 200,000,000 tons containing about 0.091 percent Mo and about 0.1 percent Cu. In addition to the molybdenum reserves there is a copper orebody containing about 100,000,000 tons of between 0.35 and 0.40 percent total copper. The amounts of sulfide and non-sulfide copper has not been determined. My overall concept of the plant is that a great deal of money was spent at least partially needlessly. The office building is almost palatial. The computer area was a showcase of equipment even though they are tied into a central computing system in Denver. The computer equipment that we saw had been downgraded as a money saving step (mostly rented). Apparently Anaconda had a considerable amount of money to spend and Bechtel did their best to spend it. Even with the money spent there are and were areas of poor design and performance excluding problems with equipment. Some of these items are as follows:

a. Process design was apparently based on averages and not ranges. SAG mill circuits are noted for the wide variations in capacity and apparently this was not considered. Also flotation circuit was designed for 0.1 percent copper and not for the range of up to 0.5 percent. Flotation capacity was undersigned.

b. SAG and ball mill feed and discharge systems were poorly designed including cyclone underflow manifolds, screen installation, etc. Plant was shutdown for three months to rectify some of these problems along with others.

c. Highly automated molybdenum concentrate load out system designed by E&M Systems in Tucson was unworkable. It has to be used as a highly labor intensive manual system.

Some equipment problems that were experienced as follows:

a. Bearing problems with the Koppers' SAG and ball mills.

b. A infrared moly concentrate dryer was installed that did not work. This had to be replaced by a hollow-flight screw conveyor.

c. Severe problems have been experienced with the traction type 400 ft. diameter Eimco thickeners. This could have been operating problems and not necessarily equipment problems.

The major equipment problem in the plant has been bearing problems with both the SAG mills and the ball mills, i.e. losing bearing clearance. The reason for the problem is still unclear whether it is a manufacturers design problem or whether it is a construction installation problem. A million dollars worth of instrumentation was added to monitor the bearing clearance of the 4 mills; the 2 SAG mills and the two ball mills. Mr. Valenti feels that the mills can be operated; however, only running time will resolve the question.

The SAG mills discharge onto double-deck Tyler screens. The screen incline had to be changed from 20 degrees to 11 degrees in order to improve screening efficiency and to wash the oversize. The oversize (+ 3/8 material) is recirculated by conveyors. The screen undersize along with the ball mill discharge is pumped to cyclones with Warman pumps. Initially GIW pumps were installed but because of maintenance costs and other problems were replaced with the Warman pumps. Cyclone underflow goes to the ball mills with the cyclone overflow going to flotation.

The copper molybdenite bulk flotation circuit consists of a rougher and two stages of cleaner flotation. Bulk cleaner flotation is greatly undersized when the copper in the feed exceeds the average by very much. The bulk concentrate contains about 15 percent copper and 15 percent Mo. Reason for the low grade copper is chalcocite coated pyrite.

Copper and molybdenum is separated using Na_2S in excess of 100 lbs/ton. Molybdenite cleaning circuit is standard. Initially infrared molybdenite dryers were installed but did not work. Replaced these with hollow-flight type dryers.

A ferric chloride molybdenite leach circuit was installed in order to produce a premium grade concentrate containing less than 0.15 percent copper. Presently moly concentrate from Butte is being leached. This concentrate contains between 1.0 and 2.0 percent copper and no problem is experienced reducing this amount to less than 0.15 percent. Apparently no problems are experienced with putting MoS_2 into solution as was experienced with test work at Mission.

Leached molybdenite concentrate is washed using two stages of drum filtration. Considerable corrosion is experienced with the filters along with other aspects of the plant. The product is dried and placed in fabric bags containing about 5500 pounds. According to Anaconda the use of these bags is considerably less expensive than using 55 gal. barrels.

The plant personnel are installing a cyclone system in order to cyclone the tailing. The cyclone underflow will be pumped directly to the dam to be used to build the tailing dam berm. Cyclone overflow will go to the thickener and the thickener underflow will go to the dam separately. Primary purpose of the installation is due to the difficulties that have been experienced with the thickeners. These difficulties may have been due to the dispersed nature of the tailing because of not using lime in the flotation circuit. Flocculants and some lime have reduced some of the thickener problems.

Many of the problems seen at the Nevada molybdenum project were similar to the problems previously seen at new Chino concentrator also designed and built by Bechtel.

Candelaria Operation

The Candelaria operation was visited on June 10, 1983 and we talked with Messrs. Keith Roach, General Manager, Bill Jacabs, Operation Manager and Ralph Van Arsdale, Process Superintendent.

Candelaria is an open pit, heap leach operation. Reserves and grade vary with the price of silver but it has been indicated that there is about 13 million tons at 3.15 opt silver and 0.002 opt gold. Stripping ratio is about 2.6 to 1.

The ore is crushed in open circuit to about minus 1 inch at the rate of 10,000 TPD, 5 days per week or between 50,000 and 60,000 TPW. The crushed ore is agglomerated with a caustic cyanide solution using two short belts. Agglomerating solution contains about 15 lb/ton of NaCN and 1.0 lb/ton NaCN.

The crushed agglomerated ore is conveyed to a truck load out bin and trucked to the leach pad area.

The leach pad seal was designed by Dames and Moore. The pad is approximately 18 inches of local compacted clay and silt from dry lake beds. They stated that the clay was compacted to 100 percent. Stated solution loss was 0.07 gpm per acre or 100 gallons per day per acre. They do not have a gravel layer between the clay seal and the ore. Overall projected height of the leach dumps is 60 ft.

The leach pads have been constructed on terrain that slopes an average of 3.5 percent in 2 directions with a maximum slope of 5 percent. Slope of the top of each lift is approximately 1-1/2 and 2-1/2 percent in each direction.

A lift of ore is approximately 10 ft. high. Each lift is leached for between 30 and 60 days even though optimum leach time is about 90 days. Leach solution contains about 3.5 lb. NaCN per ton of solution with a pH of about 11.0 using NaOH. Leach recovery of silver is somewhat less than 50 percent. Leach solutions are distributed "Wobbler" sprayheads on 40 ft. centers.

Leach solution distribution lines are buried during winter to allow for wintertime operation. However the system does not work well. Solutions are heated using a high efficiency submerged burner. However the CO_2 from the burner consumes all the NaCN and lowers the pH. A large amount of NaOH is required to raise the pH to a suitable level.

Pregnant solution collection ditches and ponds are lined with HDPE which is considerably better and cheaper than Hypalon. HDPE is much more abrasion resistant than Hypalon and thus surface preparation is much less. Average year around water consumption is about 350 to 400 gpm. They estimate evaporation loss of about 10 percent.

The plant was engineered and constructed by Bechtel Corporation at a total cost of $40 million. It was indicated that construction was good but engineering was poor. Plant began operation in October, 1980.

The pregnant solution pond has a capacity of 18 million gallons. Pond was designed to hold NaCN bearing solutions in case of a flood.

Silver is recovered using the standard Merrill-Crowe process. The pregnant solution is filtered using U.S. pressure filters and then goes to a polished pregnant solution storage tank. From this tank the pregnant solution goes to the vacuum tower and to zinc precipitation.

The precipitate is smelted once per week and with single stage smelting produces a 980 doré. Prior to smelting, the precipitate is dried in pans using weed burners to 25 to 30 percent moisture. Smelting furnace is propane fired and is rated at 15 million btu.

Slag is crushed and tabled to remove the silver shot. However the table tailing contain between 1 and 600 opt silver with the average of 50 opt. Slag is stored awaiting a good smelter schedule. Weekly silver production is about 85,000 oz.

An extensive pilot plant program was run prior to start-up. Columns 4 to 5 ft. in diameter by 12 to 15 ft. high were run. In addition a heap leach test of 10,000 tons of ore was run. Cost of pilot plant was $300,000. Cost of 115 ft. x 115 ft. asphalt pad was about $35,000. They recommended using a 150 ft. x 150 ft. pad in order to contain wind blown sprays. Also stated that you shouldn't go much less than a 10,000 ton leach test because of side effects.

TW-MNR Associates

The TW-MNR plant was visited on the afternoon of June 10. The manager is Dave Hamilton and Irv Crist is the metallurgist. The plant is located about 15 miles west of Tonopah just off the Tonopah-Reno highway. The full name of the operation is Tonopah West - Metalgeselschaft Natural Resources.

The primary reason for visiting this plant was to observe a flowsheet using SO_2 to leach a manganiferous silver ore. This is one of the flow sheets proposed for the Hardshell project. The TW-MNR ore contains much less manganese (and less silver) than Hardshell so a manganese filtration step is not required, but the basic process of an SO_2 leach followed by pH adjustment and cyanide leach would be similar.

The TW-MNR plant treats ~~tailing~~ from the early Tonopah ores containing 2 opt Ag about 1 opt of which is tied up with manganese and not directly leachable with cyanide. The ore contains about 0.2% manganese. Straight cyanidation without using the SO_2 preleach recovers approximately 50% of the silver. With the SO_2 preleach, recovery is increased to 75%.

The plant treats 1,300 tpd and operates seven days per week. The flowsheet consists of the following steps:

1. Flash grind in 9 x 14 ball mill to 75% minus 200 mesh.

2. Screen ball mill discharge at 10 mesh and reject oversize.

3. Minus 10 mesh is leached for one hour at 60% solids with SO_2 gas sparged into the leach tank. SO_2 is produced by a sulfur burner and 11 pounds of sulfur are consumed per ton of ore. The pH of pulp is 4.5. They have had much trouble with the sulfur burner used to produce SO_2 and are installing a liquid SO_2 system for back up.

4. Following the SO_2 leach, air is sparged through the pulp for 30 minutes to remove all SO_2 and add oxygen to the pulp for subsequent cyanidation. Eleven pounds per ton of lime is added to raise the pH to 11.0.

5. The ore is then leached for seven hours with cyanide at 45% solids. Cyanide consumption is 3.5 pounds per ton ore.

6. Dissolved silver values are recovered by a carbon-in-pulp circuit using 6 x 12 mesh coconut shell carbon in 4 stages of pachuca tanks. They carry a 50 ton carbon inventory and carbon loss is 0.6 lb/ton ore.

7. Loaded carbon is washed with 10% nitric acid and then the carbon is stripped in a batch process. Strip conditions are: seven hour retention time, 260°F at 40 psi, strip solution consists of 0.1% NaCN and 1% caustic. Eight bed volumes are circulated through the strip tank to achieve 95% strip recovery. The electrolyte from the strip tank will assay approximately 10% of the carbon loading and runs as high as 24 opt.

8. The electrolyte from the strip tank is cooled to 170°F and then passes through a standard electrolytic cell using steel wool as cathodes. It is necessary to bleed some spent electrolyte each day to control build up of impurities.

9. Loaded steel wool cathodes are fire refined each day to produce 2,000 oz/day of 99.7% silver doré.

Jerritt Canyon - Freeport Gold Co.

The Jerritt Canyon plant was visited on the morning of June 13. The plant is located some 45 miles north of Elko on Highway 51. The Mill Manager is Dave Collins and he conducted our tour. We also met Joe Murray, Vice President - Operations.

The attached write up on Jerritt Canyon gives an overall description of the plant so only items of particular interest will be commented on here.

1. About one-half of the ore reserves are carbonaceous and must be oxidized with chlorine before cyanidation to prevent the carbon in the ore from absorbing the soluble gold-cyanide complex and thereby preventing recovery of the gold. In practice it has been found that even the oxide ore (non-carbonaceous) benefits from chlorination (increases recovery from 80% to 85%) so all ore is now chlorinated. Recovery on carbonaceous ore is 90%. Feed grades were given as .25 opt Au for carbonaceous and .20 Au opt for oxide ores.

2. Chlorine is received in trailers and these are parked and hooked up to the distribution system directly so there is normally no transfer to storage tanks.

3. The grinding mills were initially set up as semi-autogenous mills with a steel ball charge. This arrangement caused a critical size build up at around three inches with consequent very low capacity. Grinding was very inefficient as evidenced by a high ball mill discharge pulp temperature. The grinding circuit was converted to ABC by the addition of four and a quarter foot short head crushers and the capacity returned to design or higher (3,300 tpd at present). The crushers effectively removed the critical size rock so the mills now run without a ball charge.

4. The grind is fairly fine at 75% minus 200-mesh with 5% plus 65 mesh.

5. A preoxidation step was originally used where the pulp temperature was raised to 180°F and air was sparged into the pulp. This is no longer done.

6. The chlorination step involves the following conditions:

a - 12-hour retention time. Pulp is at 50% solids.

b - Pulp temperature of 100°F.

c - Initial pH of 9.5-10 with soda ash.

d - Chlorine gas is sparged into agitation tanks. Consumption is 45 lbs. chlorine per ton of ore. The pH drops to 5.6.

e - Before the cyanide leach, the pH is raised to 11 with 5-6 lbs/ton lime.

7. The cyanidation step is conventional CIP except that the external vibrating screens initially used to separate carbon from the pulp stream during counter current carbon advance have been replaced because of carbon attrition. They now use "V" shaped troughs made of screen material which sit in the pulp at the top of the agitated tank. This keeps the carbon in a given tank while the pulp overflows from tank to tank in the normal manner. Air jets on the outside of the screen troughs keep the carbon or trash from blinding the screen.

Periodically (once each day?) carbon and pulp are advanced one tank upstream by pump with barren carbon added to the last tank and loaded carbon withdrawn from the first tank by a conventional external vibrating screen. Cyanidation consumption is rather high at 1.5 - 2.0 lb/ton. The pulp density is carefully controlled at about 45% to prevent carbon from settling with low densities (-40%) or floating at high densities (+50%).

8. Loaded carbon (100-120 opt Au) is stripped in lots of six tons by passing six bed volumes of hot (290°F) caustic cyanide solution through for six hours. Stripped carbon assays 2 opt Au. Pregnant strip solution assays 600-800 ppm Au and this is cooled to below boiling (less than 196°F) and then filtered in a candle filter.

9. In a major departure from conventional practice the pregnant strip solution is then treated in the Merrill-Crowe zinc precipitation process rather than by electrowinning. Precipitate is then leached with acid to remove zinc, filtered, and retorted to remove mercury which is sold as a by-product. The precipitate assays 40% Au before acid leaching, 55% Au after acid leaching and 75% Au after mercury retorting. The reason given for choosing Merrill-Crowe over electrowinning was the quicker recovery of gold.

10. They mentioned that the mercury contains 12 opt Au and no buyers will pay for it so they may retreat the mercury to recover this gold.

11. The tailing area has a natural clay base so no seal was necessary. The dam itself has a clay core. Tailing going to the dam has a cyanide content of 0.1 gpl NaCN but tailing water contains only .03-.04 gpl. Tailing water is not recycled because of the high chloride content.

12. The plant cost was given as 65 million dollars for the surface plant including maintenance shops and 100 million for the entire plant.

13. The total work force is 240 with 80 in the mill and 70 in mine and mill maintenance.

Pinson

The Pinson operation was visited on the afternoon of June 13. The Manager of Pinson is Dan Harper and the Mill Superintendent is R. Bruce Thorndycraft. We were taken on a tour of the operations by the metallurgist, W. B. (Rusty) Craft. The two attached papers by Messrs. Thorndycraft and Craft give a very complete and up-to-date description of the Pinson plant so this report will give only general comments plus discussing their new heap leach operations.

1. Pinson currently treats 1,500 tpd through the leach plant. Published ore reserve grade is .12 opt Au. They are presently using a 0.04 opt cut off. The 1,500 tpd capacity is 50% higher than the design tonnage of 1,000 tpd. This is due to both overdesign and the fact that the ore proved to be somewhat softer than expected. Although the grinding section handles the added tonnage without problems, rentention time in the leach and CIP circuits are shortened significantly. However, they have managed to modify the circuit and maintain the designed recovery of 85%.

2. Pinson design incorporated the in-pulp carbon screen (transverse launder) which was developed in South Africa. This will probably become standard CIP design since it is simple, effective and should greatly reduce carbon attrition and therefore carbon and gold loss in fine carbon.

3. Ore grading between .01 and .04 opt Au is now trucked to new heap leach pads. These pads are constructed with a bottom layer of 12 - 16 inches clay with eight inches of two-inch gravel laid on top. They are experimenting with truck built and front-end loader built heaps to see if there is a difference. The ore is run-of-mine (minus 24-inch) with no crushing. Each pad holds approximately 90,000 tons stacked 20 feet high. Leach time is 90 days with a planned once yearly releach of 45 days.

4. Senninger Wobblers are used for solution distribution instead of rainbirds or Bagdad wigglers since they feel that the Senninger units give better solution distribution.

5. Heap leach tests gave up to 70% recovery in a 45-50 day leach period. They are using a figure of 60% recovery for design purposes. Leach solution is 1 lb/ton NaCN and pH is maintained at 11.0.

6. Tests indicated that scaling would be a big problem and they use a Drew Chemical Co. antiscalent called Milsperse 802.

7. Heap leach pads are sloped to collection ditches at the sides and in these is laid perforated pipe covered with gravel for solution flow. Hypalon is used to line the ditches. 40 mil HDPE is used for solution pond lining.

8. Pinson ran a fairly comprehensive heap leach test using a 115 ft. square asphalt pad. Approximately 4,100 tons of ore could be placed on this pad stacked up to 15 ft. high. Senninger Wobblers were used for distribution on top of the heap and Bagdad wigglers were used on the side slopes. They stressed that with small test heaps you must pay special attention to the side slopes because they will contain a significant proportion of the values.

9. Pregnant solution from the test pad was passed through three 14-inch diameter by six-foot high carbon columns each containing 75 lbs. of 6 x 11 carbon. Barren solution flowed by gravity to a collection pond from which it was pumped back to the test pad. The optimum solution application rate was 0.0025 gpm/ft^2.

10. Wind and evaporation loss of solution are significant, particularly at the test heap. Wind loss represents an actual, unaccountable gold loss if the barren solution contains significant amounts of gold.

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THE PINSON GOLD STORY

by

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T.D.H.
MAY 5 1983

Presented at Annual Meeting of the Arizona Conference of AIME
in Tucson, Arizona - December 6, 1982

MINE

A well-oxidized hydrothermal deposit forms the Pinson orebody. Gold values are finely disseminated as micron-sized particles in a silicified matrix (locally called a jasperoid). Higher gold values are usually associated with a redder jasperoid. This jasperoid is extremely hard with a Bond Index approaching 20; overall, the orebody has a Bond Index of 17. Silver values are minimal and sulfides are absent. There is a variable amount of mercury in the Pinson deposit with some values being 10 times the gold content. There is virtually no preg-robbing carbonaceous material in Pinson ore.

The Pinson Mine is located in the Osgood Mountains at an elevation of 4900 ft. above sea level. By road, Golconda is 22 miles to the southwest and Winnemucca is 38 miles to the southwest of the Mine. The Pinson property is 4 miles south of the historic Getchell Mine.

Three ore zones (A, B, and C) have been identified and drilled out at Pinson. The A Zone consists of 1,466,000 tons at an average grade of 0.144 oz./ton; B-Zone contains 1,660,000 tons averaging 0.068 oz./ton; and the C Zone has 139,000 tons averaging 0.133 oz/ton. An additional 5,500,000 tons of heap leach grade material will be mined with the ore. Metallurgical tests are still being conducted to determine the most profitable way to treat the heap leach ore.

Mining at Pinson started in May, 1980. Around 2,400,000 tons of waste and 100,000 tons of ore were mined before mill startup. A high grade section of the A Zone was mined the first year to pay off a 15.75 million dollar bank loan as quickly as possible. The bank loan with 2.5 million dollars in interest was payed in full 15 months later.

The A and B Zone pits are mined in two stages. Overall pit slopes are 55^{0} in the hanging wall limestone and 45^{0} in the footwall shale. The hanging wall has 33 ft. safety berms every 80 vertical feet while the footwall safety berms are 25 ft. wide and are left every 60 feet. Width of the present pushback is about 250 ft. This will narrow as it goes down from the 5340 ft. elevation to the 4880 ft. level which is the pit bottom.

Primary drilling at Pinson is done with Reed equipment. Two truck mounted SK-35's and a Crawler mounted SK-40 drill 6.75 inch diameter holes 24 ft. deep in a pattern that ranges from 12 x 12 ft. to 16 x 16 ft. depending on the hardness of the ore. Cuttings from the holes are assayed for grade control. Blasting is done with ANFO to shake up the ore to improve subsequent ore loading operations.

Ore is loaded by a Caterpillar 992 front-end loader equipped with a 12 cubic yard bucket or a Liebherr hydraulic shovel employing a 5½ cubic yard dipper. The haulage fleet consists of four Wabco 50 ton trucks and two 50 ton Caterpillar units. The ore is hauled nearly one mile to a stockpile at the mill and waste is hauled two-thirds of a mile to dumps situated south and east of the pit. Leach grade material is stored in separate stockpiles for future treatment.

MILL

Industrial Design Corporation was contracted for detailed design of the mill complex with the conceptual flowsheet provided by Pinson. Design criteria was supplied by consultants, laboratory metallurgical data, and Pinson management after the review of other operating CIP and carbon absorption circuits in the United States and South Africa. The mill circuit consists of two stage ball mill grinding with carbon absorption columns for pregnant thickener overflow solution and a CIP circuit following the agitated leach of the thickener underflow pulp. Pinson was the first gold operation to use carbon columns and CIPs to recover values from solution. The Pinson mill was operational fourteen months after the beginning of design engineering.

Ore from the coarse ore stockpile is fed to the crushing plant hopper by a Cat 980 or 966 loader. A hydrostroke feeder moves the rock to a 42" x 48" Allis-Chalmers jaw crusher which discharges to a 36" primary belt. The material is passed on a 7' x 16' double deck vibrating screen with the over size going to the 5½' Symonds cone crusher and the under size going to the 1700 ton capacity fine ore bin. The mill feed is minus 1/2".

Mill feed is withdrawn from the fine ore bin by two long slot belt feeders. Dry cyanide and pebble lime are fed on the mill feed belt with the ore to the primary Kennedy Van Saun (KVS) 9' x 13' ball mill with 500 connected horsepower. An Allis-Chalmers (AC) 12' x 14' ball mill with 1000 connected horsepower provides the finishing grind. Both mills discharge into a common sump feeding a bank of four Krebs 15" cyclones for classification; cyclone underflow is returned to the AC mill and the overflow passes through a DSM screen before going to the thickener. The grind at Pinson is held around 10% plus 200 Tyler mesh as coarser grinds do not provide sufficient liberation for acceptable leach extractions.

The cyclone overflow is thickened in a 100' diameter uncovered flat bottomed thickener with a conventional Eimco mechanism. Flocculent is added to the pulp in the center well. The pulp is thickened from 15% to 40% solids. The thickener overflow is pumped to a headbox feeding five 4' x 12' upflow carbon absorption columns. The columns each contain 1 ton of 6 x 12 mesh activated coconut shell carbon which absorb the gold cyanide complex from solution. The barren solution from the columns goes to the grinding sump. The carbon in the columns is advanced counter-current every other day by water eduction.

The thickener underflow is leached in four steel tanks located outdoors. The leach tanks are equipped with 60 horsepower Lightnin agitator drives and low pressure air is injected below the lower agitator blades. From the leach tanks the pulp is fed into the mill where the gold is recovered in five 11' x 23' carbon-in-pulp (CIP) tanks. The CIP tanks are fitted with 20 mesh air swept screens and 7.5 horsepower Lightnin agitator drives with draft tubes. Each CIP tank contains 1 to 1½ tons of 6 x 12 mesh carbon. The carbon is advanced counter-current every other day using a recessed impeller centrifugal pump. Discharge from the CIP circuit is screened on a Derrick with a 28 mesh deck, to recover fine carbon and spills, and then pumped from the tails box to a clay lined tailings dam.

The advanced carbon assays approximately 175 oz/T. Stripping is done batchwise on 3 ton charges of carbon in 42" x 24' insulated steel tanks. The strip solution is 1.5% NaOH -0.15% NaCN heated to 195°F. The solution circulates from the electrolyte heating tank to the strip tank. It is then pumped through the electrolytic cell and back to the electrolyte heating tank. The strip is performed at atmospheric pressure and takes about 3 days to complete. Recovery of gold from the strip solution is accomplished in an 80 cubic foot electrowinning cell. The cell contains 13 stainless steel wire mesh anodes and 12 steel wool cathodes supported in plastic holders. The gold, silver, and mercury are plated from solution using 3½ volts and 550 amps.

The stripped carbon assays approximately 6 oz./T. and is water educted to the kiln hopper at the end of the strip. After dewatering in the hopper, the carbon is run through a 3' x 20' propane fired rotary kiln at 650° C for reactivation. The reactivated carbon is used as make-up for either the columns or CIPs.

Reagent consumptions at Pinson are very reasonable. The following table shows reagent consumption through October 1982:

Reagent	Consumption (lbs./ton milled)
Cyanide	0.36
Lime	2.7
Carbon	0.08
Caustic	0.08
Flocculant	0.02
Balls	2.6

HEAP LEACH

A significant amount of gold is contained in mined material that is too low grade for the mill circuit and yet too high for waste dump disposal. Pinson has run a series of test heap leaches to determine if any of the gold value in this material may be recovered economically. Results from the tests run to date range from 65% to 80% recovery of the total gold in the heaps.

The heap leach test ore is stacked with a Cat 992 loader on a 113' x 115' asphalt pad. An asphalt pad is used so that more than one test may be done in an area; a clay pad or plastic lined pad would be destroyed by the unloading process. To date, all of the test ore has been run-of-mine (ROM); this means material ranging in size from 24" to clay. The test heaps contain approximately 4100 tons of ore stacked 13 to 15 feet high by the loader. A Cat D6 is used to level the top of the heap to aid the subsequent laying of pipes and sprinklers.

Pinson uses Senninger wobblers to distribute the spray solution over the test heap and plans to use them on the production heaps which will be around 90,000 tons of ore. The wobblers give a more complete coverage and require less maintenance than rainbirds or wigglers. Spray solution is applied at a rate of 0.0025 gpm/ft.^{2} Higher flow rates cause ponding and runoff; lower flow rates do not adequately cover the heap.

The pregnant solution from the heap flows by gravity to a collection pipe and then to the preg pond. The pregnant leach liquor is pumped through three 14" x 6' upflow carbon absorption columns. Each column contains 75 lbs. of 6 x 16 mesh activated carbon. The barren solution from the columns flows by gravity to the barren pond and then is pumped back on the heap for the cycle to start again. NaOH, NaCN, and an antiscalant solution are added to the barren solution at specific times throughout the test. Both ponds for the test facility are lined with 30 mil CPE.

The ponds for the production heaps will be lined with 40 mil HDPE. The preg solution will be pumped to the mill where it will pass through five 5' x 12' upflow carbon absorption columns. Each column will contain 1 ton of 6 x 12 mesh activated carbon. Reagents for the production heaps will be added in the mill to the column barren solution before it is returned to the barren pond.

The production heaps will be built on clay pads since they will not be removed at the end of leaching. Present plans for the production heaps call for 93,000 tons of ore per heap. Each heap will initially be leached for 90 days. One year after the initial leach a 45 day releach will occur. These releach cycles will continue as long as economics permit. At peak production 2 new heaps and 1 old heap will be leached at a time.

There are also plans to run one old heap counter-current in addition to the other heaps. If the pregnant off solution from the heap is too low grade ($<$0.005 oz./ton) to send to the mill circuit for absorption the counter-current system will be used. The preg solution is returned to the barren pond to enhance the barren solution that is sprayed on the heaps. This way the pregnant solution fed to the columns is higher than it would be if the off solution from the counter-current heap went directly to the preg pond.

The type of ore that is to be leached and the way the ore is loaded on the leach pad are important in gold heap leaching. Crushing, agglomerating, or a combination of both may be required to prepare the ore if it has a low permeability or a high clay content. The ore should be stacked on the pad in a way which will minimize compaction. Loader or conveyor stacking are preferential to truck dumping, but they are more expensive.

The heap leach tests completed at Pinson have indicated several problems which will occur in operating a gold heap leach. The most serious of the problems is the formation of scale in the system. The scale build-up causes sprinklers to stick, flowmeters to give inaccurate readings, carbon to foul, and the leaching process will slow or stop completely. Acid washing will fix the sprinklers, carbon, and flowmeters. Blasting is the only economical way to break up scale in a large gold heap. At Pinson we use reclaimed mill water and an antiscalant compound to overcome the scaling problem; acid washing of sprinklers and flowmeters is usually only required at the end of the test.

Wind and evaporation losses of the leach solution are problems that have not been economically solved. Wind losses at the test heap were magnified due to the high amount of side slope area to the total area. The wind losses in our test heaps were actually gold losses since the columns at the test site would never take all the gold out of solution; this is probably true for most small gold heap leach operations. Larger heaps will most likely have more efficient gold absorption circuits. Evaporation losses in the test heaps at Pinson were not a major problem since make-up water was readily available. Some heap leach operations must truck in the water for leaching which adds significantly to their operating costs. The wind and evaporation losses experienced in the test heaps at Pinson ran as high as 15% of the solution sprayed.

Cold weather is the latest operation's setback experienced at Pinson. The leach solution freezes on the heap causing an effect like scaling, i.e. the cyanide solution is kept from reaching the gold by a layer of ice in the heap. Most heap leach operations close down during the winter because of this problem. I have heard of three ways that heap leach operations have run through the winter. Some operations heat the solution before it is sprayed to the pile; this causes a tremendous scaling problem. One operation added ethylene glycol to the leach solution. The solution didn't freeze and the gold leached well, but the carbon in the columns would not absorb the gold from solution; the ethylene glycol fouled the carbon. The best method buries perforated lengths of pipe in the top of the heap. A hose is run from the sprinkler connection down into the pipe which is buried deep enough not to freeze. The drawbacks of this method are channeling and the top portion of the heap is not being leached, but this seems to be the best way if an operation must operate in the winter months.

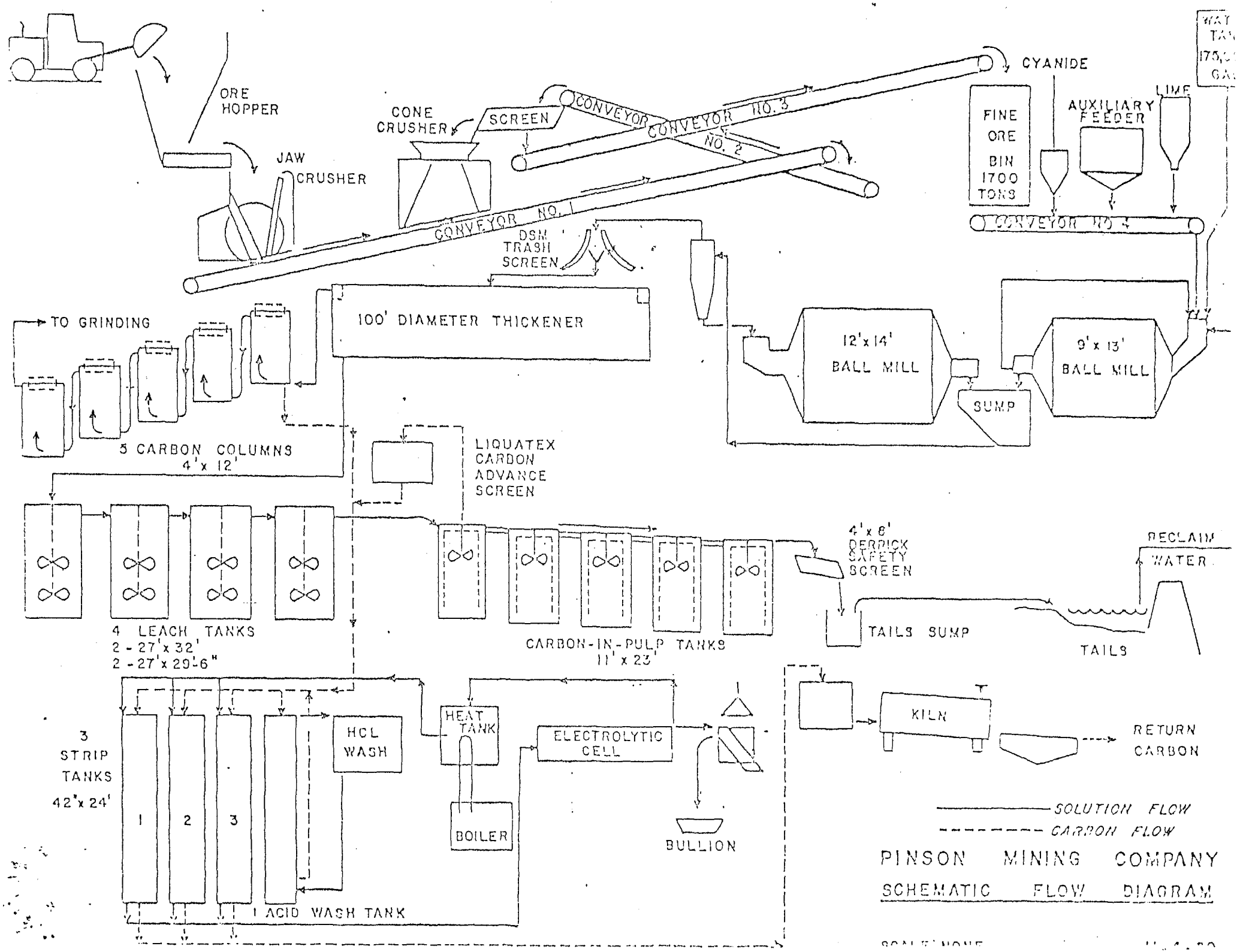
REFINERY

The cathodes for the electrolytic cell are 1½ to 2 pounds of steel wool sandwiching a three prong stainless steel bayonet enclosed in a plastic holder which measures 22" x 28" x 2". The average cathode loading is 100 oz. of gold, 1 oz. of silver, and 2 lbs. of mercury. Freshly pulled cathodes are removed from the plastic holder and placed immediately into the stainless steel retort pot or a plastic holding tank. The retort pot is surrounded by a coil which heats the pot by induction to a 700° C wall temperature. The retort is run under vacuum conditions to prevent stray emissions from leaking into the working area due to faulty seals. All of the vaporized water and mercury is pulled over into a stainless steel spiral condensor where plant water cools the vapors. The condensed water and mercury are held in a holding pot fitted with a drain valve. The recovered mercury is +99.95% pure. At the end of the heating period, an air valve is opened to permit oxidation of the steel wool which aids in the subsequent induction refining; this is done while retaining vacuum conditions. An oilless vacuum pump is used to keep the entire system under negative pressure. The vacuum pump discharge is fitted to a scrubber suction line.

The flux used at Pinson for refining is a mixture of borax, niter, and silica for steel wool cathodes; borax and niter for electrolytic cell sludge. A 500 lb. brass capacity tilting induction furnace is used to smelt the cathodes and sludge. A 100 lb. brass capacity tilting induction furnace is used to pour the final bar. The final product typically runs 970 gold fine and 10 silver fine. The remaining 20 parts are mostly nickel, copper, and iron.

The first charge in the big furnace consists of two retorted cathodes with fluxing agents. The power supply is set at 35 KW, 300 VAC, and 2700 Hz. In one hour the charge is melted and a single retorted cathode with flux is added; thirty minutes later the first rough slab is ready to be poured. The power supply is turned down, the hydraulic pump system is activated, and most of the slag is poured off into a conical mold. The conical mold is wheeled out and a bar mold is used for the gold and remaining slag. The above process is repeated until all of the cathodes are refined, but it only takes 45 minutes to smelt a two cathode charge. The most gold refined in one shift at Pinson was 17 cathodes containing over 2450 troy ounces.

The rough slabs poured from the big furnace are cleaned, weighed, and stored in the vault until there is enough gold to pour a bar. It only takes 20 minutes to melt 1000 to 1200 oz. of gold and 5 lbs. of borax in the small furnace at 30 KW.


ORE HOPPER
JAW CRUSHER
CONE CRUSHER
SCREEN
CONVEYOR NO. 3
CONVEYOR NO. 2
CONVEYOR NO. 1
CYANIDE
FINE ORE BIN 1700 TONS
AUXILIARY FEEDER
LIME
CONVEYOR NO. 4
DSM TRASH SCREEN
100' DIAMETER THICKENER
TO GRINDING
12'x14' BALL MILL
9'x13' BALL MILL
SUMP
5 CARBON COLUMNS 4'x12'
LIQUATEX CARBON ADVANCE SCREEN
4'x8' DERRICK SAFETY SCREEN
RECLAIM WATER
4 LEACH TANKS
2 - 27'x32'
2 - 27'x29'-6"
CARBON-IN-PULP TANKS
11'x23'
TAILS SUMP
TAILS
3 STRIP TANKS 42"x24'
1
2
3
HCL WASH
HEAT TANK
BOILER
ELECTROLYTIC CELL
BULLION
KILN
RETURN CARBON
1 ACID WASH TANK
SOLUTION FLOW
CARBON FLOW
PINSON MINING COMPANY
SCHEMATIC FLOW DIAGRAM

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PINSON MINING COMPANY - MILL DESIGN

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PINSON MINING COMPANY - MILL DESIGN

Pinson Mining Company was formed to develop a gold orebody located in northern Nevada near Winnemucca. A feasibility study was undertaken in 1979 by a consortium of three Toronto-based mining companies and a Reno-based exploration company. The study indicated a viable project with gold priced above $225 per ounce.

A project manager, and mine and mill superintendents, were hired in late 1979 to expedite the development of a 1000 ton per day mine-mill complex. Plant design engineering was contracted with Industrial Design Corporation in Salt Lake City. Three experienced plant metallurgists representing Pinson were stationed in Salt Lake to guide the work of Industrial Design. Among the many design criteria spelled out by Pinson, foremost were bringing the mill on line as quickly as possible at minimum capital cost. Fourteen months after the beginning of design engineering, the completed Pinson mill became operational. Total capital cost, including mine development, was $18 million. The Pinson project has proved to be far more than viable; the entire capital cost will be recovered within 18 months of the mill start-up.

The Pinson plant has been described in some detail by Mcquiston and Shoemaker in the AIME's Gold and Silver Cyanidation Plant Practices. I will review the flowsheet only briefly, highlighting significant features. The emphasis of this presentation will be on specifics of the design.

The Pinson ore is a well-oxidized hydrothermal deposit, with gold finely disseminated as micron-sized particles. The limestone-siltstone host rock has been silicified to a jasperoid, which has weathered in part to clay. The remaining jasperoid is extremely hard. Overall, the orebody has indicated a Bond work index of about 17 to date.

The crushing plant flowsheet is illustrated. It is a conventional two-stage plant, producing a minus 5/8 inch product. The most significant feature of the design, from a mill superintendent's viewpoint, was the provision of a belt plow and outdoor crushed-ore stockpile. This stockpile of about 20,000 tons is used to maintain steady mill feed during crushing plant maintenance shutdowns. The crushed ore can be loader-fed to the mill through an auxiliary Kolberg feeder which bypasses the crushing plant and fine ore bin. This system has provided tremendous availability of mill feed, holding downtime due to lack of ore to insignificant amounts, in fact to 0.01% of available time for the 13 months since start-up.

The grinding circuit is illustrated schematically also. The Pinson ore is extremely clean from a leaching standpoint. There are no sulfides or carbonaceous materials to consume reagents or rob dissolved gold. This fact allows the leaching reagents to be fed dry onto the mill feed belt with the crushed ore, eliminating the capital and operating cost of a reagent mix system. Cyanide briquettes are

added just outside the fine ore bin, at 0.25 pounds per ton. Bulk pebble lime is added at 2.5 pounds per ton. The Kolberg auxiliary feeder is also located above the mill feed belt.

The 500 hp Kennedy Van Saun primary mill uses 4" balls at 0.7 pounds per ton. It discharges through a 1/4 inch trommel screen into a sump common to both mills. Primary trommel oversize at 2 to 3 tons per hour is recycled via a cleated belt. Mill discharge slurry is pumped to four Krebs 15 inch cyclones. Cyclone underflow feeds the 1000 hp Allis Chalmers secondary ball mill with a 200% circulating load. The AC mill uses 1½" balls at 2.2 pounds per ton.

The cyclones are set to yield a 90% minus 200 mesh overflow. This fine grind is necessary to leach the micron-size disseminated gold particles to 85% recovery. Cyclone overflow at 15% solids passes through a Denver slurry sampler and then through a DSM screen to remove tramp oversize. DSM screen undersize flows by gravity to a 100 foot diameter Eimco thickener. The thickener was sized by bench testing samples from development drillholes. The testwork indicated an 80 foot thickener would be adequate. As the cost difference between an 80 foot and 100 foot thickener was small, in view of the high clay content and the potential for difficult thickening, a 100 foot thickener was purchased.

Cyanide leaching is rapid in the grinding circuit. Following the six hours retention in the thickener, about 80% of the gold is in solution.

Beyond the thickener, the circuit splits. Pinson was the first carbon-in-pulp operation to adopt a split-circuit approach in the absorption section. The thickener overflow is pumped to a headbox feeding a series of five fluidized-bed carbon columns. The design of these columns will be discussed later. The activated carbon in the columns absorbs the gold from solution. The barren solution then recycles to the grinding sump.

Thickener underflow, held at 40% solids due to highly viscous clays, is leached for 22 hours in four steel tanks equipped with Lightnin agitators. Low pressure air is injected below the lower agitator blades.

The pulp then flows by gravity to a five-stage carbon-in-pulp circuit. The CIP tanks are unique in North America. They contain Lightnin-designed draft tube agitators and were selected for their low horsepower requirements. They provide very gentle agitation which nevertheless is adequate for suspending the slurry particles. In the carbon-in-pulp circuit, we sought to minimize power input in order to minimize carbon attrition. Although carbon fines lost to attrition have a small replacement cost, such carbon fines carry absorbed gold to tails with them, and are therefore of great concern.

The pulp flows through the five CIP tanks by gravity, but the carbon must be held back in the appropriate tank. Many CIP circuits achieve this by air-lifting the pulp onto mechanical vibrating screens between each stage. This inter-stage air lifting

and screening was frequently identified as a serious source of carbon attrition losses. To avoid these losses, the Pinson design team adopted a new approach that utilizes a transverse launder for pulp flow through the five CIP tanks. The side walls of the launders are fitted with 20 mesh static screens which prevent the carbon from flowing into the launder with the pulp. Air bubblers beneath the static screens sweep the carbon away from the screen surface. Details on the static screen launder design will follow.

Before dropping into the tails pumpbox, the pulp discharging from No. 5 CIP tank is screened on a Derrick screen at 28 mesh. This last step in the continuous-flow part of the mill is designed to catch carbon fines which may have escaped the launder screens.

The tailings pond has no discharge. Solution is reclaimed and pumped to the grinding circuit. The contained cyanide and lime contribute greatly to our very low reagent consumption.

The remaining operations in the Pinson mill are done on a batch basis. The carbon in both of the absorption circuits is loaded to about 200 ounces gold per ton of carbon. When so loaded, the carbon is moved by eductor or pump into a stripping vessel.

Loaded carbon is stripped in 3 ton batches using 1.5% caustic and 0.15% cyanide at 195°F under ambient pressure. At completion, after approximately 72 hours, the carbon has been stripped to 5 ounces per ton.

The pregnant strip solution flows by gravity to the refinery where gold is electroplated on 14 cathodes. Two pounds of mild steel wool contained in a plastic basket make up the cathode. Stainless steel wire mesh is used for the anode. The cathodes are normally loaded to about 80 ounces per pound of wool before they are pulled for refining. The barren strip solution is reheated and recirculated to strip more loaded carbon.

The loaded cathodes are retorted in an induction-heated vessel at 650°C under a 15 inch Hg vacuum to recover mercury that has plated with the gold. Approximately 2 pounds of mercury are recovered per cathode. The retorted cathodes are then charged into a 125 kw induction furnace with appropriate fluxing agents for fire refining. Graphite-bonded silicon carbide crucibles are used as seceptors. Normally 14 cathodes containing about 2000 ounces doré can be refined in one shift. A smaller induction furnace is used to remelt buttons into 1000 ounce bars for shipment. The doré typically runs 965 Au fineness and 10 Ag fineness.

DESIGN CRITERIA

Carbon Columns

Empirical data was available to us from columns built and operated at Cortez Gold Mine. However, our design flowrates were significantly different and we intended to use carbon of a larger mesh size than Cortez. For this reason, bench testwork was done at Dawson Metallurgical Lab in Salt Lake to quantify the required upflow rate with the size carbon we intended to use.

The critical parameter in designing fluidized bed carbon columns is gallons per minute per square foot of cross-sectional area. Bench tests indicated the following:

	50% bed expansion	100% bed expansion
6 X 16 mesh carbon	33.0 gpm/sq.ft.	45.5 gpm/sq.ft.

This testwork data was combined with empirical data from Cortez. Using 12 X 30 mesh carbon, their columns, with 19.6 square feet of cross-section, would handle a maximum of 375 gallons per minute. This data relates to the 100% expansion values and calculates to 19 gpm per square foot.

The cross-sectional area of the Pinson columns was sized for 50% bed expansion, to allow for future throughput increases. The height of the column was selected for 100% bed expansion plus two feet of freeboard, again to allow for throughput increases.

Empirical data is now available from Pinson also, indicating the maximum throughput rate manageable to be 460 gpm, or 36.6 gpm per square foot. Obviously, the bench testwork can provide a useful design guide, but must be interpreted with some caution. It is apparent that the carbon size must be selected and accurately known before columns can be designed.

Recent experience at a neighboring Nevada gold mine has highlighted the danger in over-sizing carbon columns. For gold recovery from solution, one of the greatest merits of the columns versus zinc precipitation is the fact that the solution need not be clarified for columns. In fact, up to 3% solids slurries have been successfully run through carbon columns. However, if designers, hedging their bets, over-size the cross-sectional area of a carbon column, the bed of carbon will not be fluidized by the available upflow rate. If this happens, the bed will act more like a sand filter and become plugged off with entrained solids or slimes, or with scale formed from the high lime content common in gold cyanide leach solutions.

For practical operating efficiency, the solution flowrate to the columns should be kept within the range for 50 to 100% bed expansion.

CIP Static Screen Launders

At the time of the Pinson design effort, there were no commercial-scale static screens in carbon-in-pulp circuits to turn to for empirical design data. The concept was being developed by Dr. Peter Laxen at the National Institute of Mining in South Africa. He had tested a pilot scale carbon-in-pulp tank with a static screen, using a submerged air lance to sweep the screen clear of carbon. A brief trip to South Africa provided rough design guidelines.

It appeared that the critical parameter was not square feet of screen area, but rather linear feet of screen, measured parallel to the pulp surface. Observations of the system at Pinson have since confirmed this, although we had some misgivings about it in the design phase. The rational is illustrated in this sketch.

On average, only about 3/4 inch of the screen's height is active. The upper portion is obviously not in use, being above the pulp level. The lower portion is blinded off, with carbon particles primarily, but also with wood chips, truck tire bits, and lime scale. The air bubblers continually sweep this build-up, however near-size particles gradually blind off the screen from the bottom up. Typically, our screens need thorough cleaning every two weeks.

The value we arrived at in the design phase was 16 gpm per lineal foot of screen. Actual plant experience has indicated this flow is achieveable. However, more screen length will make the system easier to operate, being less sensitive to the natural blinding-off tendency. I would recommend designing for only 13 gpm per lineal foot of screen, assuming the use of 6 X 16 mesh carbon.

As always, the greater the size difference between the desired oversize and the desired undersize, the easier will be the screening operation. Therefore, we favor the use of 6 X 16 mesh carbon or thereabouts. Its absorption capacity is not as great as smaller carbon due to its smaller surface area, but it is much easier to operate with the larger carbon. At Pinson, we are particularly fortunate in the CIP launder screen operation, in having a pulp ground to 90% minus 200 mesh. The size difference between ore particles and carbon is great, and the screen split is easy.

Mill Layout and Operational Control

The Pinson mill was laid out with two main objectives; to minimize manpower required to operate it, and to provide for future expansions in throughput. This sketch illustrates the plan view layout of the mill. A five ton overhead crane runs throughout the length of the mill. The entire milling operation, from fine ore bin through to tailings pond and reclaim water system, was designed to be operated by two men per shift. Control and monitoring of the operation is centralized at the control panel. The operators can tour the plant quite efficiently, following the three routes illustrated. During these tours, process variable measurements are taken, to be recorded on a shift log back at the control panel. Whenever both operators are needed at one location, the control panel will continue to monitor the operation. If a process variable ranges beyond specified limits, an audible alarm sounds, alerting the operator to return to the panel.

The front of the control panel is illustrated here. The graphic displays the entire flowsheet. Green lights indicate operating equipment motors. When an alarm sounds, a red light will flash on the graphic identifying the source of the trouble. The operator pushes the acknowledge button to silence the alarm horn, and the red light changes from flashing to steady red. The red light will stay on until the problem has been corrected.

The control panel also contains process monitoring and control equipment. Digital displays on the upper right side monitor critical variables continuously. These are mainly flowmeter signals, assisting the operator in maximizing throughput. On the left side, five digital displays are each connected to ten-position selector switches. The operator can select variables to monitor as he chooses. Three of the displays are for motor amperages, one is for levels, and one is for temperatures and pressures. On the lower right side of the panel are two three-pen recorders. These recorders will accept any six signals from among the fifty selector-switch process variables or the five flows displayed above the recorders.

The Pinson mill is not automated. All decisions remain with the operator. The four existing controllers simply assist him in continuously maintaining four critical flows at the set-point he selects. The control panel alarm system permits him to roam freely throughout the plant watching for developing mechanical or process flow problems, while providing continuous overall monitoring of the operation. No "control room operator" is required with this system.

The Pinson mill was built on the basis of ten years proven reserves. The potential for expanding these reserves is good however. Careful attention was given to making the plant easily expandable in future to higher tonnage rates with minimal disruption of production. Provisions were also made for future toll-milling of ores.

At the crushing plant, cost analyses indicated little capital cost sensitivity to design throughput rate. Consequently, future tonnage increases were allowed for by sizing the crushing plant to supply the initial Pinson mill requirements on a one shift per day basis. Simply adding operators to crush three shifts per day will increase crushed ore production to 2000 tons per day, allowing for maintenance downtime.

Unlike the crushing plant, the milling circuit was very cost-sensitive to tonnage rate. In view of prevailing 21% interest financing rates for the project, our guidelines were very clear to construct only for the proven ten years reserves. However, without adding to initial capital cost, provisions were made in the design layout and equipment selection stages for future tonnage increases to 2000 tons per day. Additional equipment expenditures would be needed of course, but the plant layout would permit installation of this equipment with little disruption of ongoing production.

The addition of a third ball mill of about 1000 hp was anticipated on scale drawings in the space between the No. 2 ball mill and the thickener. Anticipating an imbalance of primary versus secondary milling capacity at 2000 tons per day, two scenarios were provided for increasing primary mill throughput.

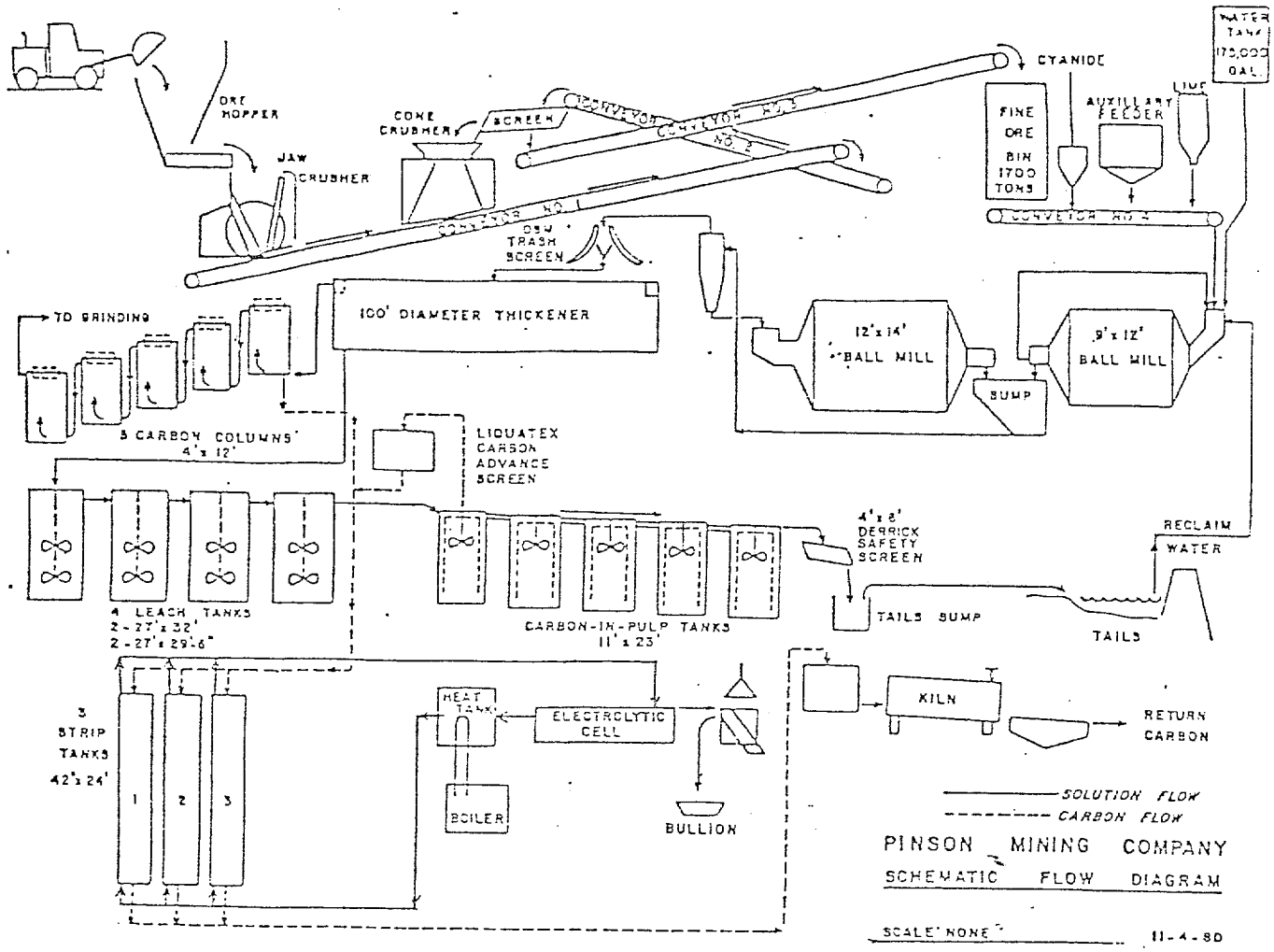
The first would convert the 500 hp KVS primary ball mill to a rod mill. Structural steel in this area was oversized sufficiently to support a rod charger and rod storage racks. Conversion to a rod mill would increase the particle size of material leaving the primary mill, shifting horsepower requirements to an expanded secondary milling stage.

The second scenario for balancing primary/secondary milling capacity would shift the primary mill feed to the 1000 hp AC ball mill, by extending No. 4 conveyor 20 feet. After addition of the third mill, this second approach would provide 1000 hp for primary milling and 1500 hp for secondary milling.

Beyond the grinding circuit, higher throughput requirements were less predictable, since no testwork was available to identify which unit processes would become bottlenecks on future ores. Again, potential expansions were layed out on scale drawings. Additional thickener capacity was anticipated just east of the existing thickener. Additional leaching capacity was provided for by extending the leach section eastward. Outlets for these future tanks were inserted during initial construction.

Inside the plant, all equipment in the carbon stripping/regenerating section was placed on break-away pads, without pinning to the underlying mill floor. This large passageway , although justified initially as truck access into the mill for construction, in reality is our space provided for future CIP system expansion. CIP retention time could be doubled by addition of a second train of five tanks.

Although allowances for expansion were made in the design phase, no equipment additions are yet scheduled. When word comes from the exploration geologists that reserves have been expanded, the Pinson mill will be prepared to react positively.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JAW CRUSHER
CONE CRUSHER
SCREEN
CONVEYOR NO 3
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CONVEYOR NO 1
DSM TRASH SCREEN
CYANIDE
WATER TANK 175,000 GAL
FINE ORE BIN 1700 TONS
AUXILLARY FEEDER
LIME
CONVEYOR NO 4
TO GRINDING
100' DIAMETER THICKENER
12'x14' BALL MILL
SUMP
9'x12' BALL MILL
5 CARBON COLUMNS 4'x12'
LIQUATEX CARBON ADVANCE SCREEN
4'x8' DERRICK SAFETY SCREEN
RECLAIM WATER
4 LEACH TANKS 2-27'x32' 2-27'x29'-6"
CARBON-IN-PULP TANKS 11'x23'
TAILS SUMP
TAILS
3 STRIP TANKS 42'x24'
HEAT TANK
ELECTROLYTIC CELL
BOILER
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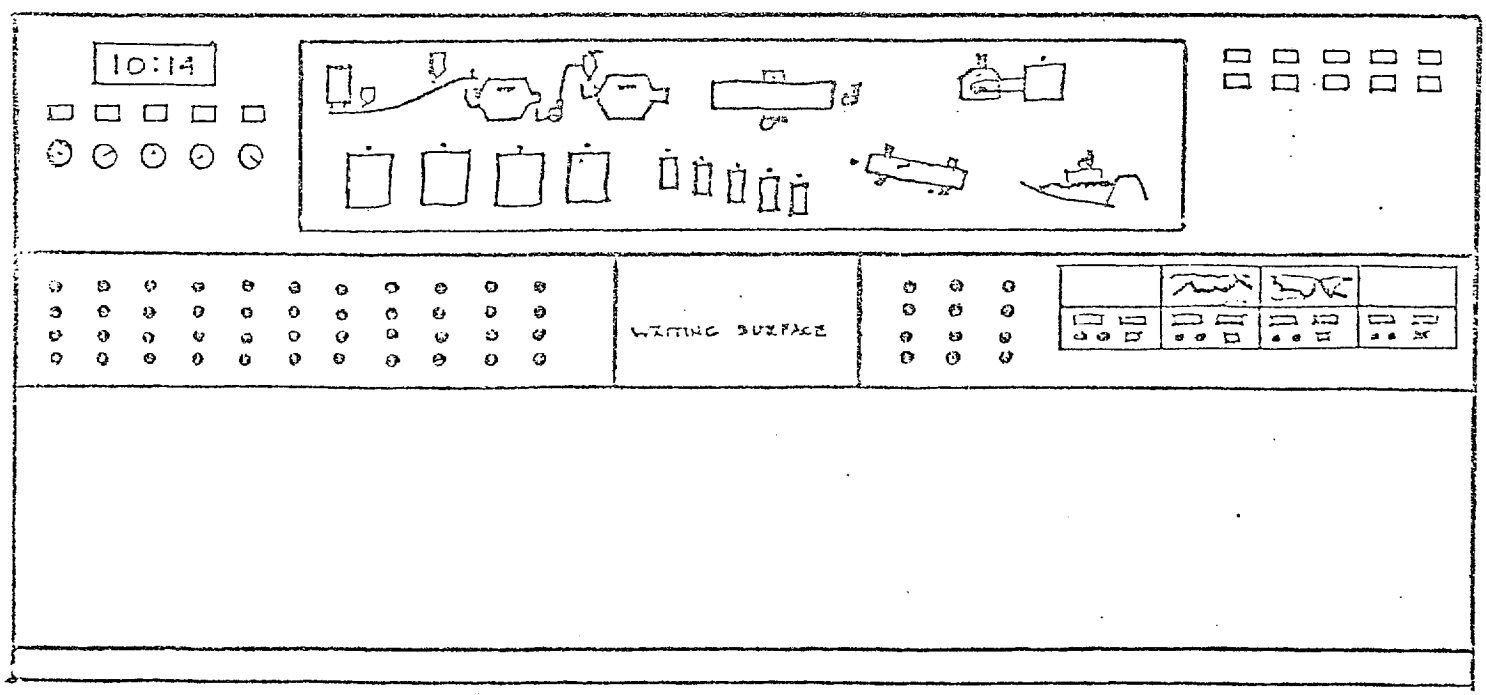
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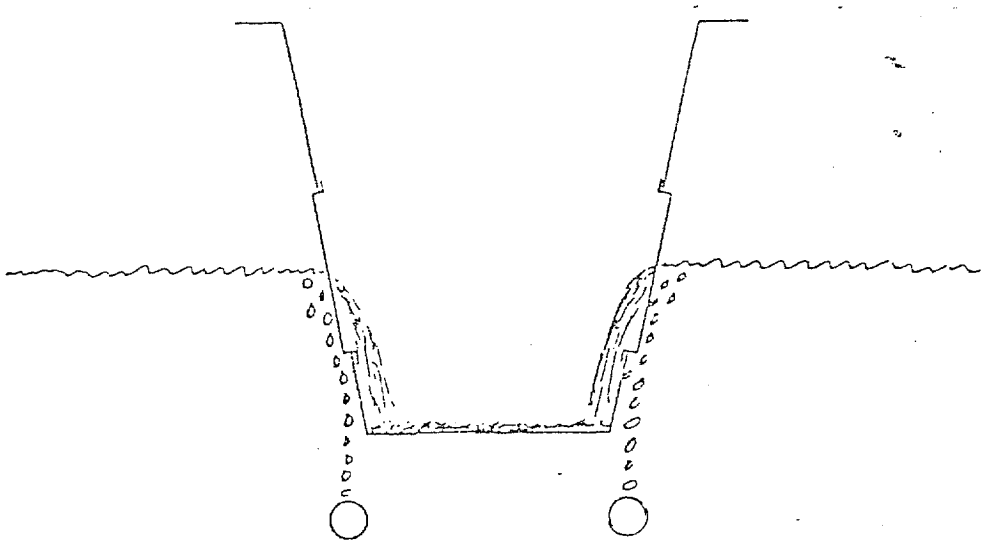
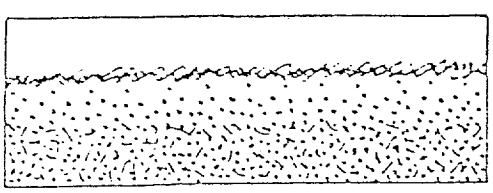
WRITING SURFACE

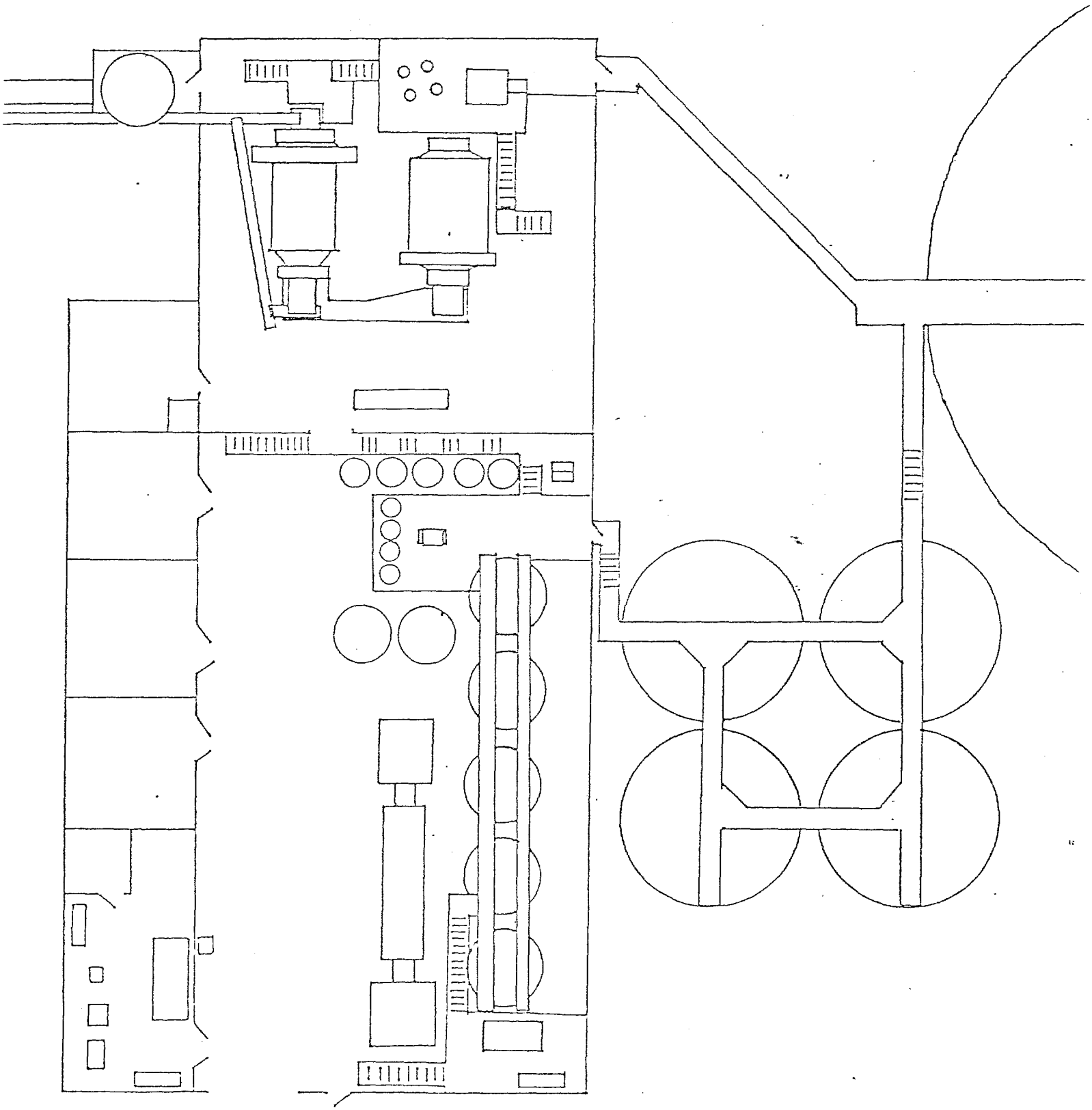
FLUIDIZED-CARBON BED FLOWRATES

(FLOWS IN GPM/SQ FT)

DATA	CARBON	BED EXPANSION 50%	BED EXPANSION 100%
TESTWORK	6x16 mesh	33.0	45.5
CORTEZ	12x30 mesh		19.0
PINSON	6x16 mesh		36.6







Alligator Ridge

The Alligator Ridge operation of Amselco Minerals, Inc. was visited on Tuesday, June 14, 1983. Mr. Ralph Maes, Process Foreman conducted our tour and we talked with Mr. Rich Womack, Process Superintendent.

The ore, a silicified siltstone, comes from three major pits and in the future, several minor areas. Ore contains about 0.1 opt leachable gold. Overall stripping ratio is about 3 to 1. A high proportion of the ore contains a considerable amount of carbonaceous material.

The plant was engineered by Bechtel Corp. Construction was started in May of 1980, and the first gold was poured in October of 1980. Full production was obtained early in 1981. Since the designed crushing equipment required considerable lead time, a portable crushing plant was installed to hasten start-up.

The ore is delivered from the mine in 30 ton rear dump Wabco trucks and dumped directly into a hopper ahead of a 36" x 48" jaw crusher. Ore is crushed to minus 3/4 inch in closed circuit at about 400 TPH. Production is about 4000 TPD, 4 days per week or approximately 80,000 tons per month. Because of winter shutdown annual production is about 750,000. The mine and crushing plant operate two 10-hour shifts per day, 4 days per week.

The minus 3/4 inch crushed rock is agglomerated using an agglomeration drum with lime (5-6 lb/ton normal ore, up to 10 or more lb/ton with carbonaceous ore) and cyanide solution. Agglomeration moisture is between 6 and 8 percent and is controlled by the operator according to the looks of the material.

The agglomerated material can either be placed in a truck load out bin or into a stockpile. The truck bin did not work very well; so it is dumped into a stockpile. Each morning the material from the stockpile is loaded into trucks using a front end loader and placed on the leach pads.

The leach pads were constructed using approximately 18 inches of compacted silt from local dry lake beds. Four inch schedule 30 PVC pipe with 180° slots are placed on 14 ft. centers are buried 2 inches (1/2 diameter) into the clay seal. These pipes are connected to header pipes which are on two sides of the pad. The pad seal is then overlain with about 12 inches of minus 2 inch plus 3/4 in gravel with about 18 inches of gravel over the slotted pipe. Leach pads are approximately 300 ft. by 600 ft. The pads follow

the general contour of the land after leveling and slope approximately 7 percent in one direction and 5 percent in the other.

Solution losses through the pad are estimated at 2 GPD through an area of approximately 60 acres. This is far different than the stated losses at Candaleria.

The leach heaps will be permanent with an ultimate height of 56 ft. The first lift was 14 ft. high and was placed with truck and dozer. A 20 ft. high ramp was built and the trucks dumped the ore on the ramp. A dozer pushed the material out over the pad. After the heap was built, the top 6 ft. of the ramp was dozed off and the leach dump was ripped and cross ripped.

After this lift was leached the dump was smoothed using a dozer then ripped with a D-9 using a 9 or 10 ft. shank. The second lift and future lifts will be 8 ft. high. The material will be dumped by truck and pushed up to the 8 ft. using a dozer.

Leach pads are in general about 300 by 600 ft. and are separated by a clay berm on the pad seal. The ore in each of the pads is separated during the first lift but the voids are filled in during the second and subsequent lifts.

Leaching is done at an average rate of 1000 gpm. This solution is added to a single dump using 8 gpm wobbler sprayheads on 40 ft. centers. Leaching of a heap is considered complete when the preg solution contains less than 0.015 opt gold. A total of between 2 and 3.5 tons of solution per ton of ore is required to leach the gold from the dump. The pH of the leach solutions are held between 11.5 and 12.0 with the use of lime. After a pad has been leached it is drilled and sampled on 10 ft. centers. Each one ft. increment is assayed separately. Each 300 x 600 x 8 ft. leached pad requires 1800 holes and 14,400 samples and assays.

Gold is recovered by the standard carbon absorption method and using pressure stripping. Absorption tanks are 5 8-ft. by 9-ft. tanks in series. According to Mr. Womack this design is bad. He would rather have cone bottomed tanks with a cone not more than 20 to 30°. Also he feels the tanks should be much higher than the diameter, a ratio of 2 to 3 times would be much better.

Total employment is 151 men with 30 in the mine, 27 in process and 40 in maintenance.

Pilot plant test work prior to start-up consisted of two test heaps, one containing 10,000 tons and the other 5,000.

Ore reserves and plant performance calculations are based on cyanide soluable gold and not on total gold content. To date gold recovery has been between 65 and 70 percent of the cyanide soluable gold contained in the dump has been shipped. They expect this amount to increase to 80 to 85 percent at the end of the plant life.

Escalante: Ranchers Exploration & Development Co.

The Escalante operation near Enterprise, Utah was visited on June 15, 1983. Escalante is an underground silver operation which treats about 800 TPD (750 TPD design) of ore by fine grinding and cyanidation. Mr. Dean Van Dyke, Mill Superintendent, conducted the tour.

The ore is crushed using a 25 x 40 inch jaw crusher and is ground to 80 percent minus 325 mesh in a 500 hp 16.5 x 5 ft. SAG mill and an 800 hp 10.5 x 15 ft. ball mill. Mr. Van Dyke did not like the use of the SAG mill for this small tonnage - far too many problems for the possible economic gains. The SAG mill consumes approximately 1.2 lb/ton of grinding balls and 0.7 lb/ton of liners. They presently are experiencing 94 percent availability.

The average silver grade is about 10.0 opt of which they recover 81.3 percent. Cyanide consumption is about 2.0 lb/ton and cyanide strength used for leaching is about 7.0 lb/ton.

Pregnant solutions containing about 4.0 opt silver is precipitated by the Merrill-Crowe process. The precipitate is leached with sulfuric acid to remove most of the copper and the excess zinc. The leach is controlled by pH and EMF to minimize silver losses. The leached residue is filtered in a separate plate and frame filter.

Mr. Van Dyke is very proud of the refinery area. Two men and a foreman handle all aspects of the refinery and produce about 45,000 ounces of silver per week. Because of the preleach, very little slag is produced during refining. This slag is not cleaned even though it contains silver BB's, but is put back into the grinding circuit.

The smelting furnace is a gas fired reverbatory furnace manufactured by U.S. Smelter. The furnace is operated once per week.

Security in the refinery area is not as severe as some of the other mines especially when recovering gold. The silver is insured once it is placed in the vault. Also it is insured during shipment.

The ore after being ground in cyanide is pumped to a preg thickener. The thickener underflow is pumped to a series of agitators that use "Pro Chem" agitators with 25 hp motors. Initially 15 hp motors were specified but these were marginal and were replaced with 25 hp motors.

Largely because high strength NaCN solution used for leaching, the slurry is washed using two stages of drum filtration instead of the more normal CCD circuit. They had considerable difficulties with filtration until the proper filter cloth was found.

The plant was designed by Wright Engineers. Construction of the mill and surface facilities began in September, 1980 and were completed in August, 1981. Cost of the mill and surface facilities not including power or water was 15.6 million dollars.

Taylor Operation - Silver King Mines

The Taylor Mine owned and operated by Silver King Mines, Co. near Ely, Nevada was visited on June 15, 1983. We talked with Mr. Jim Kemtopp, Manager and Dan Smith.

Taylor is an open pit silver mine containing between 2.5 and 3.0 opt silver. Silver recovery is reported to be about 70 percent. Average stripping ratio is about 2.5 to 1.

The plant design capacity is 1200 TPD, 7 days per week. The mine run ore is crushed with a ?30x42? Kue Ken jaw crusher and then crushed to approximately 1/2 in. in closed circuit with two stages of cone crushers.

The ore is ground to 90 percent minus 325 using 2 stages of ball milling. Old 6x8 ball mills (8 total) were purchased and installed.

Surface plant initially cost about $12.0 million. During the next year an additional estimated 2.0 million was spent so the plant would operate. This included items such as a truck shop, conveyor covers, tailing dam, etc. Capital cost is estimated at between $11,000 and $12,000 per daily ton capacity. Apparently the plant was engineered, designed, and constructed by Silver King personnel. Concrete work and building erection was contracted.

Leaching is a standard agitation leach with a CCD circuit. Reagents are as follows:

NaCN Strength	-	2.5 lb/ton solution
NaCN Consumption	-	3-3.5 lb/ton
pH	-	12.2
CaO Consumption	-	± 6 lb/ton

High capacity Enviroclear thickeners are used for the pregnant (1-32 ft. diameter) and the CCD (4-28 ft. diameter) thickeners. Underflow densities are above 50 percent. They had a great deal of problems with the Enviroclear thickeners during the first year of operation but after they learned how to operate them there hasn't been many problems. Approximately 0.5 lb/ton of flocculant is used among the thickeners. Approximately 700 gpm of wash water is used.

Silver is recovered from the pregnant solution containing about 1.0 gpl Ag using the Merrill-Crowe process. The precipitate is partially dried and sold without smelting.

Total employment is about 80 people at an average wage of $10.00 per hour for the hourly people. Approximately 90 percent of the employees are ex-Kennecott employees. It was stated that the employees are hard-working and knowledgeable and have had no union problems, although the plant is unionized.

Break-even silver price was stated to be about $8.00 per ounce. This would indicate an operating cost of between $20.00 and $24.00 per ton.

ASARCO

Southwestern Exploration Division

June 14, 1983

TO: W. L. Kurtz

FROM: J. D. Sell

Glamis Gold Ltd.
Picacho Open-pit
Imperial County, Calif.

Mr. Robert J. Toporowski, 868 1st St. West, Yuma, AZ 85364, Phone (602) 783-4261, dropped in on 6/13/83. He left the Glamis Gold Ltd. Interim Report for January 31, 1982 (Attached).

Toporowski says he was a part owner of the Picacho deposit but has since sold his portion to Glamis, along with several others of the group. This group has a little money and hence is looking for a small leach situation that Asarco might not be interested in.

He had run across our name from the Portland Mine activity in Mohave County, but Fisher-Watt had already taken up the property before they got around.

Of course, he is quite keen on Picacho and apparently still does some consulting for them. He said that he would be pleased to take Asarco on a tour of the operations.

(He also said that Goldfields practically lives at Picacho, trying to test their "Mesquite-Chief" ore via the Picacho process).

James D. Sell

JDS:mek
Att.

cc: F. T. Graybeal
Gold/Bulk File

Consolidated Statement of Source & Application of Funds for Glamis Gold Ltd. & Chemgold Inc. in Canadian Funds

For the six month period – August 1, 1982 to January 31, 1983 with comparative figures for the same period for the previous year.

SOURCE:	1982	1983
Sale of Shares	$ 9,599.40	$ 63,170.00
Interest Earned	143,270.28	130,768.00
*Chemgold (Profit for Period)	–	1,007,611.00
Plus Items not requiring Cash Outlay:		
*Depreciation	44,812.54	105,021.00
*Amortization, etc.	9,669.40	249,451.00
*Income Tax Provision	–	90,318.00
TOTAL FUNDS	$207,351.62	$1,646,339.00
APPLICATION:		
Property - South Penn	$ –	$ 12,418.00
Glamis Administration	20,829.67	31,859.00
*Machinery - Chemgold	100,445.22	346,953.00
Chemgold Interest 150,000 Shares	–	52,500.00
*Deferred Development Costs	–	334,952.00
*Chemgold (Loss for Period)	589,317.05	–
	$710,591.94	$ 778,682.00
*LESS Minority Interest in Chemgold Inc. (Glamis Holding 65.18%)	221,204.61	234,115.00
TOTAL OUTLAY	$489,387.33	$ 544,567.00
NET INCREASE (DECREASE) IN WORKING CAPITAL	(282,035.71)	$1,101,772.00
WORKING CAPITAL (DEFICIENCY) AT THE BEGINNING OF THE PERIOD	770,623.89	135,256.00
WORKING CAPITAL (DEFICIENCY) AT THE END OF THE PERIOD	$448,588.18	$1,237,028.00

* *Chemgold*
UNAUDITED

Director

Director

Corporate Data

Company Name:	Glamis Gold Ltd.
Directors and Officers:	Chester F. Millar, *President* Adolf H. Lundin H. Lutz Klingmann F. W. Maycock G. B. Finlayson, *Secretary*
Offices:	
Corporate:	P.O. Box 34183, Station D Vancouver, B.C. V6J 4N1
Executive:	405 - 750 W. Pender Street Vancouver, B.C. V6C 2T7 Telephone: (604) 689-7842 Telex: 04-51457
Share Capital - Issued:	3,015,192 common shares at no par value
Stock Exchange Listing:	Vancouver Stock Exchange (GLG)
Bankers:	Canadian Imperial Bank of Commerce 300 West Hastings Street Vancouver, B.C. V6B 1K7
Legal Counsel:	Lawrence & Shaw 2500 - 595 Burrard Street Vancouver, B.C. V7X 1L1
Registered and Records Office:	Lawrence & Shaw Vancouver, B.C.
Auditors:	Dyke and Howard 1525 West 7th Avenue Vancouver, B.C. V6J 1S1
Registrar and Transfer Agent:	Yorkshire Trust Company 1100 Melville Street Vancouver, B.C. V6E 4B6
U.S. Subsidiary:	Chemgold, Inc. *Glamis Gold Ltd. owns 65.18% of Chemgold, Inc.*

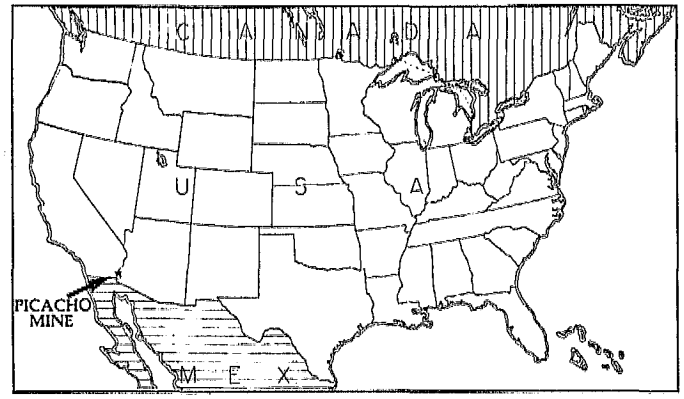
Glamis Gold Ltd.

Robert J. Toporowski
868 1st St. West
Yuma Az. 85364
* 602 783-4261
Glamis Gold Ltd

Interim Report January 31, 1982

Glamis Gold Ltd.

To The Shareholders



Glamis Gold Ltd. holds a two-thirds interest in Chemgold, Inc., a California corporation. Chemgold operates the Picacho open pit gold heap leach mine which is located in Imperial County, California, approximately eighteen miles north of Yuma, Arizona and four miles south of the Colorado River.

The property consists of 31 patented and 21 unpatented lode claims. The claims and the associated water right on the Colorado River, are owned by the Picacho Development Corp.

COVER: The Picacho mine of Chemgold, Inc. from the northwest in January, 1983.

FINANCIAL

The first six months of the current financial year showed a significant turn-a-round in the profitability of Glamis Gold Ltd.

HIGHLIGHTS

	Year Ended July 31, 1982	Six Months Ended January 31, 1983
No. of Shares Issued	2,857,192	3,015,192
Net Profit (Loss)	$ (86,632)	$929,119
Net Profit (Loss) Per Share	(3.0)¢	30.5¢
Working Capital at End of Period	$135,256	$1,237,028

Chemgold, Inc. Operations

PRODUCTION

	Year Ended July 31, 1982	Six Months Ended January 31, 1983
Gold - Troy oz.	1,610	3,995
Silver - Troy oz.	55	271

The average price received for gold sold during this period was U.S. $433 per oz.

LEACHING

A substantial upgrading program of equipment and facilities is just being completed.

Pregnant solution is now being pumped at an average rate of 830 tons per day.

Evaluation of the 55,000 ton initial heap has been completed. A total of 2,220 ozs. of gold were recovered from this heap from March, 1980 to December, 1981.

The core of the heap was found to contain no measurable gold while the perimeter of the heap was found to contain gold values ranging from .01 to .05 oz. per ton. This perimeter material is now being releached on the new heap.

Final recovery from the initial heap is estimated at 2,730 ozs. of gold vs. the original estimated gold content of only 1,705 ozs. The results of this work indicate that total recovery of gold is being obtained with the present leaching methods and that drilling assays tend to undervalue the true gold content of the ore.

Tests are underway to determine the affect of agglomerating finer materials on the rate at which gold can be recovered.

ORE RESERVES

A significant increase in ore reserves was indicated by exploration drilling done in December, 1982 and January, 1983.

A total of fourteen holes were drilled to a depth of 500 feet to determine the extent of the Diablo mineralized zone in the direction of the leach pads. Drilling done in 1933 to a depth of 120 feet had indicated an erratic zone of mineralization dipping to the north. The holes drilled recently intersected mineralization in an area of approximately 600 feet by 900 feet at a depth below 200 feet. The average thickness of the intersection was 138 feet.

This drilling indicates the total ore reserves to be in the order of 10,000,000 tons at .05 oz. per ton.

NEW PROJECTS

Glamis Gold Ltd. reached an agreement with the Crown-Sutton Joint Venture to explore and develop the South Penn property. This is a small but promising open pit prospect located approximately five miles north of the town of Republic in Washington State. The Crown-Sutton Joint Venture holds the property under four long-term mining leases with property owners in the area.

Nineteen rotary percussion holes were drilled to an average depth of 73 feet as part of the Phase I exploration program in January, 1983. Samples were shipped to Vancouver for analysis by Chemex Labs Ltd. The sampling and drilling done to date have indicated a reserve of 430,000 tons at 0.058 oz. gold per ton and 0.003 oz. silver per ton. Preliminary metallurgical testing is now being done on drill cuttings by Kappes, Cassiday and Associates of Reno, Nevada. A further series of holes will be drilled in March, 1983 to determine the extent of the mineralization in depth and to confirm an inferred ore tonnage and grade.

Glamis Gold Ltd. is actively pursuing and evaluating other open pit heap leach possibilities.

CORPORATE DEVELOPMENTS

Glamis Gold Ltd. is preparing an application for listing on the NASDAQ trading system in the United States.

On behalf of the Board of Directors,

C. F. Millar

C. F. Millar
President

March 9, 1983

Gold Bulk/Heap

ASARCO

Southwestern Exploration Division

June 8, 1983

To: JHCourtright
FRKoutz
HGKreis
WLKurtz
GJStathis
JRStringham

From: J. D. Sell

Heap Leaching Gold

Attached is a copy of the cover letter by R. L. Brown and the article by F. T. Graybeal on the heap leaching of gold.

Please note the final paragraph on page 2 of the cover letter.

Although field work and a vast number of drill holes with ore is the prerequisite for an operating mine, the ASARCO Library also has a number of publications which have pertinent thoughts on gold. These include:

a) Principal Gold-Producing Districts of the United States, by A. H. Koschmann and M. N. Bergendahl: USGS Prof. Paper 610 (1968).

b) The Geochemistry of Gold and Its Deposits, by R. W. Boyles: Geol. Survey of Canada Bulletin 280 (1979).

c) Precious Metals in the Northern Cordilleran, edited by A. A. Levinson: The Assoc. of Exploration Geochemists (1982).

d) The Geology of Gold in Ontario, edited by A. C. Colvine: Ontario Geological Survey, Miscellaneous Paper 110 (1983).

James D. Sell

James D. Sell

JDS/cg

Attachments

J. D. Sell

New York, May 31, 1983

MEMORANDUM FOR: Exploration Managers

Heap Leaching - Gold

Five years ago most of us considered that heap leaching operations in gold camps were mainly conducted by underfinanced, fly-by-night groups attempting to salvage remnants of surface oxidized surface material and old dumps around some of the various abandoned gold mining localities in the west. The view was that there was no real money to be made from this activity, but that these junior companies might make something promoting their operations on various over-the-counter and regional stock exchanges. Since that time, recognition that such organizations as Selection Trust, Goldfields, Canadian Superior, and Placer Development among the experienced mining companies; and Louisiana Land, Amoco Minerals, and Occidental as examples of very-well financed groups had made investments in heap leach operations, and were deriving from them handsome profits and cash flows, obviously necessitate a re-evaluation of that view.

Mr. Graybeal has put together a compilation of data concerning the heap leach operations and it, together with his observations concerning the state of the "heap leaching" art, is in my opinion a valuable piece of background information which I am herewith distributing to you.

The advent of heap leaching as a viable metallurgical recovery technique quite obviously has added a new class of mineral deposit - the class of deposits containing pervasively distributed free gold grading say from .04 gold to those deposits grading say .15 Au (which might be more economic under conventional milling techniques - to our list of legitimate exploration targets. There are of course, limitations - not all .04 gold deposits are heap leachable, one reason being that in some of these deposits the gold is caught up in pyrite or in some other sulfide, and the development of this technique obviously does not constitute a basis for the assumption that all areas in which there is presumptive very low grade mineralization of these grades should be drilled. Nevertheless, new technology has opened up a new group of mineral deposits which can earn attractive rates of return, and mining operations on these deposits are in operation and are now continuing to be constructed in the Western United States and in Australia, and operations of this type are either under construction or being contemplated in the Philippines, and in South Africa.

RECEIVED
JUN - 6 1983
S. W. U. S. EXPL. DIV.

In summary I think you will find the information Mr. Graybeal has compiled very useful to you, and I ask that you circulate it among your staff and make sure they understand both the possibilities and the limitations inherent in heap leaching.

R. L. Brown

Attachment

cc w/attachment

RLHennebach
RdeJOsborne
TCOsborne

New York, May 25, 1983

MEMORANDUM FOR: Mr. R. L. Brown

Review of Gold and Silver
Heap Leach Operations

Heap leaching is a simple process in which a cyanide solution is percolated through heaps of gold-bearing rock dissolving gold which is usually recovered on activated carbon. Geological factors of importance include distribution and grain size of gold in the ore, supergene oxidation which liberates sulfide-bound gold and reduces consumption of cyanide, and physical nature of the ore which governs permeability in the leach heaps.

The first significant operation to recover gold by heap leaching began at Cortez in 1971 as an addition to a conventional milling operation. The first significant independent heap leach operation began at Windfall in 1975 and there are presently 12 independent heap leach gold mines. I estimate that 320,000 oz. Au, or roughly 20% of the total USA gold production, was recovered by heap leaching in 1982 equivalent to about half of the primary open pit gold produced in the USA.

The attached fold-out chart summarizes information on 15 significant precious metal heap leach operations. Data are incomplete, change almost daily, and come from many sources of variable quality. Twelve of the 15 deposits are gold mines of which 6 are in Nevada and only 1 is outside the USA. Seven of the 15 are run by junior mining companies. Five additional operations for which only incomplete data are available are described briefly below. The following discussion will focus on the gold operations.

Reserve data in Table 1 suggest that at a gold price of $400/oz. a minimum exploration target size would be about 1 million tons averaging 0.08 oz. Au. Minimum ore grade for a large and low cost operation with recoverable gold and silver might be about 0.040 oz. Au equivalent. The mineralization in heap leach ores is usually disseminated with assay boundaries so reserves may change substantially with the price of gold. Exceptional care is required to measure the gold content of heap leach material which is near the lower limit of acceptable fire assay precision. Increased confidence in reserve estimates is obtained by running duplicate assays, assaying large samples (10-20 lbs.), drilling large diameter holes, and drilling a large number of holes.

TABLE 1, Summary of Heap Leach Reserves

Mine	Tons (millions)	oz. Au	oz. Ag
Haveluck (1)	2	0.054	-
Borealis	2	0.08	0.5
West End	3	0.085	-
Windfall	5	0.035	-
Alligator (2)	5	0.12	-
Annie Creek	7	0.043	0.16
Ortiz	7-16	0.053	-
Northumberland	10-16	0.052	-
Round Mountain	12-230	0.043	-
Zortman-Landusky	40-190	0.030	0.7

(1) Additional carbon-in-pulp ore was included in overall reserve which justified development.

(2) Highest grade heap leach operation; considering that carbonaceous ore must be stockpiled and additional reserves are present, this might have been better suited to conventional milling.

Major capital cost variables are the need for a crushing plant and whether mining is done by contractor. Capital costs in 1983 dollars for heap leach operations including Candelaria (Ag) ranged from $2300-6600/daily ton of ore for 6 of the operations where good data are available (see Table 2). Alligator Ridge was $15,000/daily ton and may be in error. Capital cost for the Zortman-Landusky complex are not known accurately, but if estimates of $1 million (1983 dollars) are close the cost would be $200/daily ton. This remarkably low cost resulted from contract mining of ore which did not require crushing. Table 2 shows that heap leach capital costs are substantially less than those for more conventional milling operations due to a relatively simple flowsheet. Capital costs per ounce of recoverable gold for heap leaching are equivalent to or less than costs for conventional milling as shown on Table 3.

TABLE 2. Capital Cost/Daily Ton of Ore Mined (1983 dollars)

Heap Leach		Conventional Milling	
Haveluck	- $2,300	Golden Sunlight	-$10,000
Santa Clara(1)	- 2,700	Pinson	- 22,000
Annie Creek	- 2,900	Mercur	- 27,000
Round Mountain	- 3,500	Jerritt Canyon	- 44,000
Candelaria	- 4,300	McLaughlin	- 66,000
Ortiz	- 6,300		
Northumberland	- 6,600	Average	-$34,000
Alligator Ridge	- 15,000		
Average	$ 5,400		

(1) Not on fold out table; see text

TABLE 3. Capital Cost/oz. of Recoverable Gold in Reserve (1983 dollars)

Heap Leach		Conventional Milling	
Haveluck(1)	- $16/oz.	Golden Sunlight	- $43/oz.
Annie Creek	- 18	Jerritt Canyon	- 49
Santa Clara	- 23	Pinson	- 63
Northumberland	- 58	McLaughlin	- 69
Round Mountain	- 59	Mercur	- 99
Ortiz	- 67		
Alligator Ridge	- 83		

(1) Heap leach and CIP reserve and plant.

Capital cost per ounce of recoverable gold in the original reserve for the Zortman-Landusky complex would be less than $1.00.

Construction intervals vary from 8-15 months. Six of the operations began production in the interval 1979-1981. Mining rates (ore) for 8 operations cluster between 1000-3000 tpd. The two largest operations are Round Mountain at 8000 tpd and Zortman-Landusky at 25,000 tpd. Round Mountain produces more gold than Zortman due to a longer operating season and higher grades. Although cold weather is a problem for plumbing systems and reduces the rate of dissolution of gold Zortman-Landusky, which is located in northern Montana, resumed mining on February 21, 1983, indicating cold weather is not a complete deterrent. The Victor Mine leaches heaps in enclosed vats at Cripple Creek and Round Mountain recently installed a heater to keep circulating fluids warmer. Because of the low ore grades stripping ratios are low and pit slopes are kept steep where possible. Cutoff grades are about 0.02 oz. Au.

Gold recoveries are in the range 60-70% for a 30-day interval. The most important and unpredictable variable in heap leaching is the movement of solution through the heaps. In order to obtain any recovery a cyanide solution must be brought in contact with the gold. Where the gold occurs on natural fractures crushing may not be required. If crushing is required or if the ore is unusually soft, excessive fines may reduce permeability and cut the flow of solution below that required to maintain production. Fines have been a problem at Alligator Ridge, Haveluck, and Candelaria. Agglomeration techniques have been developed by the U.S. Bureau of Mines to handle fines, but they increase capital and operating costs. Proper bedding of material on the leach heaps has been extensively researched to avoid compacting the ore with haulage equipment and to reduce sorting of fine and coarse material as individual loads of ore are added to the heaps. Careful selection of samples for metallurgical work is necessary as a single orebody may have several distinct ore types. Recognition of different ore types is difficult because most low grade deposits are explored using air hammer drilling methods which pulverize the sample, but careful logging can provide useful data. Most companies construct one or several 1000-5000 ton test heaps before final operating procedures are selected. The importance of accurate measurements of ore grades and recoveries cannot be overstated; for a 3000 tpd operation with $270/oz Au operating costs at a gold price of $430/oz., a change in recovered gold of six thousandths of an ounce per ton of ore (0.006 oz. Au) will change the annual net profit before taxes by $1 million.

Operating costs per ounce of gold produced seem to cluster at $250-270/oz. The Round Mountain estimate of $190/oz. Au is the 1983 GNP adjusted equivalent of estimates made in 1977 by Copper Range and Mountain States using a head grade of 0.045 oz. Au and is probably too low. Costs at Northumberland are unusually high at $402/oz., even when $68/oz. haulage costs are deducted.

Heap leach operations for which less data are available include:

1) Carlin Mining Co. Heap leaching is now in progress at Carlin. Major reserves at Maggie Creek and Gold Quarry of at least 150 million tons averaging 0.036 oz. Au will be agglomerated and heap leached. The Bootstrap dumps were previously leached as they lay after mining without prior sealing of the base. By 1984 it is estimated that 30% of the gold production at the Carlin complex (Carlin, Maggie, Gold Quarry, Rain, Blue Star) will be by heap leaching.

2) Picacho District, Calif. Chemgold, Inc. (Glamis Gold, Inc.) is leaching 300,000 tons and producing 600 oz. Au/mo. Agglomeration techniques are being studied. Recent drilling expanded reserve to 10 million tons averaging 0.05 oz. Au. Mineralization rumored to occur in mid-Tertiary detachment fault zones.

3. Santa Clara, Costa Rica. Located 40 miles from San Jose and owned 60% by United Hearne and 40% by Canadian Barranca. Reserves are 4.1 million tons averaging 0.063 oz. Au and recent drilling indicates substantial increases are probable. Came on stream September 1981 at 1500 tpd for $4 million. Operating costs are estimated at $6.00/ton.

4) Darwin Mine, Calif. Anaconda is drum agglomerating and heap leaching flotation tails averaging 1.6 oz. Ag and 0.01 oz. Au. Leach pads are compacted tailings.

5) Grubstake Mine, Montana. Resources America, Inc. is drum agglomerating and heap leaching 20,000 tons of dump material averaging 0.07-0.15 oz. Au. The Merrill-Crowe recovery plant is highly automated

Serious feasibility studies or pilot heap leach test work is ongoing at:

1) Pinson mine, Nev. A test pad is being built to study heap leach characteristics of 5 million tons of low grade material involving two ore types.

2) Bell Mtn. mine, Fairview Dist., Nev. American Pyramid is developing a combined milling and heap leach operation.

3) Dayton mill tailings, Comstock Lode, Nev. Tailings average 2.5 oz. Ag and are 65% minus 200 mesh. A 40-ton experimental heap was agglomerated and yielded 78% recovery in Ag in 5 days.

4) Packard mine, Nev. This property is located 3 miles southwest of the Rochester deposit. Reserve is estimated at 2 million tons averaging 1 oz. Ag. An initial unagglomerated 30,000 ton test was unsuccessful, but subsequent 300 ton pilot heaps with ore crushed to 80% minus 9/16 in. and agglomerated gave 65% Ag recovery in 6 days. Follow-up work on a 1000 ton heap gave similar results and work is continuing.

5) Goldfield, Nev. Initial 8000 ton test heap failed due to low permeability. Column leach tests on agglomerated rock crushed to minus 1 in. gave 80% recovery of gold in 14 days. Existing heaps will be recrushed and agglomerated as a pilot test.

6) El Plomo, Colo. E&B Explorations (German money) is evaluating 1 million tons of 0.057 oz. Au. Large test heap program was completed in 1982.

7) Gold Stripe, Plumas County, Calif. Test production by NCA Minerals will begin June 1983 from 10,000 ton test pad. Deposit has reserves 1.2 million tons averaging 0.063 oz. Au.

8) Zaca Mine, Alpine County, Calif. California Silver has operating permits for a 3000 tpd/heap leach mine. Reserves are 4.1 million tons of 0.036 oz. Au and 0.61 oz. Ag at 0.01 oz. Au cutoff. Column leach test of combined oxide and sulfide ore crushed to minus 3/8" recovered 76% Au and 38% Ag in 19 days.

9) German Gulch, Montana. Reserve is 11 million tons of 0.048 oz. Au. Heap leach recoveries on two 4500 ton heaps are reportedly 85%.

10) La Ronge Dist., Saskatchewan. Waddy Lake Resources has conducted tests said to be successful on low grade rock containing coarse gold.

11) Hedley, British Columbia. Banbury Gold Mines has applied for permit to begin heap leach production on gold-silver property.

12) Telfer mine, W. Australia. Newmont has completed a 1000 ton test heap leach pad and will crush ore to minus 1" and agglomerate. They say there is considerable heap leach ore, my guess is at least 5 million tons.

13) Masbate mine, Philippines Is. Atlas is building an asphalt heap leach test pad to evaluate characteristics of an eventual 4 million ton stockpile grading 0.030-0.045 oz. Au.

14) Gabiet Mine, Sudan. Robertson Research (U.K.) in complex ownership package is conducting pilot heap leach tests on mine tailings. Ultimate production might be 3000/oz. Au/yr.

15) Witwatersrand S. Africa. Feasibilities are reportedly underway to evaluate heap leaching on old dumps and tailings.

Heap leach operations are attractive because capital costs are low, technology is simple, and development intervals are rapid. These features permit the development of deposits otherwise too small or too low grade to justify a conventional milling operation. A substantial body of literature is available on the technology, but less information is available on existing operations, particularly operating costs. Operating costs per ounce of gold recovered may exceed costs at open pit milling operations, but are less than costs at the Homestake mine (1982 costs about $400/oz.). Although heap leach operating costs/oz. Au may be somewhat more than conventional milling operations, the capital investment may be somewhat less, and heap leaching therefore represents a valid and competitive mining investment. A 3000 tpd heap leach plant mining 0.07 oz. Au with 65% recovery at a gold price of $430/oz. and operating cost of $270/oz. would net $8 million before tax, clearly of sufficient size to be of interest to Asarco.

The many low grade gold prospects in various stages of exploration now known in the western US substantially outnumber known prospects of higher grade mineralization which might become conventional milling operations. This ratio may also apply to future discoveries and a significant effort is currently underway in low grade environments where the odds for discovery would appear to be more attractive.

F. T. Graybeal

F. T. Graybeal

CHARACTERISTICS OF GOLD AND SILVER HEAP LEACH OPERATIONS

MINE	OWNER	RESERVE (M=million)	CAPITAL COST STARTUP	PRODUCTION RATE	WASTE/ORE, CUTOFF	RECOVERY	OPERATING COST (1983)	GEOLOGY	COMMENTS
GOLD MINING OPERATIONS									
ALLIGATOR RIDGE Nevada	50% Amselco 50% Occidental	14M tons - 0.12 oz Au	$30M; permitting and construction took 15 months; startup Oct. 1980.	2200 tpd 60,000 oz Au/yr	W/O=3/1; 40-45° pit slope	70-75%	NA	Disseminated gold in zone of strong jasperoid alteration of silty limestone; deeper ore is carbonaceous.	Crushing and agglomeration required; carbonaceous ore is stockpiled; more ore in 2 other zones; pad made from compacted clay mined locally; 140 employees.
ANNIE CREEK S. Dakota	Wharf Resources	6.5M tons - 0.043 oz Au 0.16 oz Ag	$4M; startup Summer 1983.	750,000 tpy	Cutoff = 0.02 oz; 45° pit slope	74%	$225-275/oz	Similar to Gilt Edge (?); disseminated gold in pyritic (oxidized) latite and trachyte porphyry in breccia and shear zones.	Wharf holds 50% interest in adjacent Bald Mtn. property (Homestake 50%) with reserves of 9.9 million tons averaging 0.041 oz Au, 0.17 oz Ag.
BOREALIS Nevada	Houston Int. Minerals (Tenneco)	2.1M tons - 0.08 oz Au 0.50 oz Ag	Startup Nov. 1981.	2300 tpd 30,000 oz Au/yr 30,000 oz Ag/yr	NA	NA	NA	Disseminated gold and silver halides in oxidized, silicified and argillized tuff and volcanic breccia.	40 employees.
CORTEZ Nevada	Placer Development	2.2M tons - 0.041 oz Au at Gold Acres 2.8M tons at Cortez.	Startup 1971; part of a milling operation.	NA (not available)	Cutoff = 0.015 oz	50% from clayey Gold Acres ore; 65% from blocky Cortez ore.	NA	Largely Carlin-type with fine-grained disseminated gold in silty limestone, variably carbonaceous.	No crushing, 170,000 ton heaps.
HAVELUCK W. Australia	Whim Creek	1.4M tons - 0.054 oz Au heap leach.	$2M; startup June 1980; 10 mo. for construction.	1000 tpd 14,500 oz Au/yr	W/O=2/1; cutoff = 0.018 oz	70%; 30% in 30 days	$250/oz HL $270/oz CIP	Disseminated gold in strongly weathered and talc-altered mafic volcanic rock.	Began as 100% heap leach; excessive fines required conversion to 50% heap leach, 50% CIP; mining by contractor; total reserve is 4.5M tons at 0.066 oz Au.
NORTHUMBERLAND Nevada	Amoco Minerals	10M tons - 0.052 oz Au	$17M; startup April 1981; 8 mo. for construction.	3000 tpd 30,000 oz Au/yr 30,000 oz Ag/yr	W/O=1.8/1 cutoff = 0.020 oz	60% from jasperoid, 70% from shaley limestone	$402/oz excluding deprec./depl.; haulage is $6B/oz.	Carlin-type with disseminated gold in jasperoidized limestone and in decalcified silty limestone.	Crushing required; 10 mile haul pit to heaps; plastic liner under heaps; additional 6M tons averaging 0.034 oz Au at W/O=6.9/1; 110 employees.
ORTIZ New Mexico	Goldfields	6.8M tons - 0.053 oz Au	$14.5M; startup Feb. 1980; 12 mo. for construction.	3000 tpd 60,000 oz Au/yr	W/O=1.2/1; cutoff = 0.02 oz.	78% in 56 days; 60% in 23 days.	$261/oz excluding deprec./depl.	Disseminated gold in fractures and vugs in oxidized breccia pipe cutting quartzite and associated with latite dikes.	Additional 10M tons are below water table; mining done by contractor; crushing required; asphalt-lined heaps are cycled; 48 employees.
ROUND MOUNTAIN Nevada	Smokey Valley Mining Co. (50% Louisiana Land)	Started at 11.6M tons - 0.061 oz Au; now 195M tons - 0.043 oz Au, 0.08 oz Ag	$18M; startup Jan. 1977; 15 mo. for construction.	8000 tpd 75,000 oz Au/yr 30,000 oz Ag/yr	W/O=1/1; cutoff = 0.02 oz; 56° pit slope	67% in 26 days.	$190/oz (est. by FTG based on current ore grade of 0.045 oz Au).	Disseminated gold in iron-stained joints cutting dense weakly altered ash flow tuffs.	First large heap leach gold operation; crushing required; asphalt-lined heaps; heaps are cycled every 28 days; 240 employees.
VICTOR Colorado	Silver State Mining Corp.	3.5M tons - 0.06 oz. Au; potential for 8M tons.	$3M; startup March 1982; construction took 9 mo.	1000 tpd 20,000 oz Au/yr	NA	Close to 90% in 4 days.	$160/oz Au; mining cost is $.20/ton	400 ft. diameter breccia pipe.	Cripple Creek District; 75% of ore is mined by ripping; ore crushed to minus 1/2 in. and agglomerated with 90% of Au in solution by time cement cures; leaching in concrete "vats" covered for year-round leaching; 4-day cycle.
WEST END Idaho	Canadian Superior	3M tons - 0.085 oz Au.	Estimate $10M; startup July 1982; 11 mo. for construction.	3000 tpd	W/O=2/1	75%	NA	Oxidized gold-bearing arsenopyrite in shear zones in argillized schist, quartzite, and carbonate cut by trachyte dikes.	5-month operating season; 2 years to do EIS; 60 employees.
WINDFALL Nevada	Idaho Mining Corp.	5M tons - 0.035 oz Au 0.01 oz Ag	Startup July 1975.	1500 tpd	W/O - low; cutoff = 0.01 oz Au	NA	NA. Profitable in 1975 when gold was $160/oz (equal to $264/oz. Au in 1983).	Disseminated gold in jasperoidized and sanded dolomite.	No crushing; mining reduces sanded dolomite to 35% minus 200 mesh, ponds instead of sprinklers.
ZORTMAN-LANDUSKY Montana	Wharf Resources Ltd. Pegasus Explorations, Argo Gold Mining Corp., and others.	Zortman is 20M tons - 0.032 oz Au, 0.4 oz Ag; Landusky is 19M tons - 0.027 oz Au; 0.9 oz Ag.	Startup July 1979; construction took less than 12 mo.	12,000 tpd each; 67,000 oz Au and 146,000 oz Ag is total for both.	W/O=0.7/1	70% Au 35% Ag	$250/oz Au exclusive of Ag; equivalent to $4.50/ton ore including adminis., excluding deprec., depl., tax.	Disseminated gold and silver in weakly pyritic (now oxidized) shear zones cutting syenite. Zortman is old gold district, Landusky is old silver district; 8 miles apart.	Mining by contractor; heaps sealed with 2 ft. compacted bentonite; no crushing; 500K-1M ton heaps are permanent; early Zortman grades were 0.06 oz Au; additional Zortman reserves 30-40M tons, possibly 190M tons at Landusky; 65 employees; 1983 production began Feb. 21, 1983.
SILVER MINING OPERATIONS									
CANDELARIA Nevada	60% Occidental; will sell to NERCO.	13M tons - 3.15 oz Ag 0.002 oz Au.	$30M; startup September 1980.	8000 tpd 1,700,000 oz Ag/yr. 9,200 oz Au/yr.	NA	40% in 60 days	NA	Argentiferous Fe and Mn oxides in formerly pyritic chert, sandstone, and shale cut by quartz veins and chalcedonic zones.	Only full-scale heap leach silver mine in world; crushing and agglomerating required; closed Spring 1982 due to low Ag prices; 140 employees.
TOMBSTONE Arizona	Tombstone Exploration Company	Dumps contain 1M tons - 2.26 oz Ag and 0.037 oz Au. Lode averages 1 oz Ag.	Startup of dump material January 1981.	1500 tpd from dumps; 2000 tpd tpd from Contention zone.	NA	Lab tests indicate 75% for Au, 60% Ag both in 16 days.	Lode is profitable at Ag price of $10/oz.	Silver halides, sulfates in quartz-veined and silicified volcanics and siliceous limestone conglomerates, variable manganese.	Lode ore is mined by ripping; dump material crushed to minus 1/2" and agglomerated; leach pad is compacted mill tailings; agglomeration increased recoveries by 5x.
TUSCARORA Nevada	Tuscarora Assoc.	500,000 in dumps 2.5 oz Ag, 0.02 oz Au.	$700,000; startup October 1979; cost includes moving first 200,000 tons.	NA	NA	NA	$6.90/ton of ore.	Silver halides and gold in dumps in an old epithermal quartz-adularia vein camp.	Plastic-lined leach pads; plant treats 1400 tons solution/day; mining by contractor; 6 employees.

FTG - 5/26/83

JPS JMC

W. L. Kurtz

Gold Bulk/Heap

New York, May 31, 1983

MEMORANDUM FOR: Exploration Managers

Heap Leaching - Gold

Five years ago most of us considered that heap leaching operations in gold camps were mainly conducted by underfinanced, fly-by-night groups attempting to salvage remnants of surface oxidized surface material and old dumps around some of the various abandoned gold mining localities in the west. The view was that there was no real money to be made from this activity, but that these junior companies might make something promoting their operations on various over-the-counter and regional stock exchanges. Since that time, recognition that such organizations as Selection Trust, Goldfields, Canadian Superior, and Placer Development among the experienced mining companies; and Louisiana Land, Amoco Minerals, and Occidental as examples of very-well financed groups had made investments in heap leach operations, and were deriving from them handsome profits and cash flows, obviously necessitate a re-evaluation of that view.

Mr. Graybeal has put together a compilation of data concerning the heap leach operations and it, together with his observations concerning the state of the "heap leaching" art, is in my opinion a valuable piece of background information which I am herewith distributing to you.

The advent of heap leaching as a viable metallurgical recovery technique quite obviously has added a new class of mineral deposit - the class of deposits containing pervasively distributed free gold grading say from .04 gold to those deposits grading say .15 Au (which might be more economic under conventional milling techniques - to our list of legitimate exploration targets. There are of course, limitations - not all .04 gold deposits are heap leachable, one reason being that in some of these deposits the gold is caught up in pyrite or in some other sulfide, and the development of this technique obviously does not constitute a basis for the assumption that all areas in which there is presumptive very low grade mineralization of these grades should be drilled. Nevertheless, new technology has opened up a new group of mineral deposits which can earn attractive rates of return, and mining operations on these deposits are in operation and are now continuing to be constructed in the Western United States and in Australia, and operations of this type are either under construction or being contemplated in the Philippines, and in South Africa.

RECEIVED

JUN - 6 1983

EXPLORATION DEPARTMENT

In summary I think you will find the information Mr. Graybeal has compiled very useful to you, and I ask that you circulate it among your staff and make sure they understand both the possibilities and the limitations inherent in heap leaching.

R. L. Brown

Attachment

cc w/attachment

RLHennebach
RdeJOsborne
TCOsborne

New York, May 25, 1983

MEMORANDUM FOR: Mr. R. L. Brown

Review of Gold and Silver
Heap Leach Operations

Heap leaching is a simple process in which a cyanide solution is percolated through heaps of gold-bearing rock dissolving gold which is usually recovered on activated carbon. Geological factors of importance include distribution and grain size of gold in the ore, supergene oxidation which liberates sulfide-bound gold and reduces consumption of cyanide, and physical nature of the ore which governs permeability in the leach heaps.

The first significant operation to recover gold by heap leaching began at Cortez in 1971 as an addition to a conventional milling operation. The first significant independent heap leach operation began at Windfall in 1975 and there are presently 12 independent heap leach gold mines. I estimate that 320,000 oz. Au, or roughly 20% of the total USA gold production, was recovered by heap leaching in 1982 equivalent to about half of the primary open pit gold produced in the USA.

The attached fold-out chart summarizes information on 15 significant precious metal heap leach operations. Data are incomplete, change almost daily, and come from many sources of variable quality. Twelve of the 15 deposits are gold mines of which 6 are in Nevada and only 1 is outside the USA. Seven of the 15 are run by junior mining companies. Five additional operations for which only incomplete data are available are described briefly below. The following discussion will focus on the gold operations.

Reserve data in Table 1 suggest that at a gold price of $400/oz. a minimum exploration target size would be about 1 million tons averaging 0.08 oz. Au. Minimum ore grade for a large and low cost operation with recoverable gold and silver might be about 0.040 oz. Au equivalent. The mineralization in heap leach ores is usually disseminated with assay boundaries so reserves may change substantially with the price of gold. Exceptional care is required to measure the gold content of heap leach material which is near the lower limit of acceptable fire assay precision. Increased confidence in reserve estimates is obtained by running duplicate assays, assaying large samples (10-20 lbs.), drilling large diameter holes, and drilling a large number of holes.

TABLE 1, Summary of Heap Leach Reserves

Mine	Tons (millions)	oz. Au	oz. Ag
Haveluck (1)	2	0.054	-
Borealis	2	0.08	0.5
West End	3	0.085	-
Windfall	5	0.035	-
Alligator (2)	5	0.12	-
Annie Creek	7	0.043	0.16
Ortiz	7-16	0.053	-
Northumberland	10-16	0.052	-
Round Mountain	12-230	0.043 .038	-
Zortman-Landusky	40-190	0.030	0.7

(1) Additional carbon-in-pulp ore was included in overall reserve which justified development.

(2) Highest grade heap leach operation; considering that carbonaceous ore must be stockpiled and additional reserves are present, this might have been better suited to conventional milling.

Major capital cost variables are the need for a crushing plant and whether mining is done by contractor. Capital costs in 1983 dollars for heap leach operations including Candelaria (Ag) ranged from $2300-6600/daily ton of ore for 6 of the operations where good data are available (see Table 2). Alligator Ridge was $15,000/daily ton and may be in error. Capital cost for the Zortman-Landusky complex are not known accurately, but if estimates of $1 million (1983 dollars) are close the cost would be $200/daily ton. This remarkably low cost resulted from contract mining of ore which did not require crushing. Table 2 shows that heap leach capital costs are substantially less than those for more conventional milling operations due to a relatively simple flowsheet. Capital costs per ounce of recoverable gold for heap leaching are equivalent to or less than costs for conventional milling as shown on Table 3.

TABLE 2. Capital Cost/Daily Ton of Ore Mined (1983 dollars)

	Heap Leach		Conventional Milling		
	Haveluck	- $2,300	Golden Sunlight	-$10,000	Placer
	Santa Clara (1)	- 2,700	Pinson	- 22,000	Lacana
Wharf	Annie Creek	- 2,900	Mercur	- 27,000	Getty
Copper Range	Round Mountain	- 3,500	Jerritt Canyon	- 44,000	Freeport
Occi	Candelaria	- 4,300	McLaughlin	- 66,000	Homestake
Goldfields	Ortiz	- 6,300			
Amoco	Northumberland	- 6,600	Average	-$34,000	
Amselco	Alligator Ridge	- 15,000			
	Average	$ 5,400			

(1) Not on fold out table; see text

TABLE 3. Capital Cost/oz. of Recoverable Gold in Reserve (1983 dollars)

Heap Leach		Conventional Milling	
Haveluck(1)	- $16/oz.	Golden Sunlight	- $43/oz.
Annie Creek	- 18	Jerritt Canyon	- 49
Santa Clara	- 23	Pinson	- 63
Northumberland	- 58	McLaughlin	- 69
Round Mountain	- 59	Mercur	- 99
Ortiz	- 67		
Alligator Ridge	- 83		

(1) Heap leach and CIP reserve and plant.

Capital cost per ounce of recoverable gold in the original reserve for the Zortman-Landusky complex would be less than $1.00.

Construction intervals vary from 8-15 months. Six of the operations began production in the interval 1979-1981. Mining rates (ore) for 8 operations cluster between 1000-3000 tpd. The two largest operations are Round Mountain at 8000 tpd and Zortman-Landusky at 25,000 tpd. Round Mountain produces more gold than Zortman due to a longer operating season and higher grades. Although cold weather is a problem for plumbing systems and reduces the rate of dissolution of gold Zortman-Landusky, which is located in northern Montana, resumed mining on February 21, 1983, indicating cold weather is not a complete deterrent. The Victor Mine leaches heaps in enclosed vats at Cripple Creek and Round Mountain recently installed a heater to keep circulating fluids warmer. Because of the low ore grades stripping ratios are low and pit slopes are kept steep where possible. Cutoff grades are about 0.02 oz. Au.

Gold recoveries are in the range 60-70% for a 30-day interval. The most important and unpredictable variable in heap leaching is the movement of solution through the heaps. In order to obtain any recovery a cyanide solution must be brought in contact with the gold. Where the gold occurs on natural fractures crushing may not be required. If crushing is required or if the ore is unusually soft, excessive fines may reduce permeability and cut the flow of solution below that required to maintain production. Fines have been a problem at Alligator Ridge, Haveluck, and Candelaria. Agglomeration techniques have been developed by the U.S. Bureau of Mines to handle fines, but they increase capital and operating costs. Proper bedding of material on the leach heaps has been extensively researched to avoid compacting the ore with haulage equipment and to reduce sorting of fine and coarse material as individual loads of ore are added to the heaps. Careful selection of samples for metallurgical work is necessary as a single orebody may have several distinct ore types. Recognition of different ore types is difficult because most low grade deposits are explored using air hammer drilling methods which pulverize the sample, but careful logging can provide useful data. Most companies construct one or several 1000-5000 ton test heaps before final operating procedures are selected. The importance of accurate measurements of ore grades and recoveries cannot be overstated; for a 3000 tpd operation with $270/oz Au operating costs at a gold price of $430/oz., a change in recovered gold of six thousandths of an ounce per ton of ore (0.006 oz. Au) will change the annual net profit before taxes by $1 million.

Operating costs per ounce of gold produced seem to cluster at $250-270/oz. The Round Mountain estimate of $190/oz. Au is the 1983 GNP adjusted equivalent of estimates made in 1977 by Copper Range and Mountain States using a head grade of 0.045 oz. Au and is probably too low. Costs at Northumberland are unusually high at $402/oz., even when $68/oz. haulage costs are deducted.

Heap leach operations for which less data are available include:

1) Carlin Mining Co. Heap leaching is now in progress at Carlin. Major reserves at Maggie Creek and Gold Quarry of at least 150 million tons averaging 0.036 oz. Au will be agglomerated and heap leached. The Bootstrap dumps were previously leached as they lay after mining without prior sealing of the base. By 1984 it is estimated that 30% of the gold production at the Carlin complex (Carlin, Maggie, Gold Quarry, Rain, Blue Star) will be by heap leaching.

2) Picacho District, Calif. Chemgold, Inc. (Glamis Gold, Inc.) is leaching 300,000 tons and producing 600 oz. Au/mo. Agglomeration techniques are being studied. Recent drilling expanded reserve to 10 million tons averaging 0.05 oz. Au. Mineralization rumored to occur in mid-Tertiary detachment fault zones.

3. Santa Clara, Costa Rica. Located 40 miles from San Jose and owned 60% by United Hearne and 40% by Canadian Barranca. Reserves are 4.1 million tons averaging 0.063 oz. Au and recent drilling indicates substantial increases are probable. Came on stream September 1981 at 1500 tpd for $4 million. Operating costs are estimated at $6.00/ton.

4) Darwin Mine, Calif. Anaconda is drum agglomerating and heap leaching flotation tails averaging 1.6 oz. Ag and 0.01 oz. Au. Leach pads are compacted tailings.

5) Grubstake Mine, Montana. Resources America, Inc. is drum agglomerating and heap leaching 20,000 tons of dump material averaging 0.07-0.15 oz. Au. The Merrill-Crowe recovery plant is highly automated

Serious feasibility studies or pilot heap leach test work is ongoing at:

1) Pinson mine, Nev. A test pad is being built to study heap leach characteristics of 5 million tons of low grade material involving two ore types.

2) Bell Mtn. mine, Fairview Dist., Nev. American Pyramid is developing a combined milling and heap leach operation.

3) Dayton mill tailings, Comstock Lode, Nev. Tailings average 2.5 oz. Ag and are 65% minus 200 mesh. A 40-ton experimental heap was agglomerated and yielded 78% recovery in Ag in 5 days.

4) Packard mine, Nev. This property is located 3 miles southwest of the Rochester deposit. Reserve is estimated at 2 million tons averaging 1 oz. Ag. An initial unagglomerated 30,000 ton test was unsuccessful, but subsequent 300 ton pilot heaps with ore crushed to 80% minus 9/16 in. and agglomerated gave 65% Ag recovery in 6 days. Follow-up work on a 1000 ton heap gave similar results and work is continuing.

5) Goldfield, Nev. Initial 8000 ton test heap failed due to low permeability. Column leach tests on agglomerated rock crushed to minus 1 in. gave 80% recovery of gold in 14 days. Existing heaps will be recrushed and agglomerated as a pilot test.

6) El Plomo, Colo. E&B Explorations (German money) is evaluating 1 million tons of 0.057 oz. Au. Large test heap program was completed in 1982.

7) Gold Stripe, Plumas County, Calif. Test production by NCA Minerals will begin June 1983 from 10,000 ton test pad. Deposit has reserves 1.2 million tons averaging 0.063 oz. Au.

8) Zaca Mine, Alpine County, Calif. California Silver has operating permits for a 3000 tpd/heap leach mine. Reserves are 4.1 million tons of 0.036 oz. Au and 0.61 oz. Ag at 0.01 oz. Au cutoff. Column leach test of combined oxide and sulfide ore crushed to minus 3/8" recovered 76% Au and 38% Ag in 19 days.

9) German Gulch, Montana. Reserve is 11 million tons of 0.048 oz. Au. Heap leach recoveries on two 4500 ton heaps are reportedly 85%.

10) La Ronge Dist., Saskatchewan. Waddy Lake Resources has conducted tests said to be successful on low grade rock containing coarse gold.

11) Hedley, British Columbia. Banbury Gold Mines has applied for permit to begin heap leach production on gold-silver property.

12) Telfer mine, W. Australia. Newmont has completed a 1000 ton test heap leach pad and will crush ore to minus 1" and agglomerate. They say there is considerable heap leach ore, my guess is at least 5 million tons.

13) Masbate mine, Philippines Is. Atlas is building an asphalt heap leach test pad to evaluate characteristics of an eventual 4 million ton stockpile grading 0.030-0.045 oz. Au.

14) Gabiet Mine, Sudan. Robertson Research (U.K.) in complex ownership package is conducting pilot heap leach tests on mine tailings. Ultimate production might be 3000/oz. Au/yr.

15) Witwatersrand S. Africa. Feasibilities are reportedly underway to evaluate heap leaching on old dumps and tailings.

Heap leach operations are attractive because capital costs are low, technology is simple, and development intervals are rapid. These features permit the development of deposits otherwise too small or too low grade to justify a conventional milling operation. A substantial body of literature is available on the technology, but less information is available on existing operations, particularly operating costs. Operating costs per ounce of gold recovered may exceed costs at open pit milling operations, but are less than costs at the Homestake mine (1982 costs about $400/oz.). Although heap leach operating costs/oz. Au may be somewhat more than conventional milling operations, the capital investment may be somewhat less, and heap leaching therefore represents a valid and competitive mining investment. A 3000 tpd heap leach plant mining 0.07 oz. Au with 65% recovery at a gold price of $430/oz. and operating cost of $270/oz. would net $8 million before tax, clearly of sufficient size to be of interest to Asarco.

The many low grade gold prospects in various stages of exploration now known in the western US substantially outnumber known prospects of higher grade mineralization which might become conventional milling operations. This ratio may also apply to future discoveries and a significant effort is currently underway in low grade environments where the odds for discovery would appear to be more attractive.

F. T. Graybeal

F. T. Graybeal

CHARACTERISTICS OF GOLD AND SILVER HEAP LEACH OPERATIONS

MINE	OWNER	RESERVE (M=million)	CAPITAL COST STARTUP	PRODUCTION RATE	WASTE/ORE, CUTOFF	RECOVERY	OPERATING COST (1983)	GEOLOGY	COMMENTS
GOLD MINING OPERATIONS									
ALLIGATOR RIDGE Nevada	50% Amselco 50% Occidental	14M tons - 0.12 oz Au	$30M; permitting and construction took 15 months; startup Oct. 1980.	2200 tpd 60,000 oz Au/yr	W/O=3/1; 40-45° pit slope	70-75%	NA	Disseminated gold in zone of strong jasperoid alteration of silty limestone; deeper ore is carbonacous.	Crushing and agglomeration required; carbonaceous ore is stockpiled; more ore in 2 other zones; pad made from compacted clay mined locally; 140 employees.
ANNIE CREEK S. Dakota	Wharf Resources	6.5M tons - 0.043 oz Au 0.16 oz Ag	$4M; startup Summer 1983.	750,000 tpy	Cutoff = 0.02 oz; 45° pit slope	74%	$225-275/oz $8.75/ton	Similar to Gilt Edge (?); disseminated gold in pyritic (oxidized) latite and trachyte porphyry in breccia and shear zones.	Wharf holds 50% interest in adjacent Bald Mtn. property (Homestake 50%) with reserves of 9.9 million tons averaging 0.041 oz Au, 0.17 oz Ag.
BOREALIS Nevada	Houston Int. Minerals (Tenneco)	2.1M tons - 0.08 oz Au 0.50 oz Ag	Startup Nov. 1981.	2300 tpd 30,000 oz Au/yr 30,000 oz Ag/yr	NA	NA	NA	Disseminated gold and silver halides in oxidized, silicified and argillized tuff and volcanic breccia.	40 employees.
CORTEZ Nevada	Placer Development	2.2M tons - 0.041 oz Au at Gold Acres 2.8M tons at Cortez.	Startup 1971; part of a milling operation.	NA (not available)	Cutoff = 0.015 oz	50% from clayey Gold Acres ore; 65% from blocky Cortez ore.	NA	Largely Carlin-type with fine-grained disseminated gold in silty limestone, variably carbonaceous.	No crushing, 170,000 ton heaps.
HAVELUCK W. Australia	Whim Creek	1.4M tons - 0.054 oz Au heap leach.	$2M; startup June 1980; 10 mo. for construction.	1000 tpd 14,500 oz Au/yr	W/O=2/1; cutoff = 0.018 oz	70%; 30% in 30 days	$250/oz HL $270/oz CIP $10.31/ton	Disseminated gold in strongly weathered and talc-altered mafic volcanic rock.	Began as 100% heap leach; excessive fines required conversion to 50% heap leach, 50% CIP; mining by contractor; total reserve is 4.5M tons at 0.066 oz Au.
NORTHUMBERLAND Nevada	Amoco Minerals	10M tons - 0.052 oz Au	$17M; startup April 1981; 8 mo. for construction.	3000 tpd 30,000 oz Au/yr 30,000 oz Ag/yr	W/O=1.8/1 cutoff = 0.020 oz	60% from jasperoid, 70% from shaley limestone	$402/oz excluding deprec./depl.; haulage is $68/oz. 12.54/ton 2.38/ton	Carlin-type with disseminated gold in jasperoidized limestone and in decalcified silty limestone.	Crushing required; 10 mile haul pit to heaps; plastic liner under heaps; additional 6M tons averaging 0.034 oz Au at W/O=6.9/1; 110 employees.
ORTIZ New Mexico	Goldfields	6.8M tons - 0.053 oz Au	$14.9M; startup Feb. 1980; 12 mo. for construction.	3000 tpd 60,000 oz Au/yr	W/O=1.2/1; cutoff = 0.02 oz.	78% in 56 days; 60% in 23 days.	$261/oz excluding deprec./depl. 10.79/ton	Disseminated gold in fractures and vugs in oxidized breccia pipe cutting quartzite and associated with latite dikes.	Additional 10M tons are below water table; mining done by contractor; crushing required; asphalt-lined heaps are cycled; 48 employees.
ROUND MOUNTAIN Nevada	Smokey Valley Mining Co. (50% Louisiana Land)	Started at 11.6M tons - 0.061 oz Au; now 195M tons - 0.043 oz Au, 0.08 oz Ag	$18M; startup Jan. 1977; 15 mo. for construction.	8000 tpd 75,000 oz Au/yr 30,000 oz Ag/yr	W/O=1/1; cutoff = 0.02 oz; 56° pit slope	67% in 26 days.	$190/oz (est. by FTG based on current ore grade of 0.045 oz Au). 5.73/ton	Disseminated gold in iron-stained joints cutting dense weakly altered ash flow tuffs.	First large heap leach gold operation; crushing required; asphalt-lined heaps; heaps are cycled every 28 days; 240 employees.
VICTOR Colorado	Silver State Mining Corp.	3.5M tons - 0.06 oz. Au; potential for 8M tons.	$3M; startup March 1982; construction took 9 mo.	1000 tpd 20,000 oz Au/yr	NA	Close to 90% in 4 days.	$160/oz Au; mining cost is $.20/ton 8.64/ton	400 ft. diameter breccia pipe.	Cripple Creek District; 75% of ore is mined by ripping; ore crushed to minus 1/2 in. and agglomerated with 90% of Au in solution by time cement cures; leaching in concrete "vats" covered for year-round leaching; 4-day cycle.
WEST END Idaho	Canadian Superior	3M tons - 0.085 oz Au.	Estimate $10M; startup July 1982; 11 mo. for construction.	3000 tpd	W/O=2/1	75%	NA	Oxidized gold-bearing arsenopyrite in shear zones in argillized schist, quartzite, and cargonate cut by trachyte dikes.	5-month operating season; 2 years to do EIS; 60 employees.
WINDFALL Nevada	Idaho Mining Corp.	5M tons - 0.035 oz Au 0.01 oz Ag	Startup July 1975.	1500 tpd	W/O - low; cutoff = 0.01 oz Au	NA	NA. Profitable in 1975 when gold was $160/oz (equal to $264/oz. Au in 1983).	Disseminated gold in jasperoidized and sanded dolomite.	No crushing; mining reduces sanded dolomite to 35% minus 200 mesh, ponds instead of sprinklers.
ZORTMAN-LANDUSKY Montana	Wharf Resources Ltd. Pegasus Explorations, Argo Gold Mining Corp., and others.	Zortman is 20M tons - 0.032 oz Au, 0.4 oz Ag; Landusky is 19M tons - 0.027 oz Au; 0.9 oz Ag.	Startup July 1979; construction took less than 12 mo.	12,000 tpd each; 67,000 oz Au and 146,000 oz Ag is total for both.	W/O=0.7/1	70% Au 35% Ag	$250/oz Au exclusive of Ag; equivalent to $4.50/ton ore including adminis., excluding deprec., depl., tax. 5.62/ton	Disseminated gold and silver in weakly pyritic (now oxidized) shear zones cutting syenite. Zortman is old gold district, Landusky is old silver district; 8 miles apart.	Mining by contractor; heaps sealed with 2 ft. compacted bentonite; no crushing; 500K-1M ton heaps are permanent; early Zortman grades were 0.06 oz Au; additional Zortman reserves 30-40M tons, possibly 190M tons at Landusky; 65 employees; 1983 production began Feb. 21, 1983.
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TUSCARORA Nevada	Tuscarora Assoc.	500,000 in dumps 2.5 oz Ag, 0.02 oz Au.	$700,000; startup October 1979; cost includes moving first 200,000 tons.	NA	NA	NA	$6.90/ton of ore.	Silver halides and gold in dumps in an old epithermal quartz-adularia vein camp.	Plastic-lined leach pads; plant treats 1400 tons solution/day; mining by contractor; 6 employees.

FTG - 5/26/83

copy 1) RLB 2) PGV 3) WK Gold Bulk

LACANA 1982 ANNUAL REPORT

Mineral Exploration and Development — United States

General

During the year, the Company carried out exploration, development and evaluation work on a number of precious metals properties in the United States. Work in the western United States was directed through the Company's Reno and Coeur d'Alene offices and a consultant in Denver. Work in the eastern United States was managed from Toronto.

Relief Canyon

One of the more important developments for the Company was the finalization late in the year of an option agreement with a major U.S. company to acquire a 90% interest in a low-grade gold deposit located 100 miles northeast of Reno. On the basis of reconnaissance drilling, indicated reserves are in the order of 5-10 million tons grading .04 to .06 ounces of gold per ton. The gold-bearing zones are near surface and are mineable by open-pit methods with a low stripping ratio.

Preliminary metallurgical test work indicates that the gold is amenable to extraction by low-cost, heap-leaching methods. The property is well located with water, power and other facilities nearby.

The Company has budgeted over $1 million to do detailed drilling, metallurgical testing and feasibility studies in 1983.

Cordex Syndicates

The Cordex IV Syndicate, comprised of Dome Exploration (U.S.) Ltd., Rayrock Mines Inc. and Lacana, continued an active exploration program in the western United States with the Boulder Creek (Dee Gold) project, described below, receiving the most attention.

The same companies have formed the Cordex V Syndicate and will be conducting drilling programs on at least four recently acquired gold properties as well as preliminary exploration on a number of other properties in 1983.

Dee Gold

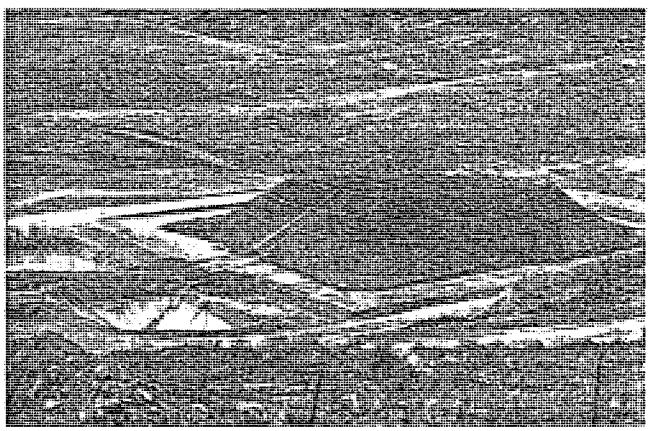
Lacana Mining Inc. has a 29.3% interest in this gold property consisting of 101 leased unpatented mining claims located at the north end of the Carlin gold belt in north-central Nevada.

Exploration and development work to date includes the drilling of 246 rotary holes totalling 26,000 feet of drilling, underground sampling via a decline, numerous metallurgical tests and two large-scale (5,000 tons) heap-leach tests at a total cost of approximately $2 million.

Open pit ore reserves are estimated to be approximately 2.8 million tons assaying 0.12 oz. gold per ton with a 7 to 1 stripping ratio.

A production decision, including the construction of a 900 ton per day mill at an estimated cost of (U.S.)$24,000,000, was made in April, 1983. Production is planned to commence in the latter part of 1984. Better than average grade ore can be milled in the first two years resulting in a faster pay-down of senior debt financing.

Mike Frannaca, Mike Easdon and Harvey Sobel at Lacana Mining Inc.'s Relief Canyon gold prospect.



Heap leaching tests on Dee Mine gold ore, Nevada.

The Louisiana Land + Exploration Co.
1992 10-K

Round Mtn Nevada
WLK Gold File Heap/Bulk

The operation, which was started in November, 1976, consists of open pit mining, crushing, heap leaching, and precious metal recovery by carbon absorption and electrowinning. The product is dore bullion, about 65% gold and 35% silver, which is shipped to outside refineries and the metals are priced at the London gold and silver daily quotations. Gross reserves are estimated by LL&E as 228 million tons of proved and probable ore at a strip ratio of 2.87:1 and containing 8.5 million troy ounces of gold. The properties are burdened with an 18% net profits interest in the profits from operations, after recovery of invested capital. LL&E earns an operator's fee of 1.5% of expenditures.

The heap leaching process now used to extract gold recovers about 60% of the contained gold. This process will not be effective on the deeper portion of the new ore. Feasibility studies were started in 1980 and are continuing in 1983 to determine an optimum method of extracting gold and silver from the known mineralization. It is possible that these studies will indicate the need for substantial new investment in milling facilities to achieve higher recoveries of metals.

The following table sets forth LL&E's 50% interest in the activities of the mine for the periods indicated.

	1982	1981	1980
Proved and probable ore reserves (thousands of tons)	114,152	97,700	18,063
Average grade-gold (ounces per ton)	0.037	0.043	0.042
Ore milled-thousands of tons	1,369	1,145	1,087
Waste mined-thousands of tons	2,980	4,357	2,410
Production (ounces)	36,281	29,469	28,795
Sales (ounces)	34,892	29,553	28,156
Average price (per ounce)	$ 366.52	460.00	620.99

CAPITAL EXPENDITURES

Capital expenditures applicable to mining operations of LL&E's wholly-owned subsidiaries are set forth in the table below for the periods indicated.

	(Millions of dollars)		
	1982	1981	1980
Copper mining and refining	$46.7	39.2	4.8
Coal mining	1.1	7.8	3.1
Total	$47.8	47.0	7.9

xc: FTG
PGV
WLK Gold File Bulk

copy for PGW 3/14/83, route to ~~EDS~~ Bulk/Hard ~~Gold~~ Description
one file Gold

New York, March 3, 1983

FILE MEMORANDUM

Haveluck Mine
Whim Creek Mining Company
Meekatharra, W. Australia

I visited this operation with Perth Office staff in November 1982. Whim Creek is an Australian company which is 36% owned by Northgate Exploration Ltd. and Westfield Minerals Ltd. both of Canada. The Haveluck Mine is part of the Meekatharra joint venture in which Whim Creek owned 85% and CRA Ltd. owned 15%. Early in 1983 CSR sold their interest to Whim Creek who are now the 100% owner of all Meekatharra properties. At the time of our visit, the joint venture controlled seven separate zones spread along a strike length of 6 miles from Haveluck, located 3 miles north of Meekatharra to Paddys Flat, located 3 miles south of Meekatharra. The aggregate reserves in these seven separate zones in November 1982 were 4.5 million tons averaging 0.066 oz Au. This reserve excludes the Paddys Flat zone which is currently being explored to be described below.

Haveluck Operations. The Haveluck Mine is an open pit with a reserve of 1,348,000 tons averaging 0.054 oz Au. The ore zone is linear, averaging 70 ft. wide and is confined to a steeply dipping talc-chlorite-carbonate schist with small amounts of fuchsite, all cut by quartz feldspar porphyry dikes. A banded iron formation runs parallel to the ore zone 450 ft. west of the pit. These rocks have been deeply and pervasively oxidized to depths in excess of 200 ft. This oxidation has resulted in gold being 80% leached within 12 ft. of the surface. In addition, there is no carbonate remaining within the first 100 ft. of the surface.

Quartz veins are present only in the quartz feldspar porphyry which forms 15-20% by volume of the ore zone. The hanging wall ores are harder than the footwall ores and correlate with a decrease in the grade of the ore from the hanging wall to the footwall portion of the ore zone. The gold is entirely in the free state both in the oxide and sulfide zones and averages 960 fineness in the oxide zone. The distribution of gold in the hanging well is much more irregular than in the footwall.

RECEIVED
MAR - 8 1983
EXPLORATION DEPARTMENT

The Haveluck Mine was originally designed as a heap leach operation and was brought onstream in April 1980 at a cost of $A2.12 million not including overburden stripping and mining by a separate contractor. Construction time was 10 months.

Haveluck was an old underground mine which produced 10,000-20,000 tons of ore averaging 0.45 oz Au, mostly from the porphyry in the hanging wall portion of the open pit orebody. Bulk metallurgical samples for heap leach testing came from 25-30 ft. deep pits in the area of the old underground mine. The metallurgical samples were not characteristic of the orebody because as mining progressed below 30 ft. it was found that most of the ore consisted of very soft, crumbly rock containing 10-30% fine grain talc and it is obvious from a visual inspection of the ore that there would be a major problem with excessive development of fines severely restricting percolation in the heaps. Although the original heap leach plant was designed to treat 240,000 tons per year, the problem with excess fines resulted in only half that amount being leached. As a result a screening and CIP plant was rushed into service at a cost of $A746,000 and brought onstream in 6 months providing the anticipated 240,000 tons/year capacity. The CIP technology was chosen over agglomeration because it was better known and could be built immediately with no pilot testing. We were told that the CIP circuit would operate at a lower cost than agglomeration, although this may simply be a rationalization on the part of Whim Creek staff to minimize the embarrassment caused by the near failure of the operation. Clearly, Whim Creek was very lucky to have survived the near-fatal error of not obtaining a full suite of ore types for metallurgical testing.

Ore is now divided roughly 50% between the heap leach and CIP operations. The crushing and screening plant sorts material at a plus or minus 10 millimeter grain size (one reference says 1mm.). The distribution of gold is approximately the same in both the fine and coarse fractions. Leach heaps are loaded with a clam shell. Originally smaller heaps were enlarged laterally in an attempt to reduce the effects of a strong evaporation gradient out the sides of the dumps. The dumps are currently 18 ft. high and contain 5000 tons each. Water consumption is 600,000 kiloliters per day of which 25% is recycled. Water is pumped from old underground workings in the area and from new discoveries of ground water which are currently being developed at a cost of $100,000. The pregnant solution from the dumps contains 0.3-0.4 grams of gold per ton of water. Leaching of a particular dump is terminated when the pregnant solution drops to 0.2 grams of

gold. Heap leach recovery is 65-70%. It takes 30 days to recover 30% and 90-100 days to accumulate the 65% minimum recovery. Heaps are then rested for 6 months and leached a second time which usually provides 10-20 days of flow to the recovery plant of economic grade pregnant solution. The incremental recovery in this second phase of leaching has not yet been determined. Reagent consumption is 1 kilogram cyanide and 2 kilograms soda ash per ton of ore. The heaps are lined with plastic which costs $2000 for a 14x60 meter sheet. Water is sprayed using wigglers. The recovery plant uses extruded carbon instead of coconut shell carbon because the former is cylindrical and is less of a problem in the CIP filters. We inquired why the coconut carbon was not used in the absorption tanks for the heap leach and no particular answer was given. Coconut carbon was recognized as more active than extruded carbon, but the loading using the extruded form averages 4000 grams of gold per ton which was deemed sufficient.

The mine operates on 2-12 hour shifts. Haveluck will be the only heap leach operation in the Meekatharra JV with the remainder using CIP technology. Cutoffs in the open pit are 0.018 oz Au. Total operating costs for the heap leach operation are $270/oz. and for the CIP $290/oz. Water is apparently a valuable commodity because there is a major dust problem in the open pit and on all roads in and around the operation.

Prohibition Zone. This deposit lies southeast of Meekatharra and is a typical nodular iron laterite. In contrast to the Haveluck deposit gold is strongly encriched in the Prohibition deposit. Host rocks are a banded cherty iron formation containing gold in narrow shear zones. Near the surface the gold-bearing zone has mushroomed out to form a high grade blanket of relatively uniform grade. This is the only lateritic deposit in the entire Meekatharra area. The laterite averages 0.21 oz Au which is so fine grained that the prospectors in the area couldn't pan the gold and as a result missed the deposit. Total Prohibition reserves including the low grade unenriched shear zone below the lateritic cap contain 1.2 million tons averaging 0.072 oz Au. The laterite varies from 0 to 8 meters in thickness.

Finian-Consols. This contains 624,000 tons of tailings averaging 0.04 oz Au which will eventually be treated in a CIP circuit.

Paddys Flat. Previous production in this area is about 900,000 oz Au from various underground ores averaging around 0.48 oz Au. The area of Paddys Flat consists of numerous old mines in a 4 mile strike length of talc-carbonate schist. The geology is incompletely known, but quartz feldspar porphyry is present at Paddys Flat and the zone of gold mines parallels and lies just east of a zone of banded iron formation. Paddys Flat is currently under option to Chevron. The option requires that Chevron spend the first million dollars on targets more than 200 ft. below the surface. They may then spend an additional $2 million on targets within 200 ft. of the surface and the aggregate $3 million expenditure will earn Chevron a 50% interest in the property. Whim Creek's chief geologist Max Maczurad estimates Paddys Flat contains roughly 40 million tons which will average about 0.06 oz Au within 200 ft. of the surface.

F. T. Graybeal

F. T. Graybeal

cc: WLKurtz
DECrowell

Cerro Gold
Bulk Heap

FROM: F. T. GRAYBEAL

TO: WE Kurtz
DM Smith, Jr.
PG Vikre.

copies of attached to each

STERLING MINE

Sounds like the Sterling mine is rather small. Said to be 15 miles E. of Beatty.
E + B boys said 90% recovery.
Much of plant in trailers.
Buy water in Beatty. Contacts are

1. Greg Austin
   Saga Exploration - Winnemucca
2. Paul Saxton
   Vancouver Office.

10/12/82

HOLE NO.	Zone	Interval	Width	Oz.Gold.t	Oz.Silver/t	Zinc	Copper
B10-82-9	West	1163.9-1193.0	19.1	0.02	trace	10.4%	0.80%
	Main	1328.3-1368.0	32.0	.19	2.0	5.2%	.30%

FURTHER ASSAYS FROM CALIFORNIA PROJECT

Dennis C.Baxter, project manager for Colony Pacific Explorations Ltd. has reported assay results from a further diamond drill hole on the Blue Moon property located 125 miles south of Sacramento, California. Hole B10-82-9 bottomed at 1,452 feet and intersected 5 separate mineralized zones. (See table above) Hole B11-82-10 is currently drilling from a station 200 feet north of B10-82-9. It's designed to test the down dip extension of the mineralization, approximately 350 feet north of B-10.

Plans are underway to begin exploratory drilling to the south of the current area of drilling. The structural features of the Blue Moon zones are estimated to persist for 810,000 feet south along strike. (See previous stories in GCNL No.182, 161 etc.)

RECEIVED OCT 13 1982 F.T.G.

E&B EXPLORATIONS INC.

OCT 13 1982

PARTICIPATION IN THIRTY-THREE EXPLORATION PROJECTS OUTLINED IN ANNUAL REPORT - Henry G.Ewanchuk, president of E&B Explorations Inc., states in the annual report for the year ended Dec.31,1981 that exploration expenditures for the year were $10,500,000, bringing the funds expended since 1979 to $22,500,000, and placing E&B amongst the more active mineral exploration firms in Canada. The company has broadened the base by expanding the nature of the mineral properties. The report reviews 33 projects-location, mineral, partners and operators. He states that the technical aspects of the feasibility study on re-opening the Bralorne mine in southwestern B.C. have been completed. "The results are encouraging but the decision whether to proceed (expected in mid-1982) depends on the cost of money and value of the product." E&B's comprehensive program of mine rehabilitation and exploration continued through 1981, raising total expenditures to in excess of $5,000,000 Canadian funds. Haulage routes were cleared out, timbering completed where necessary, and water successfully pumped 1,016 feet up from the 19 level with the dewatering continuing into 1982. As each level was drained, the exposed veins were sampled to confirm the ore grades. Established reserves between 4 and 26 levels are calculated at close to 1,000,000 tons.

E&B retains its unique partnership advantages, including continued association with Geomex Development Inc. of Calgary and Denver, the North American general partners of Sedimex Mineralexplorations GmbH of West Germany. As general operator for Geomex, E&B has assembled a diverse portfolio of joint ventures. E&B Explorations Inc. is in the process of merging operations with E&B Canada Resources, the associate company of Calgary.

In 1981, E&B spent $500,000 as part of an option to spend $1,700,000 by 1984 to earn a 50% interest in the Cariboo-Bell project from Highland Crow Resources Ltd. The 1981 program included a geochemical soil survey, over 5,700 feet of diamond drilling, nearly 4,25 feet of rotary percussion drilling and metallurgical testing. The mineral reserves are now calculated to contain 81,000,000 tons grading 0.487% copper equivalent which includes significant gold values. Recent metallurgical testing has produced encouraging results and additional work is planned.

The Newpass gold mine in central Nevada is considered a promising new venture. E&B will acquire a 66.7% interest in the property for a total expenditure of $410,000 U.S. Approximately 25,000 ounces of gold have already been produced in previous years from the mine's three main veins. During 1981, E&B undertook a program of geologic mapping and soil sampling, as well as surface and underground surveying in the vicinity of the mine's old workings.

The ElPlomo gold property consists of 324 hectares on an old Spanish land grant in Colorado. E&B will earn a 50% undivided interest in the property with an expenditure of $1,800,000 U.S. before August 1,1983. Reserves in two zones are now estimated at 1,090,000 tons grading 0.057 oz.per ton. Thus far, geological evaluations have demonstrated a high success probability for a leach operation at the mine.

In the fall of 1980, only nine months after E&B headed up this joint venture, the Sterling gold mine, deep in Nevada's Nye county, came into full production. This past year proved to be equally busy and more successful. Approximately 9,300 ounces of gold were recovered in 1981 and both mining and leaching activities were expanded. In all, the ore reserves developed up to November 1981 totalled over 176,000 tons averaging 0.285 oz. gold per ton. Additional ores have been discovered with additional exploration work. Exploration work is to resume as gold prices improve.

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WLK

Wall Street Jnl. 12/1/82
Gold Hep/Bulk

copy with GJS
copy gives file
GJS tell where
file.

File
Chocolate Mtn. Placer
Imperial Co., Calif.

GJS – any comment

circulate – JDS
~~GJS~~
FRK
HGK
JHC

Unit of Consolidated Gold Makes Discovery

By a WALL STREET JOURNAL *Staff Reporter*

NEW YORK—Gold Fields Mining Corp., a unit of Consolidated Gold Fields PLC of England, said it discovered "gold mineralization of possible economic significance"—containing approximately two million ounces — on minerals claims under its control in Imperial County, Calif.

The company said that in an area about 2,000 feet long and 1,000 feet wide, 165 relatively shallow holes intersected a zone containing "potentially significant gold values."

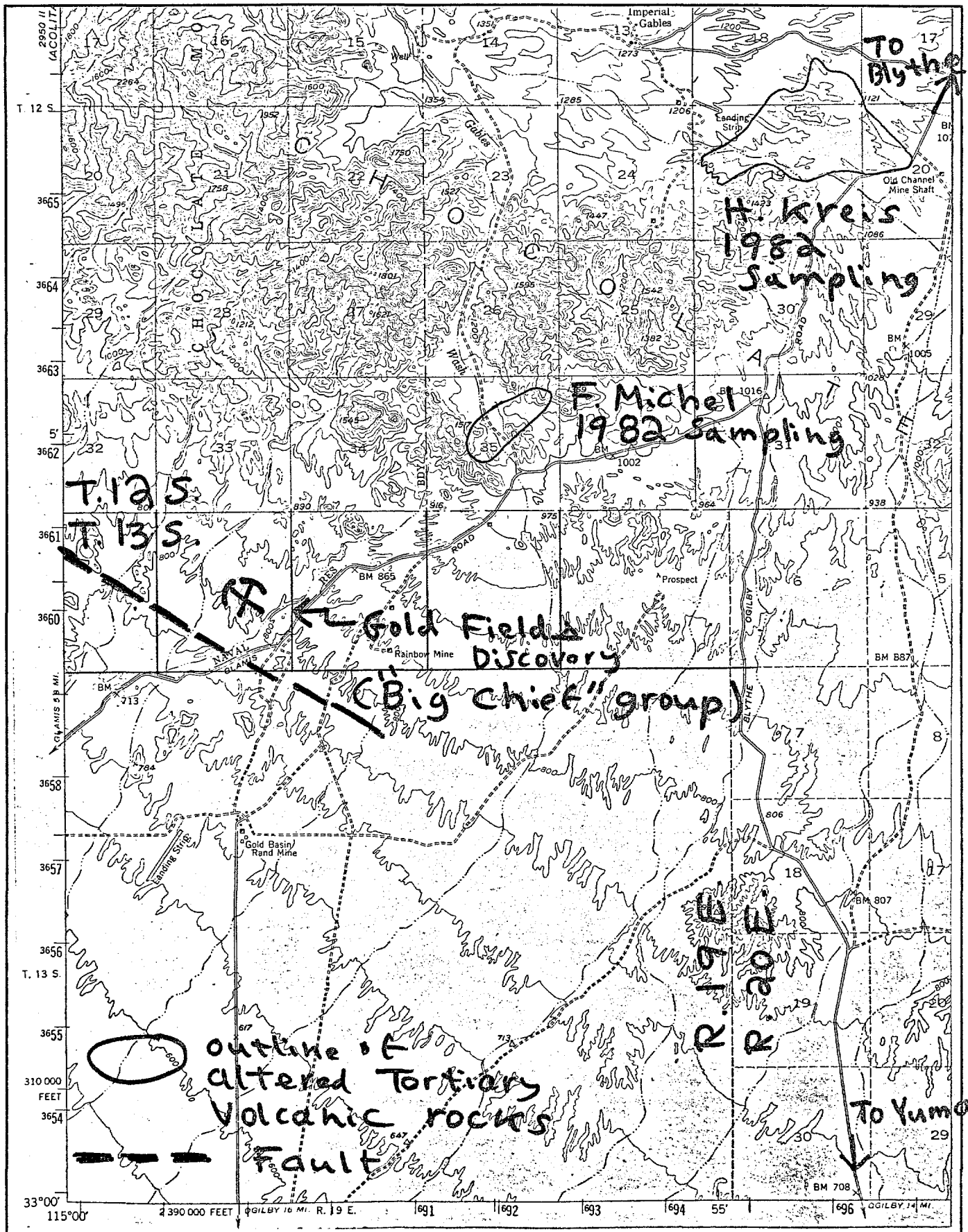
"Depending on the cut-off grade selected, and assuming the mineralization is continuous between holes, the zone may contain between 26 million tons of ore at a rate of .075 ounces per ton and 41.5 million tons at a grade of .056 ounces per ton," Gold Fields said.

The company said it plans additional drilling, bulk sampling, metallurgical testing, hydrologic and environmental studies and a review of the conditions under which operating permits can be obtained. A spokeswoman said those factors would have to be evaluated before the company decides whether the discovery is commercially practical to mine.

Is environment similar to Cargo Muchacho-Mesquite

[London Times reports that a total of 327 shallow holes were drilled by Gold Fields & 165 of these intersected mineralization]* Mineralization in quartz stringers (fault breccia zone ?) in Pre € Chuckwalla Complex gneissic rock. Mesozoic granitic rock nearby. Despite old shafts etc near the Gold Fields disc. site (workings not shown on topos map), the district is an old placer producer with the bulk of the workings south of the Glamis road. One cannot get on Gold Fields discovery site area., posted etc. G.J.S.

* FTG Verbal comm.



ASARCO INCORPORATED

area SW Corner of 15' USGS Quartz Peak, Calif. Topographic Quad.	data by	state Calif.	township-range	revisions-date
		county Imperial	map number	PLATE 1
title Location Map-Gold Fields Mng. Corp.'s 1982 Gold Discovery	drawn by GJS	1 ½ 0 1 2mi		
mining district Mesquite (Chocolate Mtns)	date Nov. 1982	contour interval 40ft.		

mx 2910 dam

Gd & Dile/Heap

Gulch Heap/Bulk

Cole gold
Bulk/Heap

N. Miner, Dec. 16, 1982, p.39

To: WKK/JDS/GHS
Tucson Office

Copy for GHS

Gdel
Bulkhead

Copy WKK gdl

12/23/82

Gold Bulk/Heap

Send to: WLKurtz - Tucson

Gold- Bulk/Heap

Gold Bull / Heap

WHARF RESOURCES LTD.

3 MONTHS ENDED OCT.31,	1982	1981
Gold & Silver Sales	$2,058,309	$ 749,430
Cost of Sales	988,950	498,634
Admin. Expenses	37,250	7,013
Other Income(Expense)	(44,716)	51,850
Cash Flow Fr.Ops.	1,037,385	295,533
Dividend Fr.Pegasus	38,719	-
Deprec'n,Depl'n,Amortiz'n.	15,000	80,041
Income Tax, Deferred	522,481	103,029
Net Income	583,623	112,513
Per Share	6-7/10¢	1-3/5¢
Shares Issued	8,057,050	6,986,258
Working Capital(Deficit)	$(340,502)	$1,348,351

PROFIT QUINTUPLED

On 7Dec82, Wharf Resources Ltd. received approval from the State of South Dakota to develop their Annie Creek gold property. Peter R. Kutney, president, says this large scale open pit mining venture is expected to start operations in 1983 with production of 500,000 tons of ore. On-site pilot leach tests indicate recoveries of 74% of the gold contained in ore crushed to 1½-inch after 50 days of leaching. Initial cash flow from this property is targetted for August, 1983. Based on all drill results to date and detailed pit designs, the open pit ore reserves are 5,251,000 tons proven grading 0.042 ounce gold and 0.15 ounces silver per ton; 1,269,000 tons probable grading 0.045 ounces gold and 0.17 ounces silver per ton; and 2,712,000 tons possible grading 0.013 ounces gold and 0.19 ounces silver per ton. The proven reserves estimate is based on a 45-degree pit slope, using a 0.02 ounces gold per ton cut-off grade. The probable estimate is also based on a 0.02 cut-off. The possible estimate is based on a 0.01 cut-off.

At Landusky, Montana, gold and silver recoveries in the quarter ended Oct.31/82 were exceptionally good with gold exceeding 31,000 ounces on this property for the1982 leach year. During September 1982, the average daily rate of recovery increased to 279 oz. per day. The operator attributes the improved recovery to improved leaching techniques and the addition of some higher grade gold ore to the leach pile. Wharf currently hold 18.4% interest in the mine increasing to 37% net interest after recovery of development and pre-production expenses by the operator. Mr. Kutney expects this recovery will occur during the 1983 leaching operations.

On the Golden Messenger property in Montana, plans for this fiscal year ending 31Jul83 include 4,000 feet of drilling to delineate the Golden Messenger and Little Dandy zones. At Wells, B.C., further testing will be deferred until gold prices increase significantly.

Wharf's sale in November 1982 of their share holdings in Pegasus for $4,140,325 enabled the company to eliminate bank debt and will facilitate development of the Annie Creek property.

OLIVER RESOURCES LTD.

IN 1982 TWELVE WELLS WERE DRILLED IN OHIO - Oliver Resources Ltd. have 25% working interest, 21.88% net revenue interest, in a 450-acre prospect in Wayne county, Ohio, on which the twelfth well, in their 1982 program, the A. Hershberger No.1, has been successfully drilled and cased to total depth at 4100 feet. Director Wayne Babcock,P.Geoph., reports that computerized electric logs run up the hole indicate some 12 feet of net porous gas sand within the target Clinton formation.

Mr. Babcock also reports that a shallow well designed to test the sometimes prolific Ohio shale formation has been drilled and completed to a depth of 2500 feet. Test results will be reported when available. Additional acreage is being acquired and the first 1983 well should start drilling about the third week of January.

RUPERTSLAND RESOURCES CO. LTD.

WELLS IN TEXAS & UTAH REPORTED - Rupertsland Resources Co. Ltd.'s wholly-owned U.S. subsidiary is currently testing its fifth well in the Brushy Creek prospect, West Purt field, Anderson county, Texas, after having set pipe at the 10,500-foot level. President Hoye J.Caine reports that the Rodessa formation has been perforated and acidized and is currently testing at rates up to 1,175,000 cubic feet of gas and 120 barrels of oil per day. This well has proved up at least 2 and possibly 3 additional offset locations.

The company's first 2 wells there are each flowing approximately 300 barrels of oil per day from the same Rodessa formation. Working interest in 3 of the wells, including the current well, is 18.75%. The company's working interest in the 2,000 gross acre field area varies from 3.469% to 28.125%.

The subsidiary has also completed and brought on stream the Uinta basin, Ice Canyon 16-9 well in Grand county, Utah, in which they hold 21.875% working interest.

Mining Journal — February 4, 1983

WCK Gold
Bulk/Heap 71

MINING

xc: PGV 12/4/83

Alk gold
Bulk Heap

New York, February 4, 1983

FILE MEMORANDUM

Pinson and Preble Mines
Humboldt County, Nevada

I was advised yesterday by Don Duncan, a Reno, Nevada gold mining consultant, that the Pinson Mine now has a full scale heap leaching operation in addition to the 1200 tpd mill. Pinson has roughly 5 million tons of heap leach ore (grade was not given) which is too low grade for the mill. Some of the leach material had so much clay that percolation through the heaps was inhibited and mixing of a second, harder, low clay ore type provided the necessary permeability.

The distribution of ore at the Preble deposit is said to be very irregular. Some is of sufficient grade to withstand the 15 mile haul to the Pinson mill, and the remainder will be heap leached at Preble. Duncan said that the Preble mineralization becomes quite carbonaceous at depth.

Original signed by
F. T. Graybeal

F. T. Graybeal

cc: WLKurtz/JDSell
JCBalla
JDSmith, Jr.
PGVikre
DRCrowell

RECEIVED
FEB 7 1983
EXPLORATION DEPARTMENT

10/12/77
Lucky Gold Bulk/Heap

Sunbeam Gold
Idaho

Visit with Bulla and prospector.

Volcanics, mainly rhyolitic-qtz latitic composition; mostly tuffaceous looking.

Gold quite well restricted to silicified zones

Outside of mineralized zone are volcaniclastics with flow fragments

WLK file

RECEIVED
NOV 6 - 1978
EXPLORATION DEPT.

The Northern Miner

CANADA'S MINERAL RESOURCES NEWSPAPER

OCTOBER 26, 1978 VOL. 64 NO. 33

GOLD

over. ½ gold ½ Ag

[illegible] on [illegible]

Ruby [illegible] 2000' long 200' wide
300' deep [illegible]

[illegible] already [illegible] gold [illegible] telluride

some evidence of Lucky of silver
to on [illegible]

Zortman – higher grade [illegible] was
at [illegible] intersection.

Landusky [illegible]
[illegible] deep

[illegible]

Leroy Swanson – Geol Aug 18, 1990
Frank Schultz

April – Sept. leaching
125 meme [illegible] (inside [illegible])

.203 / ton sol. prog.
.08

gasses off
zinc dust

65% re. gold 35% silver
500 gallon/minute / 300 average!

1.5 cyl. / ton of solution
200,000 plant

6" bentonite under pad

3" 5' ; 6" 8' 40' bench

file Gold, teaching

Landusky 500,000 last year
1 million this yr

Zortman ~~[illegible]~~
~~[illegible]~~

9 smaller residency

capitation for one treatment < 300,000

NOV 14, 1980

From: R.E. GALE

To: BILL KURTZ, TUCSON

FURTHER DATA ON
GOLD LEACH PROPERTIES
IN ARIZONA (AND CALIFORNIA)
?

REGARDS,
Bob.

RECEIVED
NOV 20 1980
EXPLORATION DEPARTMENT

Wright Gold-heap leach

GOLD HEAP LEACH PROJECTS IN PHOENIX AREA, ARIZONA

A NUMBER OF VANCOUVER COMPANIES ARE ACTIVE IN HEAP LEACH PROJECTS -- IN THE AREA OF PHOENIX, ARIZONA

One of the first developments which attracted people to Phoenix, Arizona about 100 years ago was gold and silver exploration followed by large copper discoveries. With the current gold prices, all of the old properties are being examined for large tonnage low grade 'no-see-em' gold potential. It is not only the old properties which are being explored but also areas of favourable geology where new techniques are locating areas of interest. A number of Vancouver Stock Exchange funded companies are active in this area. The properties of four of the Vancouver companies were visited recently. The stories follow: Amca Industries Ltd., Copper Lake Explorations Ltd., Ark Energy Ltd. and Phoenix Electric Oil Refiners Canada Ltd.

AMCA INDUSTRIES LTD.

GOLD RECOVERY AT SILVER CROSS PROPERTY CONTINUING WITH SECOND HEAP BEING BUILT

- Charles Martin, president of Amca Industries Ltd. pointed out that the Silver Cross gold-silver property, 40% owned by the company, and 50% by a number of Phoenix, Arizona, investors has limited known reserves and a reasonably good chance for extensions. What the property has done and will continue to do for Amca includes: providing all of its money back in a short period; promising a good chance for extended life with expanded reserves; and providing very valuable heap leaching operating experience. What Mr.Martin points out is that, "We are now in production. We have done it. We now know how to do it and what to look for in other properties where the same method of gold recovery can be profitable over a period since reserves are greater." Many such properties with potential have been examined by Amca and several are under negotiation. The operations of Amca Industries were recently reviewed in GCNL No.203(1980).

The 102-claim Silver Cross property is eight miles northeast of Cave Creek, 20 miles north of Phoenix, Arizona. It covers an area of mineralization estimated at 30,000 tons visible and a further 120,000 to 150,000 tons possible if the increasing width, seen in the pit, is maintained or increased as the mining continues. In mining out the approximate 15,000 tons to date, the width of the structure has expanded from a few feet to in excess of 20 feet in the bottom of the pit. Since May 1,1980 the company has mined and heaped 10,000 tons with an estimated grade of 0.34 oz.gold/t. To date the company has recovered 850 oz. of gold, 2,400 oz. of silver and leaching is continuing. None of the first 10,000 tons had been crushed. Now, all of the material in the second heap will be crushed. The sprinkling method in the first heap led to puddling and some channeling in the heap. A modified sprinkling method on the second heap appears to have eliminated the puddling and the crushing, plus loading the heap with front end loaders rather than with trucks, has reduced the chances of channeling. The Silver Cross property, unlike many in Arizona, is blessed with not only a spring but also a small year round creek. The company has gone to considerable effort to insure that none of the solution should escape into the local stream. The heap pad is 600 feet by 200 feet and able to hold 40,000 tons of material. In addition to using 30 mill plastic, which was factory welded, the company has used a ¼ inch thick blanket made of fiberglass-plastic called Bedem which is expected to prevent puncturing of the plastic waterproof layer. On the Bedem is laid three-inch perforated pipe on top of two feet of crushed rock. The Silver Cross is a compact little operation pumping 60 to 70 gallons per minute to the sprinkling heads and the same to the four carbon gold recovery towers and stripping tank.

With the building of the second heap, the company is increasing its pumping capacity to 150 gallons per minute. The company's objective is to have 60,000 tons on the heaps in six to eight weeks. With the 700 tons per day crushing capacity, bulldozers, trucks making the 200-yard haul from the pit to the heap pad, the chances of meeting the objective appear to be good. An atomic absorption assay unit is installed at the plant to assist in the daily control of grades at various locations in the operation.

Mr.Martin pointed out that an application for a Toronto Stock Exchange listing is well advanced as is a NASDAQ listing. As these proceed the company is continuing with negotiations to acquire several other properties, one of which has very large potential gold reserves. In the meantime, the Silver Cross is generating a cash flow and the company is gaining valuable experience.

COPPER LAKE EXPLORATIONS LTD.

LARGEST HEAP LEACH PLANT UNDER CONSTRUCTION AT THE PHOENIX, ARIZONA GOLD SILVER PROJECT

- Gold recovery will start Dec.15 at the Clementine gold-silver property of Copper Lake Explorations Ltd. near Phoenix, Arizona, if the present construction schedule is met. All present indications are that, with the capital available, the mining machinery now at work on the property, all the necessary gold recovery plant components in place, the crews available that the schedule, while tight in a number of respects, will be met on the projected dates.

On Nov.4,1980 crews and machinery were grading, leveling, slopping the foundation for the heap pad which will hold the first 100,000 tons of gold mineralization for the test leach. The pad is to be 500 feet by 300 feet. As the next few weeks pass the foundation will be completed, covered with the compacted gravel, paved with asphalt, plastic will be laid down, recovery pipes put in place. Then the ore heaped on the pad and sprinkling with dilute cyanide solution will start. (CONTINUED ON PAGE THREE)

NUGOLD ENTERPRISES CORP.

LARGE GOLD PROSPECT STAKED IN QUEBEC - M.G.Berretta, president, reports that, some 60 miles SW of Val d'Or, Quebec, Nugold Enterprises Corporation have acquired, subject to regulatory approval, 581 claims and 101 lots in Blondeau, Latulipe and Gaboury townships.

This "Belleterre" property is underlain by granodiorites similar to those hosting the Belmoral and Bras d'Or deposits. To date, this ground has been relatively unexplored. Preliminary prospecting, done during the claim staking, detected the presence of several shear zones as well as gold values in the range of 0.02 to 0.05 oz/ton. A major program consisting of airborne geophysics and geological mapping is to start soon.

SUNSHINE COLUMBIA RESOURCES LTD.

ENCOURAGING GOLD ASSAYS RECEIVED - George G.Alce, secretary, reports that Sunshine Columbia Resources Ltd. have received the assay results from 5 diamond drill holes, designed to intersect No.4 vein below the lowest working level, No.10, on the company's property near Camborne, SE B.C. Holes 1 and 3 intersected the vein 50 feet below 10 Level and 2 and 4 intersected 110 feet below 10 Level. Hole 5 was 85 feet below the level and 100 feet south of the Hole 1 intersection. The true width of the vein is between 16 and 18 feet.

Hole No.	Intercept Width	Lead %	Zinc %	Silver Oz/Ton	Gold Oz/Ton
1	18.0 feet	5.02	3.06	6.01	0.20
2	17.5	4.45	4.25	4.56	0.067
3	18.5	3.47	3.27	4.00	0.146
4	23.0	9.52	7.37	10.82	0.124
5	18.0	8.18	7.65	8.39	0.10

GOLD ANGEL RESOURCES INC. GILLIAN MINES LTD.

OPTION ON OIL/GAS LANDS IN S. AFRICA CONFIRMED - Gillian Mines Ltd. president J.Paul Stevenson and Gold Angel Resources Inc. president John Plourde report receiving confirmation that the companies have the option on an initial 225 square kilometers (55,598.4 acres) of oil/gas leases in the Clocolan district of South Africa, being part of 225,000 km^2 they are examining there. The initial 225 km^2 are in a region noted for oil seeps and gas flares. A standard lease is to be effected with an eighth interest ownership maintained in South Africa. There is to be full repatriation of profits and no duty charges on equipment brought in to South Africa. All agreements will be subject to regulatory approval.

COPPER LAKE EXPLORATIONS LTD.

CONTINUED FROM PAGE TWO - The roughed out contours for the three solution ponds(barren solution of 6,000,000 gallons, pregnant solution of 6,000,000 gallons and make-up water of 2,000,000 gallons) were complete ready for final shaping and installation of water proof lining. The three trailers containing the filters, zinc presses and gold recovery units are in place and the piping to interconnect various phases of the concentrating plant is underway.

The Clementine (formerly called the Charlotte)gold silver property includes 53 unpatented claims and 48 placer leases located on White or Pikes Peak, six miles north of Sun City, 30 miles north west of Phoenix, in Maricopa county, Arizona. The property is on the southeast slope of Hieroglyphic mountain and has been held by the Triphahn family since 1935. Over the years the showings have been explored by various groups including work by Copper Lake ten years ago. In the intervening years the prices for gold have changed and the heap leach techniques have improved.

The first 100,000 tons to be leached have an estimated gold content of 0.06 to 0.10 oz./t, plus 0.06 to 0.5 oz.silver/t. Mining will start in about two weeks in a trench from which a bulk sample graded 0.08 oz.gold/t for its 150-foot length. The face of this trench grades 0.11 oz.gold/t across 10 feet.

There are a number of other trenches, drill holes and a 120-foot deep shaft where values range from 0.01 to 0.02 oz.gold/t. One of the mineralized zones on the property has been mapped as 400 feet wide and 2,500 feet long indicating a possible 100,000 tons per vertical foot. A further zone is 150 feet wide and maybe 5,000 feet long for a very large tonnage potential if grades are established. The narrow veins within the zones carry better values which may improve the overall grade. With the sampling problems on this type of property,gold recovery can exceed the ore grade as calculated. The limited metallurgical work to date indicates an 85% recovery and,with indicated operating costs of $6.00 per ton,the 0.02 oz.gold/t grade should be profitable. A breakeven would be in the range of 50% recovery of the 0.02 oz./t.

One of the items to be learned from the test heap is whether or not the ore will have to be crushed. The first 100,000 tons are not to be crushed. An area near the pad has been selected as the logical crusher site should one be necessary.

Some of the unique features of the leach plant were pointed out by Carl Toporowski and his son,John, who hold a contract for plant installation. The Toporowskis have built a reputation in the gold heap leach recovery business since they have been involved in the construction of five similar plants in recent years. The most successful of these being the Pegasus-Argo-Wharf operation in Montana. They point out that,when completed,the Copper Lake plant will have the largest liquid handling capacity in North America. It will have a rated capacity of 850 gallons per minute,that is 5,000 tons of mill solution per day. With minor modifications this can be increased to 1,200 to 1,500 gallons per minute both through the sprinklers and the six zinc recovery presses.

VANCOUVER STOCK EXCHANGE - SHORT POSITION REPORT - NOVEMBER 7, 1980

+ Reported Short Positions of Approximately 10,000 and More Shares

Amca Indus.Ltd.	30,685	Double Eagle Energy & Resources	10,500	New Frontier	16,100	Taseko Mines	9,000
Aurus Mining	10,500	General Energy	12,050	Prism Resources	9,400	Thunderwood Explorations	22,400
Belmoral Mines	13,750	New Cinch Uran.	24,800	Stampede International Resources	10,700	Tri-Basin Resources	17,600
Copper Lake Explorations Ltd.	12,000						

ARK ENERGY LTD.

HEAP LEACH PROGRAM PLANNED FOR EARLY IN 1981- AS SAMPLE - Harry Williams, president of Ark Energy Ltd. has reported that the widespread mineralization on the Golden Ark property has proved difficult to sample. To overcome this problem the company will do a test heap leach of between 10,000 and 15,000 tons from several areas, first, to determine the grade and, second, because it will probably be highly profitable even on a test basis. The heap will be built in the next two months with the first gold recovery expected in late January 1981. The construction and program will be under the direction of S.B.Reamsbottom, Ph.D., who gained valuable experience working on the Pegasus-Argo project.

The Golden Ark 195-claim property is three miles off of the highway, 50 miles north of Wickenburg , 7 miles southeast of the famous Cyprus Bagdad copper-gold 50,000 ton per day open pit. (Capacity is currently being increased to 100,000 tons per day.) It is 120 miles north of Phoenix, in Yavapai county, Arizona. The fine, free gold, some visible gold, on the property is in fractures of brown oxidized hemitite in granites intruded by pegmitites. The testing by extensive trenches, pits, three adits and several short shafts indicate oxidation to 200 feet and tonnage available to open pit mining. A small production a few years ago demonstrated that satisfactory gold recovery may be achieved by heap leaching. Sampling of the showings by Paul Sawyer, P.Eng., returned a high assay of 2.25 oz.gold/t and an average of 0.3 oz.gold/t while sampling by Stan Reamsbottom returned 0.15 oz.gold/t. Recent drilling planned as 5,000 feet of percussion hole at 40 locations has returned what is considered unrepresentatively low 0.02 and 0.03 oz.gold/t in areas where surface sampling has yielded 0.2 oz.gold/t. Sampling in one area returned 2.5 oz.silver/t.

A soil sampling program located four gold and mercury anomalies, each about 1,500 feet long and from 1,000 to 1,200 feet wide. Further exploration in these areas is planned.

Dr.Reamsbottom said, "The widespread distribution, grade and possible leachability of gold mineralization within the Golden Ark property, makes it an attractive target for the development of a large gold deposit which may be mined by open pit techniques and the gold extracted by a low cost heap leach process."

As the drill sampling program continues, the company plans to proceed with the engineering and installation of a small tonnage heap leach program."When we have a handle on the tonnage and grade as well as the recoverability of the gold then the logical thing would be to expand the leaching operation as indicated by the reserve numbers," Mr.Williams said.

Ark Energy has interests in several other companies and properties where exploration is either underway or planned for the near future.(See GCNL No.198, page two, for recent assay results from the percussion drilling program on the Golden Ark property.)

PHOENIX ELECTRIC OIL REFINERS CANADA LTD.

DECISION TAKEN TO PROCEED WITH HEAP LEACH TEST - The Kellwebb gold property located 58 miles north of Phoenix, Arizona and under option to Phoenix Electric Oil Refiners Canada Ltd. was reported on in some detail in GCNL No.205, page three, Oct.23,1980. The story dealt with the probable solution to the assaying difficulty encountered in duplication of the values recovered in earlier production. The property has been tested, in addition to various stages of production since 1880, by several trenches and open pits on each of a number of mineralized structures. These trenches have established a number of strong structures each with substantial dimensions and mineralized with varying amounts of gold and silver. As pointed out in the previous story, the values as calculated are most encouraging. When and if substantiated, these tonnages and grade could support a large tonnage heap leach operation. Machinery and equipment now on the property are working on the water supply system, continuing trenching and making early preparation for the installation of the leach plant which is now being assembled in Oregon. The plant will be shipped to the Arizona property and assembled as soon as possible, probably later this year or early in 1981.

As this is underway the company management has been examining a number of other properties in the general area for possible acquisition.

FOR THE RECORD

Gallant Gold Mines Ltd., by Oct.15,1980 agreement effective Nov.12,1980 will offer within ten days 300,000 units comprising one share and 2 series A warrants on Vancouver Curb Exchange at a price yet to be fixed but not less than $3.00 per unit through agents, Canarim Investment Corporation Ltd., as to 80% and McDermid, Miller & McDermid Ltd. as to 20%. Each two series A warrants will be entitled to purchase one further share at a price yet to be fixed until May 11,1980. Directors Frank Lang, R.W.Hughes and R.C.Brown have been granted options on 35,500 shares each exercisable at prices of $4.30 to Oct.7,1981, or $4.55 until Oct.7,1982.

AMCA INDUSTRIES LTD.

SECOND HEAPS IN PREPARATION FOR GOLD SILVER LEACHING - Charles Martin, president of Amca Industries Ltd. has reported that progress is being made in the preparation of the second phase of heap leaching for gold and silver on the Silver Cross Industries property located in Maricopa county, 30 miles northeast of Phoenix, Arizona.

Crushing facilities have been installed and three heaps of 10,000 tons each are being built. Production from the test heap of 10,000 tons of uncrushed rock is approximately 850 oz. gold, 2,000 oz.silver, valued at $460,750. The heap is still producing and there is no way at this time of estimating total recovery from this heap. The crushed rock in the new heaps is expected to speed the gold and silver recovery. Amco has a 40% property interest.

Amca has a 30% interest in a small oil well now on pump test in Clark A lease, Young county, Texas. Other oil and gas leases are being examined toward possible participation.

The company is also in negotiation with Mineral Economics Inc. toward participation in several gold properties in Arizona with large tonnage low grade 'no-see-em' gold potential.

Directors of the company recently exercised their stock options on 90,000 shares at $2.11 each to provide the company with $189,900 in new funds.

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EXPLORATION DEPARTMENT

AARON MINING LTD.

DRILL PROGRAM UNDERWAY TO ESTABLISH GRADES AT NEVADA GOLD HEAP LEACH PROJECT - Andrew Robertson, chairman of Aaron Mining Ltd. has reported that to date 33 holes have been drilled for a total of 11,432 feet to generate 2,000 samples in the current program directed to establish grades on the gold property at Crescent Valley, Nevada. Final assessment of the results will await assay results which are expected to be slow in coming since the laboratories are overloaded.

He said that the gold and silver recovered from the leach pad has not yet been reduced to bullion. It is now underway. Until now, because of a limited water supply, the leach process was conducted on a small portion of the pad only. The deep water well recently brought on stream will greatly increase the leaching capabilities at the property. Extensive testing of the leach materials already on pads is being carried out and a decision as to when the leaching process will resume will be based on the results. During the change-over in water supply, the solution storage ponds and circuit were checked and upgraded. The recovery of the gold from the precipitation towers and any further leaching are to be carried out by Miller-Kappes Company, mineral management consultants of Reno,Nevada.

Aaron Mining recently sold 44,445 treasury shares to E & B Explorations Inc. at $4.50 per share. E & B have agreed to spend $250,000 U.S. on exploration of the Gold Pan and Altenberg claims, Lander county, Nevada, by Nov.15,1980. When the work is completed, E & B may either turn over the results without earning an interest or they may earn 25% interest in the property by spending a further $250,000 U.S. in exploration by Feb.15,1981 and providing Aaron with a loan related to Aaron's working capital deficit with respect to the property, being up to $400,000 U.S. interest free, plus, if needed, up to $200,000 more repayable from 75% of first net proceeds with interest at prime plus 2%.

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June 1993

HEAP LEACH PROJECT EVALUATION DURING THE INITIAL STAGES

PROGRAM FOR EVALUATION OF THE OREBODY

The first step in designing a more complete program prior to testing the orebody is to have a conference between the field geologist and the metallurgists who are designing the test program to determine such things as:

- availability of core samples for testing deep-level ore
- accessibility for near-surface and deep-level bulk samples
- variability of ore and ability to geologically identify tonnage/geological parameters of the various ore types

INITIAL EXAMINATION

A preliminary test program designed to tell quickly if an ore is heap leachable or not should be limited to a few column leach tests. These tests are normally inexpensive.

For a low-grade hard (non-clayey) rock, a good first test would be a column leach test at an intermediate crushed size, such as one inch. Normally at least two samples from the orebody should be tested, and maybe more if there is significant variation in rock type. The best samples for evaluation are bulk samples composed of fairly large rocks.

A head sample of the column material, and the test tailings would be screened into five or six size fractions and then assayed to determine if there is an approximate relation between recovery and crushed size. For accurate percent recovery calculations normally 4 to 12 assays would be run on tailing samples.

If the 30-day percent recoveries are low on a column test, it is sometimes advisable to suspend the test, air dry the sample, crush to a finer size, and then restart the test.

The test program should include a cyanide solubility test on an agitated portion of the pulverized head sample so total cyanide soluble gold levels can be used to monitor the progress of the column test.

Preliminary data (such as estimated gold recovery based on solution assays) will typically become available from the column test about three weeks after the sample arrives at the laboratory. Tests should normally run 60 days, so final information will be available three months after the sample arrives at the lab and the final report will be issued a month later.

The sample size placed in the leach column would range from 50 to 150 pounds, and would generally be tested in a column 6 to 12 inches in diameter and five to six feet in height. The total cost of the test program including the report should be about $2000.00 per test. Involved sample preparation procedures would increase the test cost.

STEPS FOR A GENERAL TESTING PROGRAM

1. Determine Ore Types. In conjunction with the geologists, review rotary drillhole chip logs, and/or core photographs to try to determine important metallurgical ore types.

2. Determine Distribution of Cyanide Solubility of Gold Within the Orebody. Run a series of cyanide solubility tests (centrifuge tube tests) on all 5-ft (or appropriate) intervals of the rotary drillhole samples,

to determine uniformity of cyanide response. These tests are usually run on the same pulverized pulps which were used for fire assay of the drillhole intervals. Each test will cost about $5.00 to $6.00.

A typical major orebody might require 500 such tests. The tests show recovery trends throughout the orebody, and can identify zones with special metallurgical problems. They are used as a guide to further testing and to selection of areas for bulk sampling.

3. Run Column Leach Tests. On bulk samples taken from underground and surface exposures, or if necessary from drill cores, a series of laboratory column leach tests at various crushed sizes would be run. The preliminary tests mentioned above would provide one data point; typically, the ore may be tested at three crushed sizes: 3, 1, and 1/2 inches. The column test program on each ore type would cost approximately $ 6,000.00. Simple small orebodies might end up with two ore types: complicated large orebodies might involve four to six sets of such tests.

   This outline of column leach tests assumes the ore is hard and clay-free. The test program can be modified for clayey ores - for instance, testing at various crushed sizes is not so important as testing using agglomerated samples.

   The general laboratory small column test program for a large orebody can therefore range between $ 12,000.00 and $ 40,000.00. A few progress meetings and test reviews are normally included as part of the cost; sample procurement costs, feasibility studies or extended scoping meetings, and client oversight costs are not included in that estimate.

4. Need for Field Leach Test. Proceeding beyond the small column test phase is a function of both technical and managerial requirements. For hard, chemically inactive, simple ores that are going to be crushed before leaching, it is technically possible to proceed directly from the lab tests to production heaps, and this is normally done on small ore bodies.

   Large projects, especially on marginal grade ores which contain actively oxidizing sulfides and possibly copper minerals which may lead to high cyanide consumption, usually require scale up tests to reduce the degree of

risk. A field test program consisting of one to three 2,000 to 3,000-ton field test heaps may be necessary. The cost for an individual test at a remote location with no existing support facilities ranges from $200,000.00 to $300,000.00, inclusive of mining the sample in a small open pit, conducting the test for 60 days, and all final shutdown and report costs.

5. Design/Basic Cost Study. Simultaneous with or subsequent to the field test, a pre-feasibility study can be developed which contains the basic design and cost elements (or alternatives). For orebodies up to 5,000 tons per day such studies take about two months and cost about $ 25,000.00.

6. Final Engineering/Design Study and Construction. The pre-feasibility study is generally sufficient to be accepted without a final full-blown feasibility study. Design and construction can then proceed.

SCALEUP EXPERIENCE IN GOLD-SILVER HEAP LEACHING

A Talk for Presentation to

the

AIME Annual Meeting
Chicago, Illinois

February, 1981

by

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SCALEUP EXPERIENCE IN GOLD-SILVER HEAP LEACHING

By
Daniel W. Kappes

A precious metals heap leach is a fairly straightforward mechanism. Ore is stacked on an impervious pad and is sprinkled with a pH-controlled cyanide solution. The solution dissolves gold (and/or silver) from the rock and carries it down and out of the heap. Gold is removed from the solution and the solution is recycled.

The prediction of field heap behavior from laboratory tests should be relatively straightforward, provided that the laboratory test procedures are faithful mini-heap leaches. Detailed comparisons of field heap behavior versus lab behavior now exist for several ores. They indicate that field heap performance differs significantly, but predictably, from the laboratory tests. Specifically, field heaps reach the same level of recovery as the lab tests, but they take from three to six times as long to get there.

The data suggests that all production heap leach systems should be designed with flexibility to extend leach times for as long as the ore requires. Fortunately, Aesop's lesson that slow is not necessarily bad, is one of the guiding principles of heap leaching. The capability for unlimited leach times is inherent in the heap leaching process, both technically and economically. Surprisingly, this capability has been "designed out" of several operations in which the ore is stacked, leached, and removed on regular intervals. The data presented here,

and evidence in the literature from other producing heap leaches such as Cortez and Round Mountain,[1] suggest that eliminating this flexibility greatly increases the investment risk.

Example 1. - The Extremely Long Delay: A Clayey Ore

Figure 1 shows recovery curves in lab and field tests for two ores. Both ores leach quickly in the lab; but in the field they are at the extremes of heap behavior - one leaches very fast, the other very slow.

The laboratory test curves in Figure 1 were developed in typical small laboratory bucket leach tests, containing 20 Kg of ore in a 34 cm deep and 28 cm diameter column. A drawing of the test layout and details of the test procedure have been published previously.[2]

Ore A is a soft, almost pure kaolinite clay containing fine gold. Ore "lumps" are stable and permeable, but ore "fines" can form clay layers that are almost impermeable. To minimize the fines problem, the ore for the field heap was carefully blasted and was not crushed before leaching. The field heap was intentionally built in a manner that would result in delayed, steady gold recovery by using a conveyor stacker to construct the heap as a series of 1 meter high cones. Within each cone, the ore fines created an almost stagnant "pocket" about 0.6 meters in diameter. Around each cone the "throwing" action of the conveyor stacker created a layer of coarse rock which allowed free-flow of solutions downward through the heap.

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1. For a good general article and bibliography on heap leaching, see "Design Factors for Heap Leaching Operation" by George M. Potter, Mining Engineering, March 1981, pp 277 - 281.

2. "Leaching of Small Gold and Silver Deposits" by D.W. Kappes, pp 381 - 388, in THE FUTURE OF SMALL SCALE MINING, edited by R.F. Meyer and J.F. Carmen, McGraw-Hill Publishing Co., 1981, 50 pp. Copies available from the author.

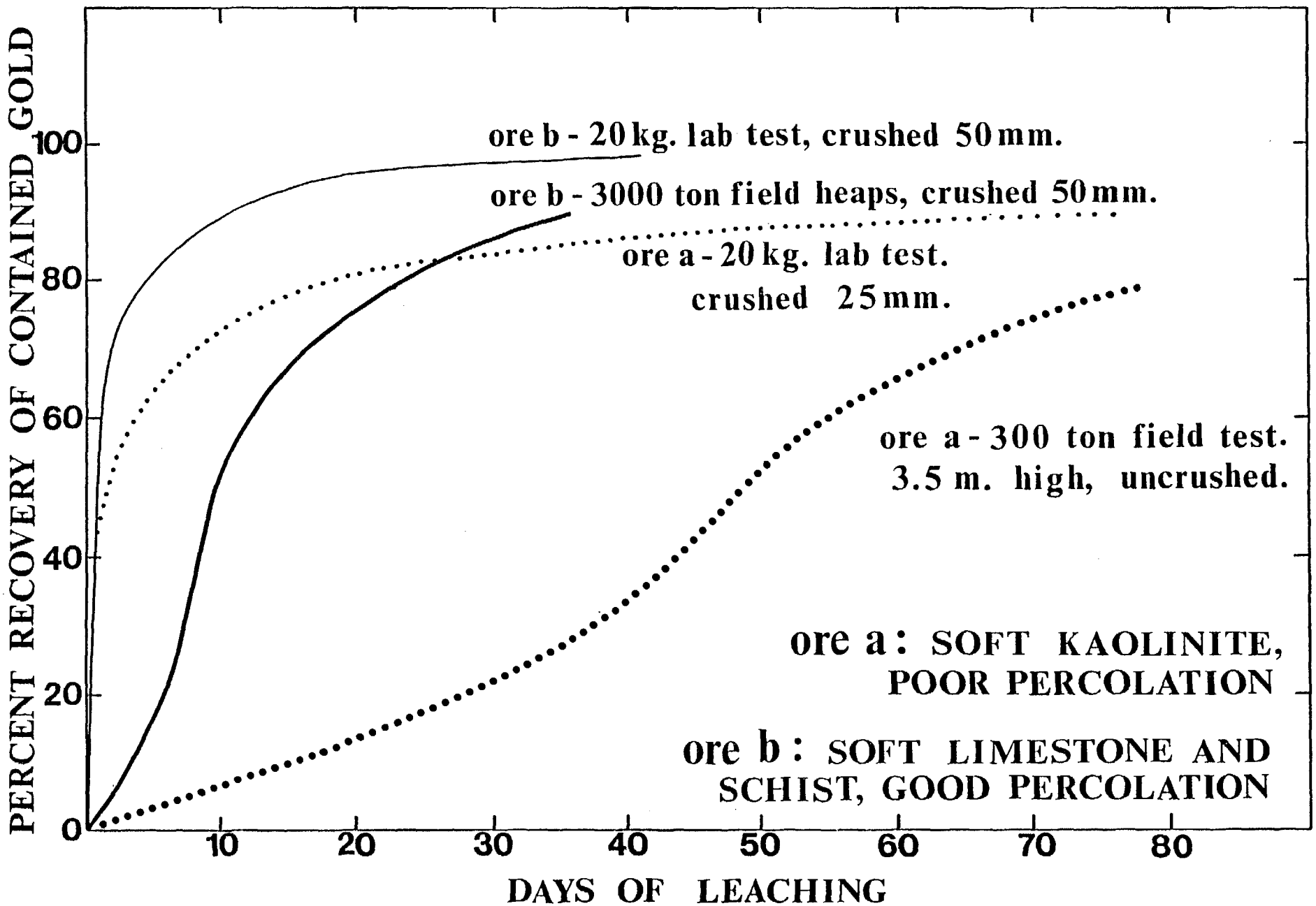


Figure 1: Comparison between lab and field leaches (two ores).

As the curves for ore A in Figure 1 show, 60 percent recovery was achieved in lab tests in ten days, whereas the same recovery took 55 days in the field. It is not too surprising that the field and lab recovery curves for this ore are different since the tests differed in two significant ways:

1. There is no way to take an accurate sample of run-of-mine sized rock for small lab tests (these tests were run on ore crushed to 50 mm).

2. The throwing and segregating actions of the conveyor could not be duplicated in the lab.

The first factor appears to have been not too significant for this ore. Large production heaps have since been run on crushed ore with recovery curves that duplicate that from the uncrushed field test heap. The controlling factor appears to be heap permeability, not rock size.

The second factor is part of a problem common to all laboratory test columns, which is, that there is no way to duplicate the "structure" that is built into the field heap. Ore cannot be compacted the same way it is in the field, and ore segregation, which results from rocks rolling down the heap slopes, cannot be duplicated.

Example 2. - The Short Delay

Ore B in Figure 1 is a mixture of permeable limestone and shale. Both the laboratory tests, and the field heap, were run on coarse-crushed ore.

The ore is hard enough so that ore fines do not clog the heap. As a result, it was possible to build the field heap as a vertical "stack"

of 1.9 meter lifts, to a final height of 3.9 meters, using trucks which drove directly on top of each ore lift.

Laboratory tests were run in short bucket tests and in 2.5 meter high columns, and no difference in recovery times was noted. As in all other tests reported here, flowrates of leach solution in the lab and field tests were roughly identical.

Like ore A, the laboratory tests on ore B appeared to be faithful "mini-heaps", except for one aspect: the important, unduplicatable factor of heap structure. As Figure 1 shows, the laboratory tests achieved 85 percent recovery in eight days. The field heap reached this level of recovery in thirty days. Subsequent production heaps have all behaved identically to the first heap.

Example 3. Ores That Require Crushing

The ore presented in Figure 2 is physically very different from either of the ores in Figure 1. This ore is a hard chert or jasperoid, in which very fine gold is apparently well disseminated. Recovery of the gold in laboratory tests is a straightforward function of the size to which the rock is crushed.

The field test curve shown in Figure 2 was generated from a 3.5 meter high field heap, with rock crushed to 8 mm. The ore in this heap was stacked using a conveyor stacker. There was no clay in the ore and no percolation problems could be detected in the field or in tall-column laboratory percolation tests.

As Figure 2 shows, the ultimate field recovery exactly matched the recoveries predicted from 13 cm high laboratory leach columns. However, as with ore B in Figure 1, there was a significant delay in field recoveries. Fifty percent recovery was achieved in eight days in the lab, versus twenty-seven days in the field.

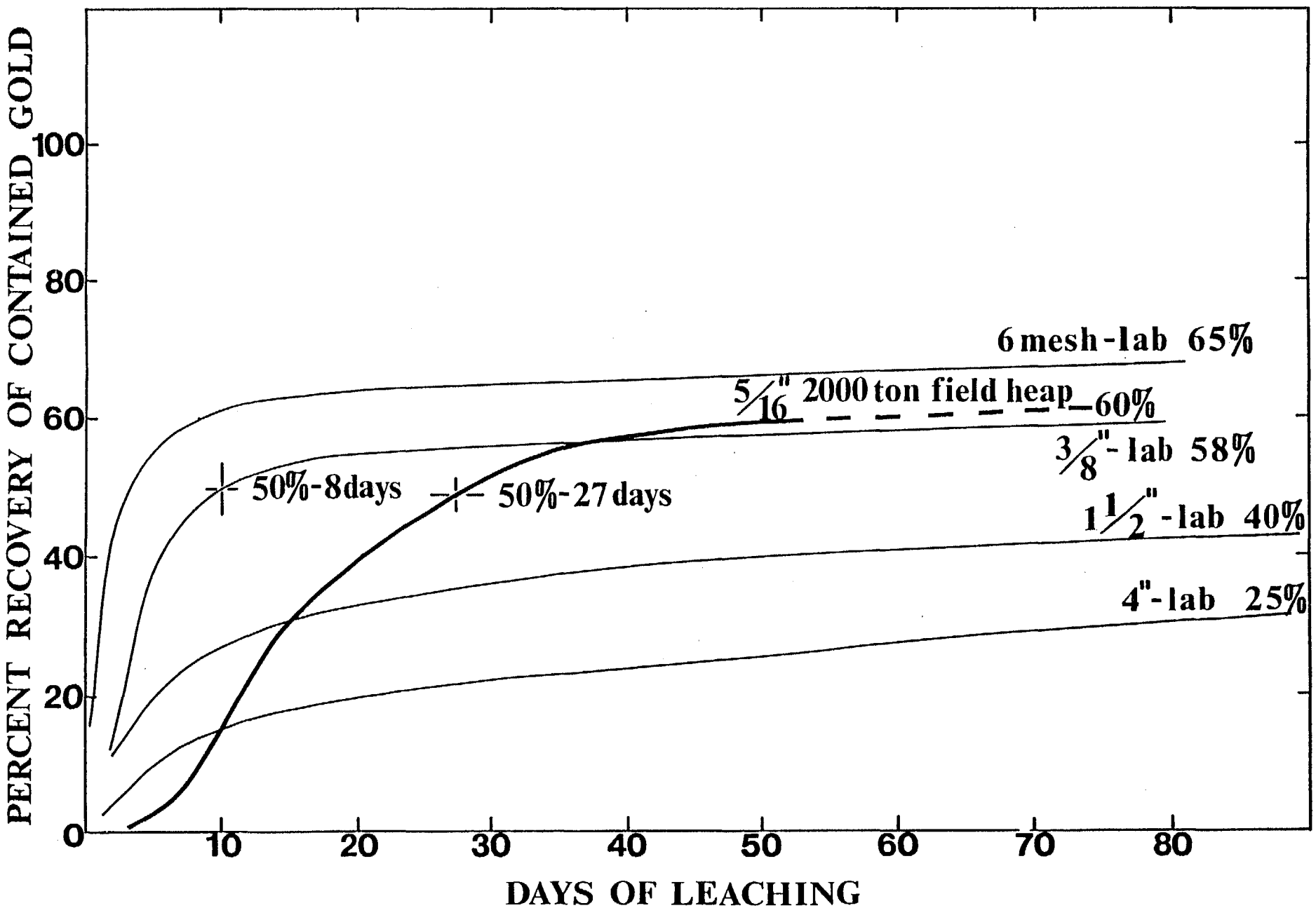


Figure 2: Comparison between lab and field leaches, crushed ore - no percolation problems

The field delay appears to be a function of heap structure, not of heap height. This same ore was leached in a 6 meter high, 0.6 meter diameter laboratory column located at the USBM Reno Metallurgical Research Center. The recovery curves closely matched the small bucket tests rather than the field leaches.

Figure 3 shows a set of four gigantic columns, measuring 12.3 meters high, 1.25 meters diameter. The ore in these tests is "run-of-mine" size volcanic rock, with gold on fracture surfaces. The correlation for both ultimate recoveries and recovery times, between 13 cm high lab tests and these tall columns, has been very close.

Example 4. - Flowrate and Rock Size Variations

It is easy, in a paper such as this, to present only the data which clearly makes the author's point. The ore in Figure 4 tends to muddle things a bit.

This ore is a soft porous limestone. It is physically similar to ore B in Figure 1, but it breaks differently. Simple bulldozer ripping results in "run-of-mine" ore which is 90 percent smaller than 1-inch. Because of its fine, earthy appearance, the heap was stacked (with ripped rock) using a conveyor stacker, in 1 meter high cones, to a final depth of 3.5 meters. Subsequent laboratory and field percolation tests showed that percolation rates were very satisfactory.

Three laboratory curves are shown on Figure 4. The effect of drastically lowering the flowrate had only a minor effect on recovery. Selected large rocks leached slowly and not very well, but the ore in the field heap was mostly small. The "prediction" from the lab tests would be for a 60 percent recovery after approximately 12 days of leaching. It took 50 days of leaching to achieve the same recovery from the field heap.

FIGURE 3. Tall leach columns can provide data on scale-up behavior of chemically-reactive ores. These four leach columns, four stories high, hold 25 tons of ore each.

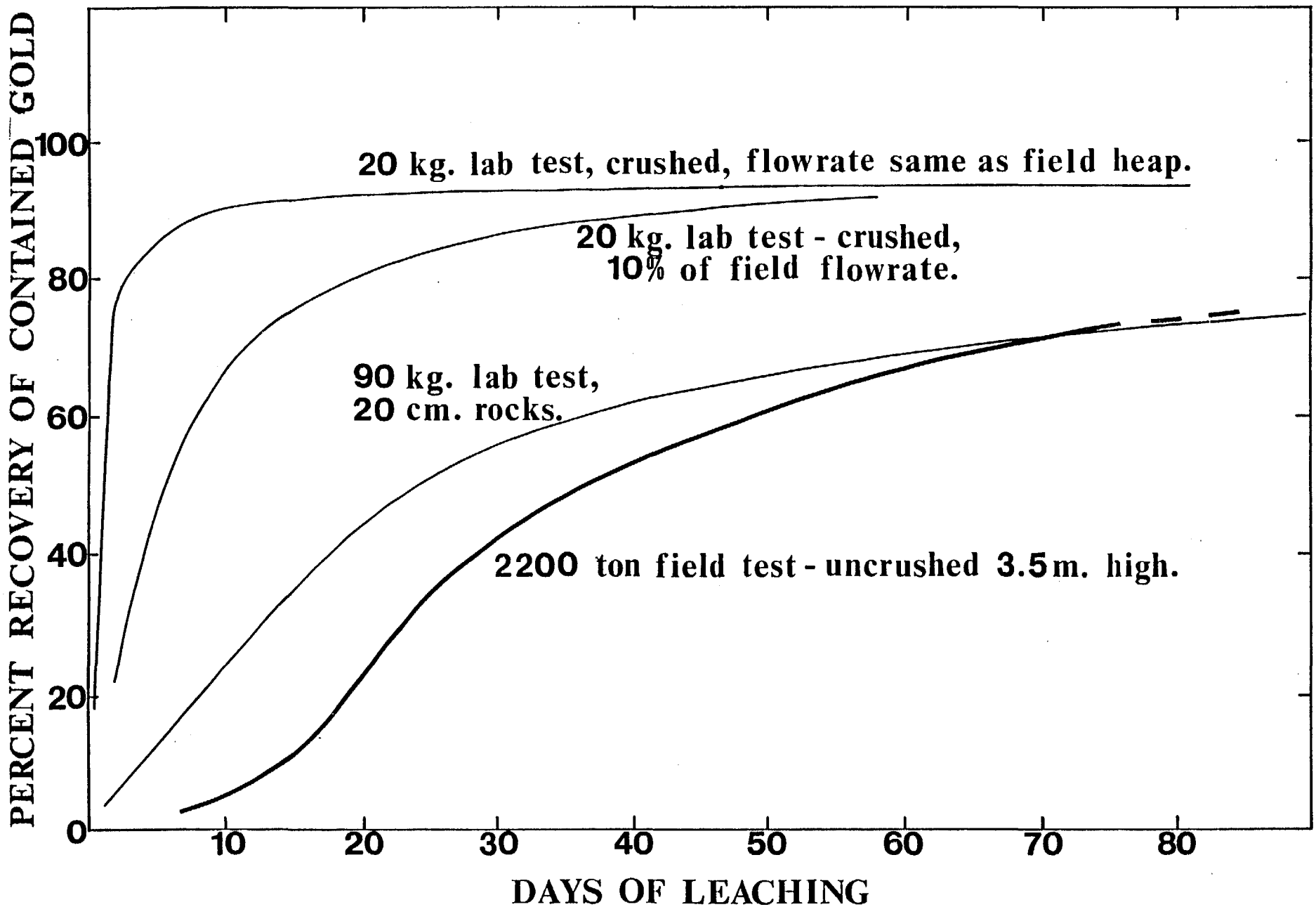


Figure 4: Effect of lab test variations.

Example 5. - Time Factors in Testing

Figure 5 makes a point somewhat removed from the general scheme of this paper. The figure shows laboratory test curves for three different ores. All three ores have approximately the same ultimate recoveries, near 60 percent. Had only short-term laboratory tests been run on these ores, however, the apparent picture would have been radically different. Recoveries after ten days ranged from 14 percent to 50 percent. A problem commonly faced by every testing laboratory is the "when can I have the answer?" syndrome, and a common response is to shorten the laboratory testing time.

Conclusion

In production heaps, allowing extra leaching time does more than insure against scaleup errors. At an ongoing project where the heaps are run counter-currently, the cost to leach "old" heaps is roughly $0.07 per ton per month. Significant additional cash flow can be generated from heaps producing only 0.5 percent additional recovery per month.

It appears that "time" must be added to a heap leach like "reagents" to a conventional mill; the approximate levels can be determined in the lab, but field operators must be given the flexibility to make final adjustments.

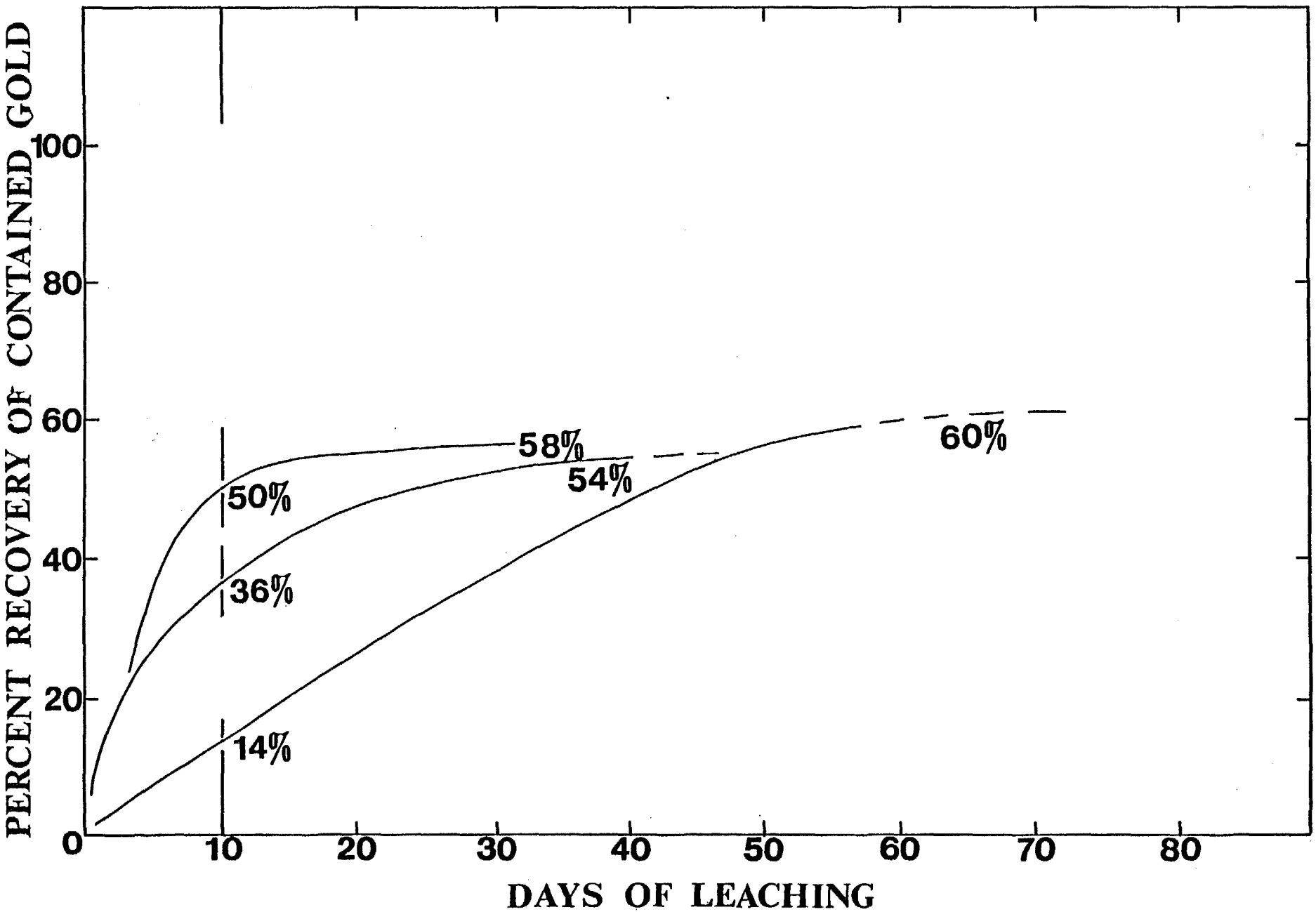


Figure 5: Leach time comparison laboratory small bucket tests, three ores, same conditions.

Copy for RC Brown
Smith
Bell
Vitor
Balla

10/29/85

GRAPHICS PRESENTED BY ROSHAN BHAPPU AT THE 1985 RANDOL GOLD FORUM IN SANTA FE, N.M.

BHAPPU tends to be quite optimistic

NEWLY DEVELOPED PROCESSES

HEAP AND DUMP LEACH

FINE CRUSHING - AGGLOMERATION - CYANIDE CURING

VAT LEACH MODIFICATIONS

SILVER STATE, CHILSON & KAMYR

IN-SITU LEACH

AUSTRALIAN DEEP LEADS

ION-EXCHANGE (IX) & SOLVENT EXTRACTION (SX)

DIRECT ELECTROWINNING FOR REFACTORY ORES & CONCENTRATES

HIGH PRESSURE OXIDATION (180°-210°C & 350-1400 kPa)

"ARSENO" LOW PRESSURE OXIDATION (80°- 100°C & 400-800 kPa)

BIOLOGICAL OXIDATION (THIOBACILLUS FERROOXIDANS, ph1-2)

THIOUREA LEACH (Fe^{III}, ph 1-2, NUTRIANTS)

THIOSULFATE LEACH (FOR Ag-Mn ORES)

CHEMICAL BLOCKING-CYANIDATION (FOR CARBONACEOUS ORES)

TABLE I - APPLICABLE PROCESSES BY ORE CLASSIFICATION

Ore Classification	Applicable Process	Mine/Location
Free Milling	Gravity - Amalgamation - Cyanidation	Dome, Vaal Reefs Blyvooruitzicht
	w/CIP	Homestake, Battle M
	w/Flotation-Cyanidation	Cambell Red Lake Kalgoorlie, Knob
	Direct Cyanidation (all slime circuit)	Benguet, Kloof Camflo, Kinross Kloof, Pueblo Viejo Pinson, DeLamar(Ag)
Gold in Pyrite	Flotation-Cyanidation of Tailings	Knob Hill
	Flotation-Cyanidation of Concentrates	Itogon Suyac
	Flotation-Roasting-Cyanidation	Kerr-Addison
Gold W/Pyrrhotite	Lime Aeration-Cyanidation	Dome, Homestake
Gold in Tellurides	Flotation-Chem.Oxidation-Cyanidation	Emperor
	Flotation-Roasting-Cyanidation	Kalgoorlie
Gold w/Arsenopyrite	Flotation-Roasting-Cyanidation	Campbell Red Lake
	w/CIP	Giant Yellowknife
	Flotation/Autoclaving/Cyanidation	
	Flotation/Bio-oxidation/Cyanidation	Under Development
Complex Ores (Au w/ Cu,Pb & Zn Sulfides)	Flotation/Cyanidation of Tailings	Benguet Itogon Suyac
Carbonaceous Ores	Chemical Oxidation-Cyanidation	Carlin, Mercur, Jerritt Canyon
	Chemical Blocking-Cyanidation	Kerr-Addison
Lower Grade Ores	Heap and Dump Leach with or without Agglomeration	Round Mountain Ortiz, Gold Quarry Alligator Ridge Mesquite

TABLE II

HEAP AND DUMP LEACHING PRACTICES IN USA

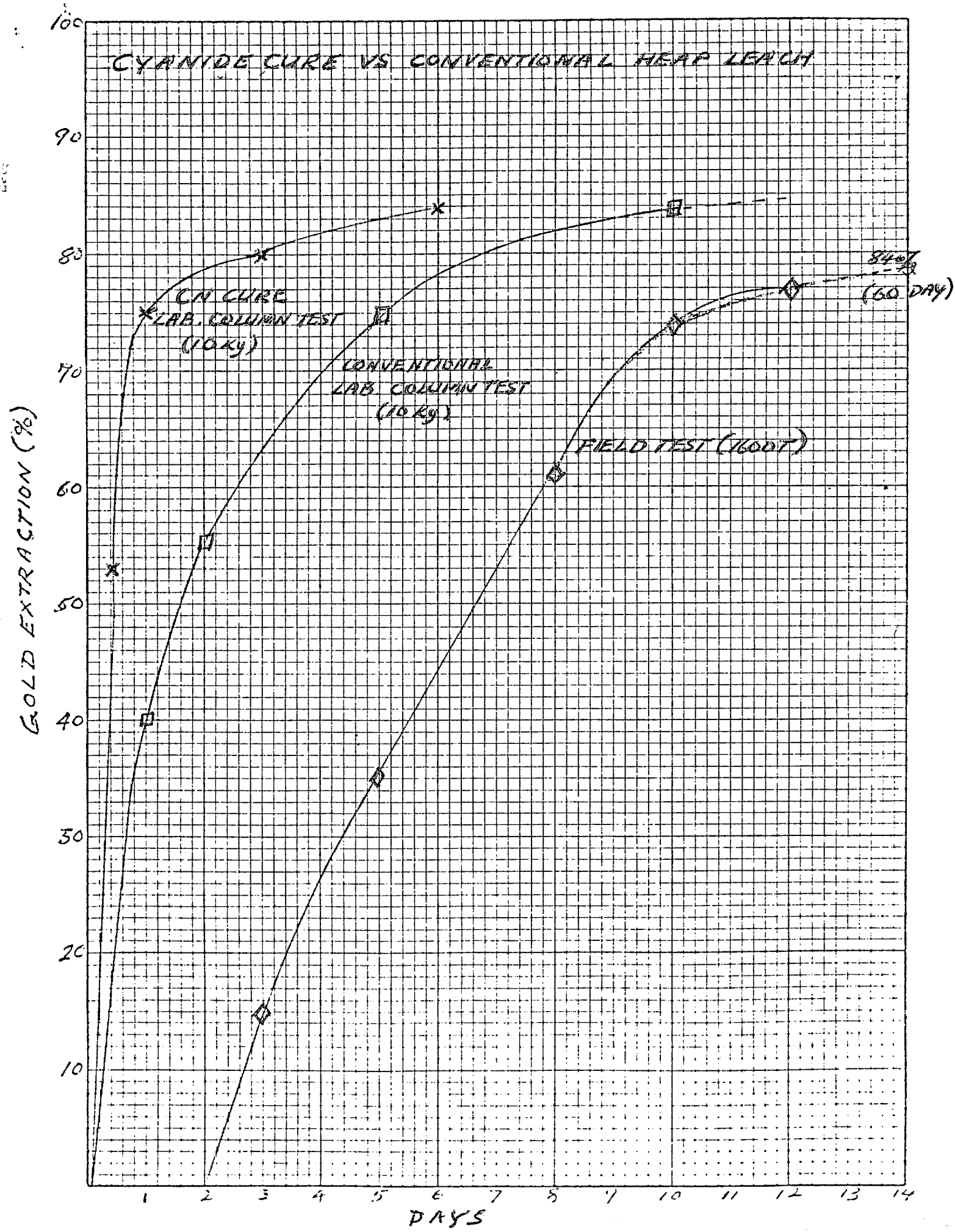
MINE/OPERATOR	RESERVES (000's)	GRADE (oz/t Au)	CAPACITY (stpd)	TYPE OPERATION	RECOVERY (%)
MAGGIE CREEK–NEWMONT	3,600	0.049	2,000	UNCRUSHED-HEAP-CA-EW	45
GOLD QUARRY–NEWMONT	90,000	0.035	20,000	UNCRUSHED-DUMP-CA-EW	45
ZORTMAN/LAND–PEGASUS	28,000	0.035	16,000	UNCRUSHED-DUMP-MC	50
RELIEF CANYON–LACANA	9,200	0.032	4,000	UNCRUSHED-HEAP-CA-EW	50
PINSON–PINSON MINING	2,200	0.025	1,500	UNCRUSHED-HEAP-CA-EW	60
PREBBLE–PINSON MINING	1,800	0.062	1,000	UNCRUSHED-DUMP-CA-EW	40
CANDELARIA–NERCO	15,000	3.15 (Ag)	10,300	-2"-AGGL.*-HEAP-MC	45
BOREALIS–TENNECO	2,000	0.090	2,500	-1"-AGGL.*-HEAP-MC	70
ALLIGATOR RIDGE–AMSELCO	3,000	0.070	3,000	-3/4"-AGGL.*-HEAP-CA-EW	80
ORTIZ–GOLDFIELDS	7,100	0.053	3,500	-3/8"-AGGL.•-HEAP-CA-EW	80
ROUND MT.–SMOKY VALLEY	200,000	0.037	18,000	-3/8"-HEAP-CA-EW	75
NORTHUMBERLAND–CYPRUS	16,000	0.050	4,000	-5/16"-AGGL.-HEAP-CA-EW	70

*AGGL. - AGGLOMERATION
•AGGLOMERATION WITH LIME OR CEMENT AND CYANIDE

SUMMARY OF AGGLOMERATION CONDITIONS

(USBM - IC 8945)

Operation	Cap. (stpd)	Size (in)	Binder Used (type)	(lb/ton)	Agglomeration (equip)	(H2O%)	Cure (hr)	Rate (days)	Recovery (%)
#1-AZ	20	3/8	CEMENT	10	DRUM	10	48	5	90
#2-CO	1,500	1/2	CEMENT	3	DRUM	13	8	3	90
			FLY ASH	7					
#3-AZ (Ag)	2,000	1/2	LIME	7	BELT	11	72	7	90
#4-NV	2,500	5/8	CEMENT	8	NONE	12	60	20	90
#5-NV	3,000	3/4	CEMENT	7	DRUM	8	120	20-80	70-90


CYANIDE CURE VS CONVENTIONAL HEAP LEACH
CN CURE
LAB. COLUMN TEST
(10 Ky)
CONVENTIONAL
LAB. COLUMN TEST
(10 Ky)
FIELD TEST (1600T)
84%
(60 DAY)
GOLD EXTRACTION (%)
DAYS
100
90
80
70
60
50
40
30
20
10
1
2
3
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14

TABLE III

CAPITAL COST ESTIMATES FOR PROCESS ALTERNATIVES

ALTERNATE PROCESSES

A. HEAP LEACH: UNCRUSHED ORE-CA-EW

B. HEAP LEACH: CRUSHED ORE-CA-EW

C. HEAP LEACH: FINE CRUSHED ORE-AGGLOMERATION-CA-EW

D. CYANIDATION: CARBON-IN-PULP (CIP)

E. CYANIDATION: CONVENTIONAL CYANIDATION-CCD-MC

F. CYANIDATION: PRESSURE OXIDATION-CYANIDATION

CAPITAL COST ESTIMATES

(DOLLARS PER TON PER DAY)

CAPACITY	ALTERNATE PROCESSES					
(STPD)	A	B	C	D	E	F
1,000	$2,640	5,700	10,325	17,035	25,815	43,410
2,000	2,000	4,330	7,825	12,910	19,560	32,900
3,000	1,700	3,680	6,655	10,980	16,640	27,975
4,000	1,515	3,280	5,925	9,780	14,820	24,935
8,000	1,150	2,485	4,695	7,420	11,240	18,895
16,000	870	1,875	3,370	5,565	8,430	14,320

TABLE IV

OPERATING COSTS FOR PROCESS ALTERNATIVES

ALTERNATE PROCESSES

A. HEAP LEACH: UNCRUSHED ORE-CA-EW

B. HEAP LEACH: CRUSHED ORE-CA-EW

C. HEAP LEACH: FINE CRUSHED ORE-AGGLOMERATION-CA-EW

D. CYANIDATION: CARBON-IN-PULP (CIP)

E. CYANIDATION: CONVENTIONAL CYANIDATION-CCD-MC

F. CYANIDATION: PRESSURE OXIDATION-CYANIDATION

OPERATING COST ESTIMATES

(DOLLARS PER TON ORE TREATED)

CAPACITY	ALTERNATE PROCESSES					
(STPD)	A	B	C	D	E	F
1,000	$2.40	4.85	5.80	7.75	9.70	19.40
2,000	2.35	4.70	5.65	7.55	9.45	18.90
3,000	2.30	4.50	5.50	7.35	9.20	18.40
4,000	2.20	4.25	5.35	7.15	8.95	17.90
8,000	1.95	3.75	4.50	6.00	7.50	15.00
16,000	1.60	3.25	3.50	5.20	6.50	13.00

COST FACTORS FOR PROCESS ALTERNATIVES

PROCESS ALTERNATIVES	COST FACTORS	
	CAPITAL	OPERATING
CONVENTIONAL CYANIDATION-MC	1.00	1.00
CYANIDATION: CARBON-IN-PULP	0.66	0.80
CYANIDATION: PRESSURE OXIDATION (ORE)	1.68	2.00
CYANIDATION: BIO-OXIDATION (CONC.)	1.48	1.75
HEAP LEACH: UNCRUSHED ORE-CA-EW	0.10	0.25
HEAP LEACH: CRUSHED ORE-CA-EW	0.22	0.50
HEAP LEACH: FINE CRUSHED ORE-AGGLOMERATION-CA-EW	0.40	0.60

EXPLORATION PROJECT EVALUATION
(CRUSH-AGGLOMERATE-CURE-CA-EW)

	700,000 STPY	1,500,000 STPY
ESTIMATED CAPITAL COSTS (MINE & MILL)	$20,000,000	$30,000,000
ESTIMATED OPERATING COSTS	$11.00	$8.00
BREAKEVEN RECOVERABLE GRADE (@ $300/OZ. GOLD)	0.037	0.027
ORE GRADE @ 80% RECOVERY	0.046	0.034
ORE GRADE REQUIRED FOR 25% ROI (MINE LIFE 6 YEARS)	0.086	0.054
REQUIRED ORE RESERVES (EXPLORATION TARGET)	4,200,000	12,000,000

HEAP LEACHING OPERATING COSTS

UNIT OPERATION	AV. (%)	700,000 STPY	2,000,000 STPY
MINING	33	$ 3.30	$2.31
CRUSHING/HANDLING	12	1.20	0.84
LEACHING	22	2.20	1.54
CA/CD/EW	11	1.10	0.77
REFINING/ASSAYING	2	0.20	0.14
DIRECT COSTS	80	$ 8.00	$5.60
INDIRECT COSTS	20	2.00	1.40
TOTAL COSTS	100	$10.00	$7.00

OPERATING COST DISTRIBUTION

	(%)	2,000,000 STPY
LABOR		
OPERATING	30	2.10
MAINTENANCE	10	0.70
SUPPLIES		
REAGENTS	24	1.68
MAINTENANCE	6	0.42
UTILITIES	2	0.14
SUPERVISION	5	0.35
ASSAYING & LAB	3	0.21
INDIRECTS	20	1.40
	100	$7.00

Arizona Bureau of Geology and Mineral Technology
FIELDNOTES
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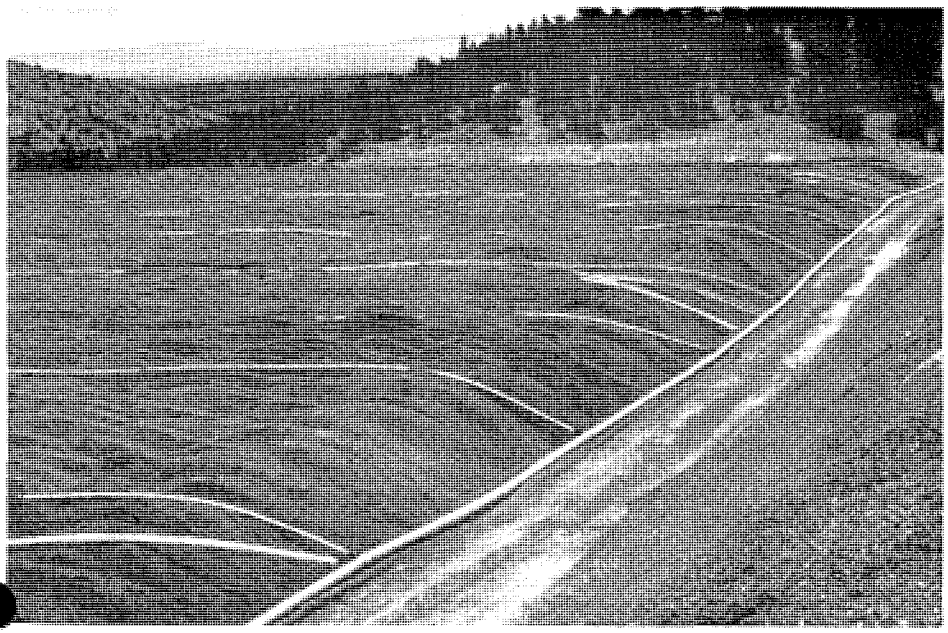


Figure 1. Typical heap-leaching operation for recovery of gold from crushed ore using dilute cyanide solutions. Heap leaching of crushed ore usually involves 30- to 60-day leach cycles and yields gold recoveries of 60 to 90 percent.

Gold and Silver Extraction: the Application of Heap-Leaching Cyanidation

by J. Brent Hiskey
Assistant Director
Arizona Bureau of Geology
and Mineral Technology

In the last decade, heap leaching has established itself as an efficient method of treating oxidized gold and silver ores. Also known as solution mining, heap leaching involves the extraction of soluble metals or salts from an ore by distributing solutions over an open ore heap piled on an impervious pad. For example, gold and silver can be separated from their ores by the application of a weak solution of cyanide and lime or caustic soda. Figure 1 shows a typical precious-metal heap-leaching operation. This processing technique is an extremely efficient way of extracting metals from small, shallow deposits, but is especially attractive for treating large, low-grade, disseminated deposits. Compared with conventional milling (crushing, grinding, and agitation leaching), recovery of gold and silver by heap leaching offers several advantages: lower capital and operating costs, shorter start-up times, and fewer environmental risks. These advantages, however, are sometimes offset by lower metal extractions.

The first commercial application of heap leaching, by the Carlin Gold Mining Company in northern Nevada, occurred in the late 1960's. Since that time, advances in solution-mining technology and the continued high price of gold have sustained a strong interest in heap leaching. About 25 percent of the new gold and 10 percent of the new silver currently produced in the United States are obtained through heap leaching. Nevada is clearly the leader in precious-metal heap leaching with about 50 stable operations and another 50 that operate depending on the price of gold and silver (Carrillo, 1985). Two examples of large-scale heap-leaching operations, both of which are in Nevada, are the Smoky Valley operation, which mines 11,000 metric tons of ore per day (mtpd) and produces about 3,700 kilograms (kg) of gold and 2,200 kg of silver annually; and the Candelaria plant, which is primarily a silver mine that processes 9,400 mtpd and produces about 90,000 kg of silver annually.

The locations of the major western U.S. gold-and-silver heap-leaching operations are shown in Figure 2. The newest of these are the Picacho mine of Chemgold, Inc. and the Mesquite mine of Goldfields Mining Corporation. Both plants are in California, approximately 25 miles outside Yuma, Arizona.

The purpose of this article is to review important factors related to cyanidation and heap-leaching practice. The article briefly summarizes the history of cyanidation and discusses the operational features of heap leaching using dilute cyanide solutions.

HISTORY OF CYANIDATION

The historical fabric of cyanidation is very colorful: it is woven with many threads of controversy and disagreement. Numerous patent disputes have been documented in technical journals. Cyanidation technology appears to be evolutionary, with advances progressing from prior work. Seven such advances provide a historical summary of cyanide leaching: early cyanide chemistry, the cyanide process, Merrill-Crowe (zinc-dust precipitation), carbon adsorption, carbon-in-pulp (CIP), Zadra (stripping and electrowinning), and heap leaching.

Early Cyanide Chemistry

As early as 1793, aqueous solutions of potassium cyanide were known to exhibit a solvent action on gold (Habashi, 1970). In 1843 Bagration produced the first scientific treatise on the subject. During his investigation, he observed the following:

(1) Dissolution is more rapid when the gold is divided into very small particles.
(2) Dissolution increases with heating.
(3) Gold dissolved in cyanide will precipitate on metallic surfaces in the absence of an applied potential.
(4) The presence of air decreases the time necessary for dissolution (McCann, 1912).

Figure 2. Major western U.S. gold-and-silver heap-leaching operations.

In 1844 Elsner discovered that gold dissolution in cyanide was due to the action of dissolved oxygen and not the decomposition of water, as previously believed (Wilson, 1896). The "Elsner Equation," stated in its original form, is as follows:

$$2Au + 4KCy + O + H_2O = 2KAuCy_2 + 2KOH$$

This early formula established the overall stoichiometry of gold cyanidation and recognized the presence of a gold-cyanide complex.

Bodlaender (1896) rapidly dissolved gold in an aerated cyanide solution and found that significant quantities of hydrogen peroxide (H_2O_2) were produced during the reaction. The formation of hydrogen peroxide as an intermediate product suggested a two-step reaction sequence:

$$2Au + 4CN^- + O_2 + 2H_2O = 2Au(CN)_2^- + 2OH^- + H_2O_2$$

$$2Au + 4CN^- + H_2O_2 = 2Au(CN)_2^- + 2OH^-$$

The sum of these reactions is equivalent to the overall reaction as proposed by Elsner.

Barsky and others (1934) provided some of the first fundamental thermodynamic and kinetic data for the cyanidation of gold and silver. Their work confirmed the accuracy of the equations offered by Elsner and Bodlaender. They determined the values for the free energy of formation of the aurocyanide ion, $Au(CN)_2^-$ and the argentocyanide ion, $Ag(CN)_2^-$. They also investigated the effects of cyanide concentration and pH on the rate of gold and silver dissolution. The maximum rate of gold dissolution was obtained with a 1×10^{-2} M NaCN solution. The rate of gold dissolution was found to be insensitive to pH between about pH 10.5 and pH 12.5.

Boonstra (1943) was reportedly the first to recognize the similarity between gold dissolution in cyanide and metal corrosion processes. This observation established the importance of the electrochemical dissolution mechanism involving distinct anodic and cathodic steps.

The Cyanide Process

It was not until the pioneering work of MacArthur and the Forrest brothers (1887, 1889) that cyanidation became a commercial process. Their patents of 1887 and 1889 were milestones in the metallurgy of gold because they recognized a chemical process for treating gold ores. Their contributions included two important advancements: (1) they used dilute cyanide solutions to produce a selective action on gold instead of using stronger lixiviants, which have a tendency to dissolve impurities; and (2) they proposed a method for recovering gold from cyanide solution by precipitation with zinc shavings.

Although cyanide had been used as early as 1870 to treat gold ores in the United States, the MacArthur-Forrest process was not officially introduced to domestic mining operations until 1892. The first cyanide gold mill in the United States was reportedly established at the Vulture mine near Wickenburg, Arizona (Young, 1967; Figure 3). Haynes (1892) reported the successful treatment of tailings and rebellious ores using cyanide by the Yavapai Gold and Silver Extraction Company in Prescott, Arizona. At about the same time, several plants near Tombstone, Arizona were using cyanide leaching to recover silver. By 1896, there were seven major cyanide plants in the United States, the largest of which was the Mercur mill in Utah. This plant had a capacity of 183 mtpd and achieved gold extractions ranging from 80 to 87 percent (Packard, 1896).

Merrill-Crowe

Early cyanide practice involved gold precipitation onto zinc shavings. Cementation of gold onto a metallic zinc surface is represented by the following reaction (Wilson, 1896):

$$2Au(CN)_2^- + Zn^0 = 2Au^0 + Zn(CN)_4^{2-}$$

The electrochemical cementation reaction shown above is relatively simple, involving the discharge of a noble metal ion (the gold-cyanide complex) at the expense of a more reactive metal (the zinc dust).

Zinc-dust precipitation, known as the Merrill system, was introduced in the United States in 1897 and is the basis of modern practice (Julian and others, 1921). Oxygen necessary for the oxidation of gold during cyanide leaching is detrimental to efficient zinc-dust precipitation. Merrill recognized this and designed a process to avoid air contact with the zinc during precipitation. Crowe (1919) improved the process by removing dissolved oxygen from the gold-bearing cyanide solution prior to addition of the precipitating agent. This approach improved the efficiency of gold precipitation and decreased zinc consumption. The Merrill-Crowe process consisting of solution clarification, deaeration, zinc-dust precipitation, and precipitate filtration continues to be an important precious-metal recovery method used by many modern plants.

Carbon Adsorption

Gross and Scott (1927) undertook the first rigorous study of the adsorption and desorption of gold and silver cyanide on carbon. They reviewed the early history of carbon adsorption, noting that charcoal

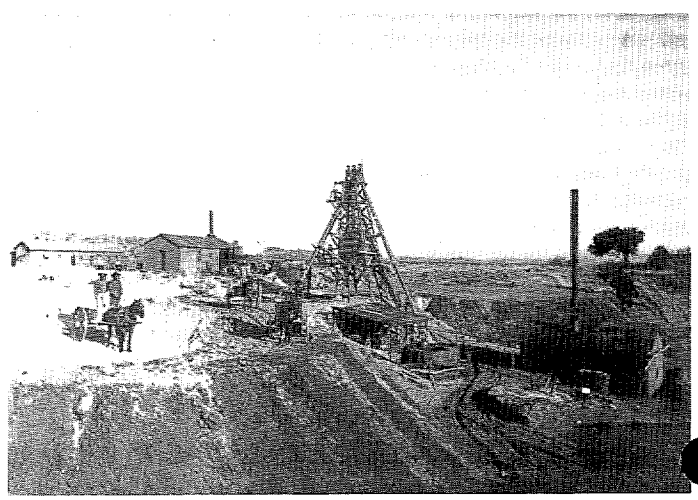


Figure 3. Reportedly the first U.S. cyanide plant, as seen in the early 1890's, at the Vulture mine, Wickenburg, Arizona. Photo courtesy of the Arizona Historical Society.

was first used in chlorination plants to recover gold. As cyanidation became more popular than chlorination, it was only natural that charcoal would be used to recover precious metals from cyanide solution. Charcoal was used at a number of locations by the turn of the century. As improvements in zinc-dust precipitation emerged, however, interest in the use of charcoal declined.

Early investigators believed that charcoal precipitated gold from solution (Green, 1913; Feldtman, 1915; Edmands, 1918). Actually, a reduction mechanism by which gold is deposited in the metallic state on the carbon surface does not explain the experimental evidence. Unfortunately, complete understanding of the adsorption mechanism is still lacking. High surface area (active sites) and pore diffusion are important aspects of the adsorption and desorption processes.

Carbon-In-Pulp

A major contribution to carbon-in-pulp (CIP) technology was made during the 1930's by Chapman (1939) of the Department of Metallurgy, University of Arizona. Chapman and his graduate students investigated dissolution of gold by cyanide and adsorption of dissolved gold onto activated charcoal in ore pulps. The general process used flotation to separate and recover the gold-bearing charcoal from the leached tailings. The flotation of powdered carbon proved to be highly selective, producing a charcoal concentrate that could be either smelted directly or ashed to recover the gold. A portion of the early research at the University of Arizona was devoted to the activation of carbon prior to adsorption of gold from cyanide solution (Rabb, 1939).

The lack of a convenient method for stripping adsorbed gold and silver from loaded carbon handicapped the development of CIP practice. Without stripping, it was impossible to recycle carbon in a closed-circuit system (adsorption, stripping, and reactivation). These techniques were pioneered by the U.S. Bureau of Mines (Zadra and others, 1952; Hussey and others, 1979) and have been improved and engineered for large-scale operations by MINTEK in South Africa (Laxen and others, 1979). Advantages of the CIP process include the ability to handle ores with poor settling and filtration characteristics by eliminating the need for costly liquid-solid separation systems; efficiency in the recovery of gold from dilute process streams; high capacities; and relatively simple design and operation.

Zadra

The next major contribution to the cyanide process was the recovery of gold by activated carbon, stripping the carbon with hot caustic cyanide solution, and electrowinning the gold and silver onto stainless-steel wool. This process was developed in the early 1950's by Zadra and others (1952). Stripping times ranged from 24 to 48 hours, as originally practiced. These times were quite long, consuming more chemicals and requiring more carbon in the circuit. In recent years, the U.S. Bureau of Mines and the Anglo-American Research Laboratory have developed pressure stripping of gold from carbon, which requires only 6 to 8 hours. Other improvements in the Zadra technique include the design of advanced electrowinning cells.

Heap Leaching

During the last decade, heap leaching of gold and silver ores has evolved into an extremely efficient method of treating small deposits once considered uneconomic; heap leaching as a mineral technology, however, has been practiced for centuries. As early as the mid-16th century, some mines in Hungary were recycling copper-bearing solution through waste heaps. By the 18th century, large-scale heap leaching was practiced by the Rio Tinto Company in Spain to recover copper from cupreous pyrites. By 1900 these leaching operations were employing such techniques as leach/rest cycles to maximize copper recovery (Taylor and Whelan, 1942).

As mentioned earlier, heap leaching of precious metals was commercially developed in the late 1960's. Improvements in heap-leaching performance and efficiency have continued to emerge in the areas of feed preparation (agglomeration), heap design and construction, solution distribution, and metal recovery.

GOLD AND SILVER HEAP LEACHING

To be amenable to heap leaching, a gold-bearing rock should be competent, porous, and relatively cyanicide free and should contain clean, fine-grained gold particles (Potter and Salisbury, 1974). It is also essential that good aeration and uniform solution contact be maintained. These same factors influence the heap leaching of silver ores.

Since precious-metal ore bodies vary significantly in geology, mineralogy, and metallurgy, it is difficult to generalize about flow-sheet design. The layout of a typical heap-leaching operation is depicted in Figure 4. Like other solution-mining methods, heap leaching is sensitive to site-specific factors. These factors include topography and space, climatic conditions, availability of pad construction materials (i.e., clays), environmental restrictions, and water. Because of its intrinsic simplicity and flexibility, heap leaching is ideally suited to deal with these factors.

Leaching Methods

There are basically two variations of the heap-leaching method that are used on a commercial scale. The first approach is based on the leaching of run-of-mine ore. Long leaching cycles and low-grade ores are usually associated with this variation. The second approach, which involves the leaching of crushed ore, normally requires shorter leaching cycles. High-grade deposits generally justify the increased cost of crushing and are often expressly treated to maximize gold and silver recovery by increasing the exposure of gold and silver to the leaching solution.

Heap-leaching operations that use crushing typically leach ore crushed to 20 millimeters (mm), and sometimes as fine as 6 mm. The leach cycle for this type of operation usually takes a few weeks to several months. Most ores that do not contain excessive amounts of clay will exhibit good permeability down to ore sizes of approximately 10 mm. Feeds below this size or those that contain high proportions of clay can be treated by agglomerating techniques that stabilize fine clay particles. Run-of-mine feed sizes are those produced by the mining operation (blasting or ripping) and may typically contain 150-mm rocks, but may also include some boulders. The leach cycle for run-of-mine uncrushed ore frequently takes months or years.

The cycle commences when ore is delivered to specially prepared, impermeable drainage pads. The leach pad serves two important functions: to protect the environment and to collect and eliminate loss

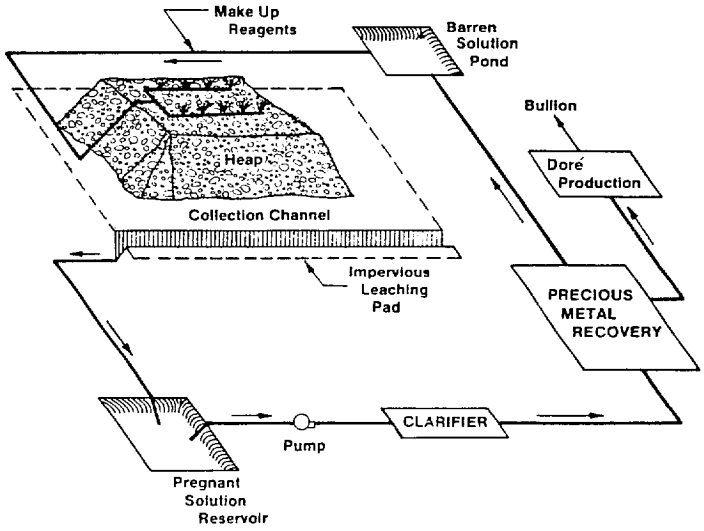


Figure 4. General layout of a heap-leaching operation. In addition to the main leaching circuit, which is described in the text, the figure shows the ancillary operations of doré production and reagent makeup. Doré production involves the smelting and refining of steel wool and zinc precipitates to produce bullion. Reagent makeup relates to the addition of chemicals to control cyanide concentration, pH, and scaling problems.

of pregnant solution, the solution that is saturated with the dissolved metal. Several types of materials are used to construct leach pads, including both natural and manmade materials such as synthetic membranes and geotextile fabrics. Van Zyl (1984) has listed the criteria that pad materials must satisfy: pad-permeability requirements (environmental restrictions), cost constraints, pad-construction considerations, and heap-construction factors. Single-use pads are often constructed from clay or plastic liners, whereas multiuse pads, which are used for more than one leaching cycle, are best made from asphalt or reinforced concrete. In all cases, pad construction must entail careful preparation of the base and subbase layers. In single-use pads, it is common to cover the pad area with a network of perforated pipes to aid drainage and to collect pregnant leach solution. These pipes are normally protected by a layer of coarse gravel.

There are several techniques for placing ore on the leaching pad. Chamberlin (1981) discussed heap-construction methods to maximize permeability and leaching efficiency. Some techniques used to build heaps include haulage trucks and dozers to spread the ore, front-end loaders, conveyor-stackers, and movable bridge-conveyor distribution units.

Solution distribution is of special importance in every heap-leaching operation. Leach solutions can be applied to the surface of heaps by a variety of methods such as ponding/flooding, trickle systems, multiple low-pressure sprinklers, single high-pressure sprinklers, and subsurface injection.The most popular systems used in gold-and-silver heap leaching are the impulse sprinkler and the wobbler-type sprinkler. These types of sprays provide uniform solution coverage of the heap surface. Solution application rate is also a critical factor in heap leaching. For typical heap-leaching operations, the maximum effective application rate that can be used without causing channeling and short circuiting of solution is about 4 cm/hr (1 gal/ft^2/hr). Higher application rates restrict the movement of oxygen through the heap, dilute the pregnant-solution grade, and increase pumping costs. In general, solution-application rates for heap leaching range between 0.8 and 1.2 cm/hr (0.2 and 0.3 gal/ft^2/hr).

Solutions migrate downward through the heap under free-flow conditions (gravity). The percolating solutions dissolve gold and silver as they contact the ore minerals. Gold and silver occur mainly along fracture surfaces in oxidized ores. Values exposed by crushing are readily accessible to the lixiviant and are recovered by simple surface flushing. Leaching solution can also penetrate particle fractures by capillary action; in this domain, long-range diffusion must occur. The pregnant leach solution is collected on the pad and drains to a collection system, which delivers it to a pregnant-solution reservoir. From this reservoir, the solution is pumped to a precious-metal recovery circuit.

Gold and Silver Recovery

There are two primary commercial methods of recovering gold and silver from alkaline cyanide heap-leach solutions: zinc-dust (Merrill-Crowe) precipitation and adsorption by activated carbon. The choice between carbon adsorption and zinc precipitation depends on several factors including solution concentration, solution volume, and solution clarity. Potter (1981) concluded that large volumes of low-grade solutions, mainly containing gold, are most economically treated by carbon adsorption, whereas small flows of relatively rich solutions or solutions containing large quantities of silver should be treated by zinc-dust precipitation. The feed solutions to a carbon-adsorption circuit do not have to be clarified. For optimal efficiency of zinc precipitations, however, it is essential that feed solutions be clarified.

Merrill-Crowe zinc-dust precipitation, as noted earlier, is a very mature and well-established technology. The basic process consists of solution clarification, deaeration, precious-metal precipitation, and precipitate filtration. As stated earlier, cementation of gold onto a metallic zinc surface is represented by the following reaction:

$$2Au(CN)_2^- + Zn^0 = 2Au^0 + Zn(CN)_4^{2-}$$

A similar reaction can be written to express the precipitation of silver. Based on the stoichiometry of this reaction, the theoretical zinc requirement for gold precipitation is equivalent to 0.17 grams of zinc per gram of precipitated gold. Actual zinc consumption in practice is much higher than this, ranging from 10 to 30 grams of zinc per gram of precipitated gold for dilute heap-leach solutions. The difference between actual and theoretical ratios is attributed to the presence of impurities and dissolved oxygen.

Zinc precipitation is the preferred process for silver ores because of the high silver concentrations and poor silver-loading characteristics of carbon. Zinc-dust precipitation is especially attractive for small volumes of solution because they can be processed in modular Merrill-Crowe units.

A continuous multistage carbon-adsorption circuit is an efficient way of recovering gold and silver from high-volume, low-grade, heap-leach solutions. The standard design involves pumping pregnant leach solution countercurrently to activated carbon in a series of five or more columns. Carbon in the size range of 16 x 30 mesh is fluidized by the upward flow of solution and is advanced through the circuit to achieve loading in the range of 3,430 to 6,860 grams per metric ton (100 to 200 ounces per short ton). Carbon loadings in excess of this range are generally avoided because of gold losses to the barren solution. Loadings lower than these values would require advancing the carbon more frequently. This additional handling could result in a higher rate of carbon attrition and an attendant gold loss with fine carbon.

Loaded carbon is advanced from the first-stage adsorption circuit to stripping. The popular stripping methods involve variations of hot, atmospheric sodium hydroxide (NaOH) and sodium cyanide (NaCN), pressurized NaOH and NaCN, and alcohol stripping. Gold and silver are usually recovered from the rich strip solution by electrowinning onto steel-wool cathodes.

In time, surface sites on carbon are contaminated with organic materials and other impurities, and pores are blocked by precipitated salts (calcium carbonate). It is necessary to periodically reactivate the carbon to remove these contaminants and to restore the intrinsic chemical activity of the carbon. Reactivation requires a series of chemical and thermal treatments: acid washing to remove surface deposits, and calcining to 750°C by indirect means to activate the surface site. After screening to remove fines, the regenerated carbon is returned to the final stage of the adsorption circuit.

SUMMARY

The extractive metallurgy of gold and silver from ores is primarily based on cyanidation practice. Cyanidation has been practiced commercially for a century. The chemistry of leaching gold and silver in cyanide solutions is elegant and the process of metal recovery by this approach is very efficient. Unlike stronger lixiviants, dilute cyanide solutions exhibit a high degree of selectivity in the dissolution of gold and silver from their ores. Another attractive feature of cyanide is that gold and silver can be conveniently recovered from solution by several methods.

Heap leaching using the principles of cyanidation has recently developed into an important alternative for treating gold and silver ores. Compared to other techniques, heap leaching is a simple process. Though it is especially appropriate for treating large, low-grade, disseminated deposits, heap leaching is also an expedient way of extracting precious-metal values from small, shallow deposits.

New leaching technologies are being developed in the United States and abroad. There is considerable interest in the development of ion-exchange resins for the recovery of gold from cyanide solutions. Resins have the potential of being more selective than carbon, and the design of resin-in-pulp circuits could eliminate such processes as pressure stripping and carbon reactivation. Other emerging technologies include solvent extraction and direct electrowinning from dilute solutions. Alternative lixiviants for extracting gold and silver are also being developed; for example, thiourea and thiosulfate have been examined for various gold- and silver-bearing materials.

Heap-leaching technology is continuing to advance. In-situ leaching, however, is another option for extracting gold and silver from their ores. The possibility of in-situ leaching opens up an area of technology with many challenges as well as the potential for considerable rewards.

REFERENCES

Barsky, George, Swainson, S. J., and Hedley, Norman, 1934, Dissolution of gold and silver in cyanide solutions: American Institute of Mining and Metallurgical Engineers Transactions, v. 112, p. 660-677.

Bodlaender, G., 1896, Die Chemie dos Cyanidver-fahnens: Zeitschrift für angewandle Chemie, v. 4, p. 583-587.

Boonstra, B., 1943, Uber die Loseungsgeschwindigkeit von Gold in Kaliumcyanid-Losungen: Korros Metallschutz, v. 19, p. 146-151.

Carrillo, Fred, 1985, oral communication: U. S. Bureau of Mines, Spokane, Wash.

Chamberlin, P. D., 1981, Heap leaching and pilot testing of gold and silver ores: Mining Congress Journal, v. 67, no. 4, p. 47-52.

Chapman, T. G., 1939, A cyanide process based on the simultaneous dissolution and adsorption of gold: American Institute of Mining and Metallurgical Engineers Transactions, v. 134, p. 207-223.

Crowe, T. B., 1919, Effect of oxygen on precipitation of metals from cyanide solutions: American Institute of Mining and Metallurgical Engineers Transactions, v. 60, p. 111-117.

Edmands, H. R., 1918, The application of charcoal to the precipitation of gold from its solution in cyanide: Institution of Mining and Metallurgy Transactions, v. 27, p. 277-325.

Feldtman, W. R., 1915, The precipitating action of carbon in contact with auriferous cyanide solutions: Institution of Mining and Metallurgy Transactions, v. 24, p. 329-371.

Green, Morris, 1913, The effect of charcoal in gold-bearing cyanide solutions with reference to the precipitation of gold: Institution of Mining and Metallurgy Transactions, v. 23, p. 65-84.

Gross, John, and Scott, J. W., 1927, Precipitation of gold from cyanide solutions by means of charcoal: U.S. Bureau of Mines Technical Paper 378, 78 p.

Habashi, Fathi, 1970, Principles of extractive metallurgy: New York, Gordon and Breach Scientific Publishers, v. 2, 457 p.

Haynes, J. P., 1892, The cyanide process in Arizona: Engineering and Mining Journal, v. 54, no. 1, p. 3.

Hussey, S. J., Salisbury, H. B., and Potter, G. M., 1979, Carbon-in-pulp gold adsorption from cyanide leach slurries: U.S. Bureau of Mines Report of Investigations 8368, 22 p.

Julian, H. F., Smart, Edgar, and Allen, A. W., 1921, Cyaniding gold and silver ores: London, Charles Griffin and Co., Ltd., 417 p.

Laxen, P. A., Becker, G. S. M., and Rubin, R., 1979, Developments in the application of carbon-in-pulp to the recovery of gold from South African ores: Journal of the South African Institute of Mining and Metallurgy, v. 79, p. 315-326.

MacArthur, J. S., Forrest, R. W., and Forrest, W., 1887, Process of obtaining gold and silver from ores: British Patent 14,174.

MacArthur, J. S., Forrest, R. W., and Forrest, W., 1889, U.S. Patents 403,202 and 418,137: U.S. Patent Office Official Gazaette.

McCann, Ferdinand, 1912, Cyanide practice in Mexico: San Francisco, Mining and Scientific Press, 199 p.

Packard, G. A., 1896, The cyanide process in the United States: American Institute of Mining Engineers Transactions, v. 26, p. 709-721.

Potter, G. M., 1981, Design factors for heap leaching operations: Mining Engineering, v. 33, no. 3, p. 277-281.

Potter, G. M., and Salisbury, H. B., 1974, Innovations in gold metallurgy: Mining Congress Journal, v. 60, no. 7, p. 54-57.

Rabb, D. D., 1939, The activation of carbon for the adsorption of gold from cyanide solutions: Tucson, University of Arizona, M.S. Thesis, 37 p.

Taylor, J. H., and Whelan, P. F., 1942, The leaching of cupreous pyrites and the precipitation of copper at Rio Tinto, Spain: Institution of Mining and Metallurgy Bulletin 457, p. 1-36.

Van Zyl, Dirk, 1984, Construction and investigation of clay heap leach pad, *in* Hiskey, J. B., ed., Gold and silver heap and dump leaching practice: New York, Society of Mining Engineers—American Institute of Mining, Metallurgical, and Petroleum Engineers, p. 59-68.

Wilson, E. B., 1896, Cyanide process: New York, John Wiley and Sons, 116 p.

Young, O. E., 1967, How they dug the gold: Tucson, Arizona Pioneers Historical Society, 163 p.

Zadra, J. B., Engel, A. L., and Heinen, H. J., 1952, Process for recovering gold and silver from activated carbon by leaching and electrolysis: U.S. Bureau of Mines Report of Investigations 4843, 32 p.

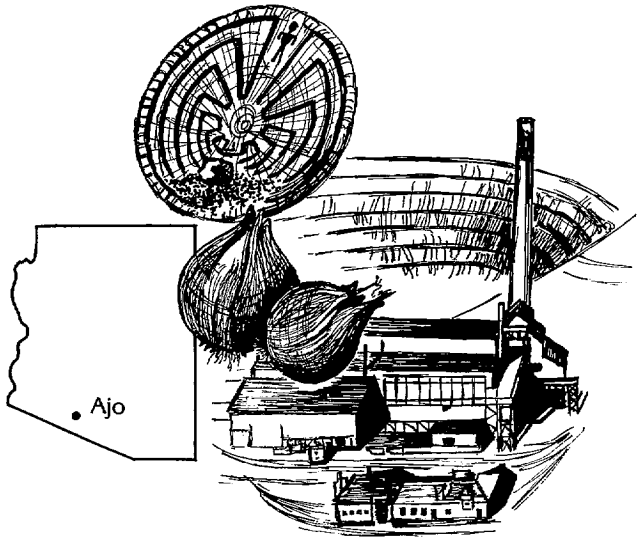
GEOLOGIC PLACE NAMES: AJO

Ajo, Pima County, Arizona. Population 5,650. Pronounced ä´hō. Derived from the Papago word *au'auho*, which means "paint."

A similarity between the sound of the Papago word for this locality and the Spanish word *ajo* for many years led to misapprehension concerning the origin of the name of present-day Ajo. The Papago Indians used *au'auho* in connection with the mines at Ajo because the ores were a source of red paint which the Papagos used to decorate themselves. This was so noted by one of the earliest American travelers in the region. Nevertheless, the fact that the Mexican miners pronounced the word without the double pronunciation of the *au* of the Papago resulted in a word that sounded much like *ajo*. This, added to the fact that the Ajo lily (the root of which looks and tastes much like a spring onion) grows abundantly in this area, led to the belief that the locality was named Ajo because of the wild lilies.

The first American citizen to notice the mining possibilities in the region was Capt. Peter R. Brady, who was with the surveying party for the Thirty-Second Parallel railroad in 1853. When the party broke up in San Francisco, Brady was influential in organizing a group of men to explore mining possibilities at Ajo. This group soon had shipped out all the rich, easily smelted ores from Ajo's mines. Despite the fact that the remaining ores were unquestionably rich, there was no satisfactory way to reduce them economically, and for many years the treasure in copper at the Ajo mines remained relatively untouched. The hills with their rich exposed ores were a speculator's paradise.

In 1910 the population—including Mexicans, Indians, and American citizens—was fifty people. The main business among these people was grazing cattle. Lack of water was a serious problem and poverty [was] rampant. In February 1911, there were only four Americans at what later came to be known as Old Ajo. However, Ajo was on the verge of becoming a boom town. With the



discovery of a leaching process which made it possible to work the ores efficiently and inexpensively, Ajo entered into a prosperous period. The New Cornelia Copper Company was organized, a smelter built, and wells dug. From three to five thousand people were employed by the mines.

The battle between the few old timers in Old Ajo and the powerful mining company was soon joined. The old town of Ajo was far too close to huge deposits of low grade ore which the copper company wanted to develop. The company located its own town a mile to the north, which it proposed to call Cornelia. However, nearly all of Old Ajo burned down, and the name Ajo became attached to the new town.

—Excerpted from Granger, B. H., 1960, Will C. Barnes' Arizona place names: University of Arizona Press, p. 257-258.

Arizona State Trust Lands: Mineral-Resource Revenues and Activities

by Robert A. Larkin
Manager, Nonrenewable Resources and Minerals Section
Arizona State Land Department

The Arizona State Land Department was established in 1915 to administer lands held in trust for the public schools and other beneficiaries. The department's primary goal is to maximize trust revenues through the sale and leasing of State trust lands. In fiscal year (FY) 1984-85, revenues reached an all-time high of $53,074,616, a 69 percent increase from the $31,435,300 generated in FY 1983-84. Of the FY 1984-85 total, $4,286,137 or 8 percent, came from nonrenewable-resource activities. The highest revenues from nonrenewable resources, $11,313,970, were generated in FY 1979-80, just prior to the recession. This represented roughly half of all revenues received that year by the Land Department.

The following article explains how the State Land Department was created and describes the activities of its Nonrenewable Resources and Minerals Section. The latter include such diverse duties as issuing leasing permits and protecting archaeological sites.

BACKGROUND

The Territory of Arizona was established by Congress in 1863. At that time, specific sections of land in each township (sections 16 and 36, if available) were set aside for the benefit of the public schools. In 1910 Congress prepared Arizona for statehood by passing the Enabling Act, which set aside additional land in each township (sections 2 and 32, if available). The act also identified 2,350,000 acres of additional land grants to the State and established a framework by which State lands were to be managed. The act included the following requirements and restrictions, which are still in effect today:

(1) The lands must be held in trust and used only as specified in the act.
(2) Revenues received from lease of the lands and natural products of the lands are subject to the same trust as the lands that produce them.
(3) The lands are not subject to mortgages or other encumbrances.
(4) Lands and natural products are to be leased or sold only after establishment of fair market or "true value."
(5) Lands and natural products may not be sold except to the highest bidder at public auction after 10 weeks of advertising.

Testing gold-recovery equipment at placer operation along the Agua Fria River north of Phoenix.

By enactment of the State Land Code, the Arizona State Land Department was created in 1915 to administer State lands held in trust. This law also gave the department authority over stone, gravel, timber, and other natural resources produced from these lands. The trust officer is the State land commissioner. Today the department has more than 140 employees and manages approximately 10 million acres. Its main office is in Phoenix; field offices are maintained in Flagstaff, Prescott, and Tucson.

NONRENEWABLE-RESOURCE ACTIVITIES

The Nonrenewable Resources and Minerals Section is part of the Natural Resources Division of the State Land Department. Nonrenewable resources include minerals, common mineral materials, oil and gas, geothermal energy, and cultural resources. The section oversees mineral-material sales and issues prospecting permits; mineral, oil and gas, and geothermal leases; and permits for geophysical exploration. Ancillary programs are also conducted, including research on proposed land sales and exchanges, environmental impact studies, review of mining and reclamation plans, and contracting for archaeological surveys and excavations.

The variety of functions performed by the section presents a challenge for its staff members. Decisions regarding land use are not simple or straightforward. Multiple land use is necessary and helps to maximize trust revenues. The section must use management strategies that allow development of nonrenewable resources.

Staff time is mostly spent on processing applications for leases and permits. Application categories are briefly described below. For further information, potential applicants should contact the Nonrenewable Resources and Minerals Section, Arizona State Land Department, 1624 W. Adams St., Phoenix, AZ 85007; telephone: 602-255-4628.

Prospecting Permits

Permits are issued to individuals or companies who wish to explore for valuable mineral deposits. The section receives approximately 3,000 applications per year. During the 1984-85 fiscal year, prospecting permits covered more than 300,000 acres of State land, indicating a strong interest in exploration for a variety of commodities, especially gold and uranium. Upon receipt of a permit application, the section conducts the necessary land records review, and in most cases, a field review. A bond amount is established and a plan of operation is requested from the applicant. After the application has been processed, the permit is mailed to the applicant for signature. A copy must then be returned to the section with the appropriate rental payment and restoration bond. The permit is issued for 1 year at a time, but may be renewed four times for a total of 5 years. Rentals received from prospecting permits in FY 1984-85 totaled $454,460, a 31 percent decline from the FY 1983-84 total of $654,403.

Mineral Leases

Mineral leases are issued to applicants with proof of valuable mineral discovery. The lease, which may extend to 20 years, confers the right to mine and ship ores and to conduct all support operations. In addition to a nominal annual rental, the State is entitled to 5 percent of the net value of the minerals produced. Net value is defined as the gross value after processing minus transportation costs from production to processing sites, processing costs, and taxes. The normal procedure for issuing a lease is to convert a prospecting permit. The section recommends that sufficient mineral exploration be conducted under a prospecting permit before a mineral-lease application is submitted.

More than 600 mineral leases are currently held on State land for commodities such as copper, silver, gold, iron, tin, uranium, limestone, marble, gypsum, zeolites, pumice, bentonite/silica sand, building stone, mica, clay, slate, fire agate, onyx, and pipestone. Exploration for uranium has increased in the past few years. Most activity is occurring north of Flagstaff and in the Arizona Strip north of the Grand Canyon. The bulk of royalty revenues comes from four copper-mining operations in Pima County: Magma in San Manuel; and ASARCO, Eisenhower, and Cypress-Pima near Green Valley. During the past several years, mineral royalties have diminished significantly because of decreased copper prices and reduced production. Total royalties from mineral leases in FY 1984-85 were $1,246,661. This represents a 42 percent increase from the $879,053 generated in FY 1983-84, but an 86 percent decline from the $9,061,000 generated in FY 1979-80.

Mineral-Material Sales

Common mineral materials, classified as natural products of the land, are sold at public auction. Resources in this category include sand, gravel, rock, common building stone, riprap, cinders, decomposed granite, topsoil, and any other mineral material used in the construction industry. Sales agreements with the highest bidder extend a maximum of 10 years.

After the section receives an application to purchase natural products, a section appraiser determines the surface land value of the parcel in question. The land value is used to determine an annual minimum guaranteed royalty to the State trust. A minimum royalty rate is also established, which is expressed as a price per ton of material, the price at which bidding begins. The annual royalty and the price per ton are computed based on the following factors: location of the site; quality and quantity of material; marketability; access to the site and the market; surface land value; residual value of the extraction area; past production, if applicable; and length of the term. Once the annual royalty and price per ton are determined, notification of the auction is published in newspapers for 10 weeks. The successful bidder must pay all appropriate administration and advertising fees and submit a plan of operations to the section. Prior to pit entry, the purchaser must survey and stake the leased area, including the haul road.

There are currently more than 300 active sales agreements. During FY 1984-85, royalties from mineral-material sales totaled $1,154,819, a 7 percent increase over the previous year's total of $1,078,366.

Oil and Gas Leases

Leases are issued to allow exploration for and production of oil and gas. Although no producing wells are currently on State land, interest in exploration is active in Cochise, Coconino, Mohave, and Yuma Counties. During the last 5 years, the total acreage under lease has fluctuated greatly, but is now stable at approximately 1.6 million acres. The section issues noncompetitive leases on a first-come, first-served basis. If oil or gas is discovered on State land, a "known geological structure" of a producing field is identified by defining the trap or accumulation of oil or gas. All acreage presumed to have production potential is designated. This acreage is then subject to competitive leasing, which involves submission of sealed bids by interested applicants. After the section establishes leasing terms, the State Oil and Gas Conservation Commission regulates exploration and production procedures.

Leases are issued for an initial term of 5 years. If oil or gas is produced, the trust receives a royalty of 12½ percent of the value (excluding oil or gas used in operations), minus the cost of dehydrating the oil and extracting liquid products from the gas. Annual rentals for initial lease terms are $1.00/acre/year and increase to $1.50/acre/year if the lessee renews the lease. In FY 1984-85, oil and gas leases paid $1,233,164 in revenues to the State, a 20 percent decline from the FY 1983-84 total of $1,547,364. In FY 1979-80, oil and gas leases paid $1,822,144.

Geothermal Leases

Regulations regarding geothermal leasing on State land are essentially the same as those for oil and gas leasing, with one major difference: the initial term for geothermal leases is 10 years. The Oil and Gas Conservation Commission regulates geothermal exploration and production in Arizona. There is currently one geothermal lease on State land near Agua Caliente.

Special Land Use Permits

This permit category is normally administered by another unit in the State Land Department because a variety of land-use activities are included within it. The Nonrenewable Resources and Minerals Section does, however, process Special Land Use Permits (SLUPs) that allow large-scale geophysical surveys on land not otherwise leased by the applicant. In addition to the application fee, rentals are due based upon the total number of sections to be traversed during exploration. Statewide permits are also available within the SLUP category.

OTHER PROGRAMS

Land Exchanges

Selected State trust lands are currently being exchanged with Federal agencies, private firms, and individuals. The exchanges, which are administered by two other units within the State Land Department, are conducted for several reasons:

(1) To consolidate surface and subsurface ownership;
(2) To remove State ownership from national parks and monuments, wildlife refuges, and wilderness areas;
(3) To consolidate "checkerboard" land ownership for more effective land management; and
(4) To acquire high-value land that will provide revenue to the State trust.

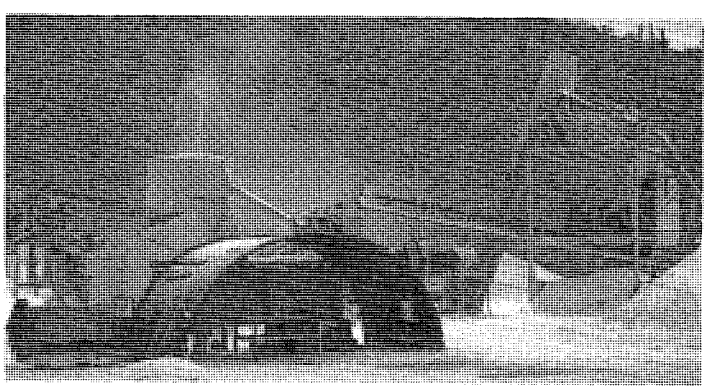
To support the program, the Nonrenewable Resources and Minerals Section researches the mineral potential of the land involved. State lands are not exchanged unless the land to be acquired is at least equal in value to the land to be sold. Major land exchanges allow the State Land Department to manage resources more effectively by "blocking up" landholdings. This involves trading sections of land that are dispersed throughout Arizona for contiguous sections of land.

Environmental Impact Studies

Prior to surface disturbance, lessees and permittees are required to submit a plan of operations for exploration or mining and for subsequent reclamation. The section determines potential impacts of proposed operations and works with lessees to determine appropriate land-reclamation activities.

Cultural Resource Program

The section represents the State Land Department in matters related to archaeological sites. Because the sites are protected by law, land-use planning must consider the location and significance of each site. The section has received several grants to conduct research at archaeological sites and to intervene when sites are threatened by leasing activities.



Crushing marble north of Tucson.

The Arizona Bureau of Geology and Mineral Technology: A Brief History

Mineral resources have played an important role in the settlement and economy of Arizona. Recognizing this importance, the U.S. Congress, the Governor of Arizona Territory, the University of Arizona, and the State legislature established the Arizona Bureau of Mines, which today is the Arizona Bureau of Geology and Mineral Technology. During the two decades before statehood, research and service activities by faculty members at the university served as the foundation for the Bureau.

Although Congress established the Territory of Arizona in 1863, the Civil War discouraged settlement until the 1870's, when the discovery and mining of mineral deposits brought an influx of newcomers. These deposits had been exploited by Spanish miners during the 1700's and by Indians before them. A number of mining camps were active by the early 1880's. At that time, little was known about the geology of Arizona; information that was available was general and of little use to prospectors. Despite this dearth of information, prospectors had little difficulty identifying mineral deposits because Arizona was not heavily vegetated and many deposits were exposed at the land surface.

Because of the abundance of these deposits, it was logical for Governor Tritle to request detailed geologic surveys of the territory in 1883. It was not until 1888, however, that the U.S. Congress created the post of "territorial geologist," which was filled on an irregular basis until statehood in 1912. The primary responsibility of this unpaid position was to prepare a summary of Arizona's geology and mineral resources for the territorial Governor's annual report to the Secretary of the Interior. Those who served as territorial geologist were John F. Blandy (1889-90), Dr. William P. Blake (1898-1904), and Dr. Cyrus F. Tolman, Jr. (1911-12). Although they did not carry the official title of territorial geologist, Dr. Theodore B. Comstock and Blake performed comparable duties during the periods 1891-94 and 1896-97, respectively. All but Blandy were faculty members in the University of Arizona School of Mines.

When it first opened in 1891, the University of Arizona was comprised of just two "schools": the School of Mines and the School of Agriculture. Comstock, who served as director of the School of Mines, established a metallurgical testing laboratory in 1893. The laboratory was referred to as the "University of Arizona Bureau of Mines" or the "University School of Mines Testing Laboratory." Rock and mineral determinations, ore testing, and complete assaying were done, largely by faculty in the School of Mines. Comstock served as director of the laboratory until 1895, when he became the first president of the university.

In 1915, three years after statehood, the Arizona legislature established the Arizona Bureau of Mines, an official State agency under the authority of the board of regents of the University and State Colleges of Arizona. The regents, in turn, specified that the new agency be administered by the University of Arizona, where a "Bureau of Mines" was already in operation. The functions of the territorial geologist and the University of Arizona "Bureau of Mines" were incorporated and expanded. Charles F. Willis, an instructor in geology and mining engineering, became the Bureau's first director in 1915 and served until 1918, when the College of Mining and Engineering was created. When Willis resigned, the dean of the college, Dr. Gurdon M. Butler, was named director of the Bureau. He served in that capacity until 1940.

JOHN F. BLANDY

John F. Blandy, first territorial geologist of Arizona, was one of the most respected mining engineers of his time. His education included two years at the School of Mines in Freiburg, Germany. His professional experience was notable: engineer at the Lake Superior copper mines for 3 years; president and general manager of the Little Schuylkill Company in the anthracite coal region of Pennsylvania for 15 years; author of several technical articles; and organizer of the American Institute of Mining Engineers.

Born in Newark, Delaware in 1833, Blandy moved to Prescott, Arizona in 1880, where he represented some of the largest mining companies in the country. He was frequently asked to compile mining statistics for the U.S. government, a job he also performed as territorial geologist of Arizona from 1889 to 1890. N. O. Murphy, acting Governor of Arizona in 1890, commended Blandy's role as territorial geologist: "In my judgment a more capable and efficient man for the office cannot be found in Arizona. He has both the experience and theoretical education, being a practical miner, as well as a graduate from a school of mining engineers." Blandy died in Prescott in 1903 at the age of 70.

At first, the Arizona Bureau of Mines served mainly as a mineral-testing facility (Figure 1). Gradually the Bureau became involved in activities comparable to those of State geological surveys. These activities included preparing geologic maps and studying relationships between the geologic framework and ore genesis. The major mineral deposits, present at or near the land surface, had been relatively easy to discover. Because they were being depleted and new deposits were much more difficult to locate, the Bureau's expanded activities were essential.

In 1940 the College of Mining and Engineering was divided into two separate colleges. Responsibility for the Bureau of Mines was assigned to the dean of the College of Mines, Dr. Thomas G. Chapman, who served until 1956. During this period, Bureau staff conducted research and pilot-plant studies that led to the design of the ore-concentrating facilities at five of the nine major low-grade copper deposits developed since 1945. Chapman was succeeded by Dr. James D. Forrester, who served until 1971, followed by Dr. William H. Dresher.

Dresher, who realized that the Bureau's enabling act was outdated, was primarily responsible for the writing and passage of the revised act in 1977. The new act changed the name to the Arizona Bureau of Geology and Mineral Technology and established two branches: the Geological Survey Branch, which is the Arizona State Geological Survey, and the Mineral Technology Branch, which continued the Bureau of Mines function. The Bureau's responsibilities were expanded to include research and information on geologic hazards and land-use limitations. For the first time, Arizona officially had a State geological survey. The new enabling act specified that the Bureau was to be a division of the University of Arizona administered by the Board of Regents and supervised by the president of the university or his designate. The president has continued to ask the dean of the College of Mines to supervise the Bureau, a practice that has been followed since 1918.

Dresher resigned in 1981 and was succeeded by Dr. Richard A. Swalin in 1984. In the interim period, Dr. William P. Cosart was acting dean of the College of Mines and acting director of the Bureau.

In July 1985 the separate colleges of Mines and Engineering were reunited as the College of Engineering and Mines, becoming one college as they had been from 1918 to 1940. Swalin was appointed dean of the merged college and continues to serve as director of the Bureau.

Current research by the Geological Survey Branch of the Bureau emphasizes improving the understanding of Arizona's geologic framework. This work includes the preparation and interpretation of detailed geologic maps. In much of the State, the geology has never been mapped in detail. Assessment of mineral potential and identification of mineral-exploration target areas requires

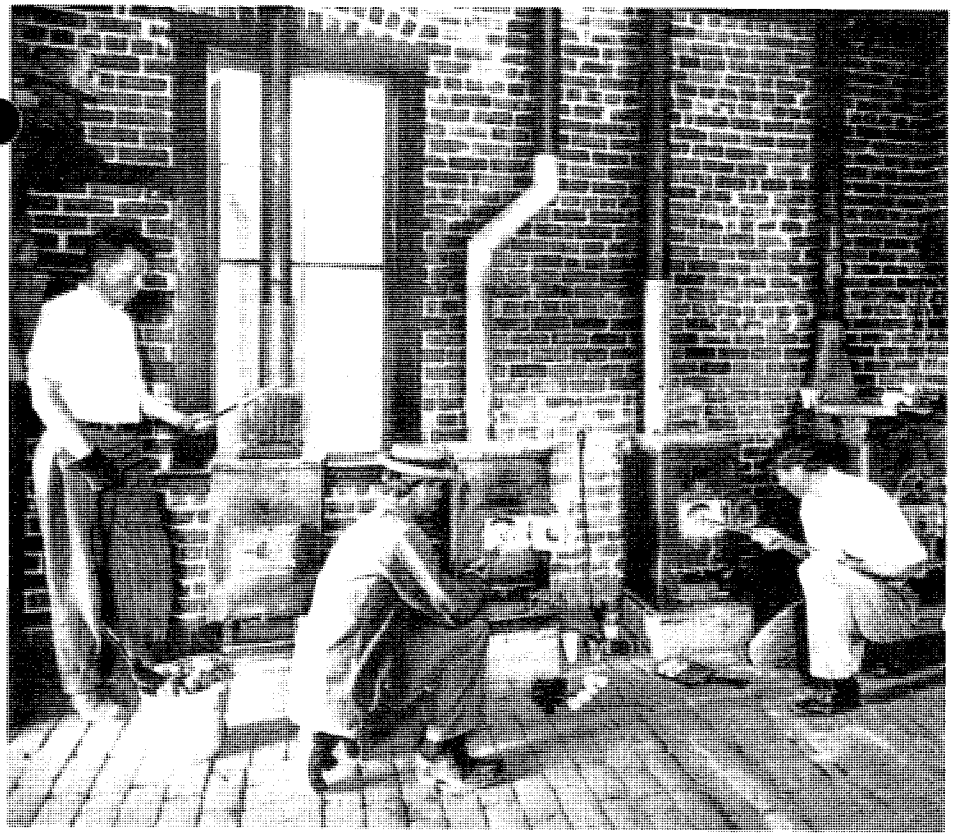


Figure 1. University of Arizona students conduct fire assays for gold and silver at the Arizona Bureau of Mines, circa 1920.

thorough understanding of the geologic framework and its relationship to ore genesis. Understanding of geologic hazards and land-use limitations is also based on knowledge of the geologic framework and detailed geologic maps. To make geologic information more available to the public, a computerized database, the Arizona Geologic Information System, is being compiled.

The Mineral Technology Branch is focusing current research on both basic and applied studies in mining, mineral processing, and extractive metallurgy. Mining research is contributing to two important areas: (1) the detection and characterization of gases produced by explosives in underground mining operations; and (2) the application of robotics in mining to eliminate personal injury and increase production. In the areas of mineral processing and extractive metallurgy, basic investigations are being conducted in the hydrometallurgical treatment of refractory gold and silver ores and the recovery of gold and silver from solution. A survey of Arizona copper dump-leaching activities is being made to determine the potential for recovery of minor elements from the leach solutions. Some of these metals may be of strategic importance to the State and Nation, as well as to copper producers as valuable by-products. In-situ leaching is another area in which the Mineral Technology Branch plans to develop experimental programs.

Selected References

Anthony, J. W., 1956, Arizona Bureau of Mines: unpublished manuscript, 7 p.

Moore, R. T., 1971, The Arizona Bureau of Mines, *in* Oakeshott, G. B., ed., Origin and development of the State geological surveys: Journal of the West, v. 10, no. 1, p. 136-141.

PROFESSIONAL MEETINGS

Frontiers in Geology and Ore Deposits of Arizona and the Southwest. Symposium and field trips, March 17-23, 1986, Tucson, Ariz. Contact AGS Conference, University Conference Dept., 1717 E. Speedway, Suite 3201, Tucson, AZ 85719; (602) 621-1232. (See also "Spring Symposium on Geology and Ore Deposits," p. 11.)

Arizona-Nevada Academy of Science. Annual meeting, Glendale (Phoenix), Ariz., April 19, 1986. Contact Jim Bales, 8643 E. Buena Terra, Scottsdale, AZ 85253; (602) 949-1549.

Cenozoic History of the Colorado Plateau Margin—the Stratigraphic Record. Rocky Mountain Section, Geological Society of America. Symposium and field trips, Flagstaff, Ariz., April 28—May 4, 1986. Contact Stanley S. Beus, Dept. of Geology, Box 6030, Northern Arizona Univ., Flagstaff, AZ 86011; (602) 523-4561.

STAFF NOTES

Janet Christner, recently promoted to Administrative Assistant I at the Bureau, is responsible for accounting and budget procedures, coordinates administrative paperwork, and acts as assistant to the head of the Geological Survey Branch. She has secretarial, accounting, and administrative experience and owns an Arizona real estate license. She is currently working on a B.A. degree at the University of Arizona.

Olga Hernandez, who oversees the Publications Sales Office of the Bureau, has been employed as Secretary II since July. She handles publication sales and distribution, answers phone inquiries, assists visitors, and performs various word processing tasks. She has 4 years of secretarial experience in personnel administration and is bilingual in Spanish and English.

Mark Pritzker has been appointed to a joint position in the Bureau's Mineral Technology Branch and the Department of Materials Science and Engineering at the University of Arizona. He received his Ph.D. in materials engineering science from the Virginia Polytechnic Institute and State University (VPI) in 1985. Dr. Pritzker's dissertation involved an electrochemical study of the solution and flotation chemistry of the mineral galena. He was the recipient of the 1985 Gillies Ph.D. Graduate Award from VPI for "leadership and great promise."

Dr. Pritzker received his B.S. in engineering from McGill University and his M.S. from the University of California at Berkeley. From 1978 to 1979, he worked for the Canada Centre for Mineral and Energy Technology (CANMET). His research interests include the electrochemistry and numerical modeling of processes associated with the flotation and hydrometallurgy of sulfide minerals and precious metals.

Margaret Stalker has been hired as Secretary III for the Mineral Technology Branch of the Bureau and the Mining and Mineral Resources Research Institute. Her job involves accounting, word processing, and conference coordination. Her previous experience includes personnel administration and promotional research. She holds a B.S. in business education from the University of Arizona.

Recent Publications on the Geology of Arizona

The following publications were recently added to the Bureau library, where they may be examined during regular working hours. Copies may also be obtained from the respective publishers.

U.S. Bureau of Mines

Bulletin

675 Mineral facts and problems, 1985, preprints: aluminum, cement, gem stones, graphite, helium, indium, quartz crystal, sand and gravel, silver, tellurium, thorium, and titanium.

Mineral Land Assessment Reports

MLA 22-85 Lane, M. E., 1985, Mineral investigation of the Arrastra Mountain Wilderness Study Area, La Paz, Mohave, and Yavapai Counties, and Peoples Canyon Wilderness Study Area, Yavapai County, Arizona, 37 p., 2 plates, scale 1:62,500.

MLA 34-85 Light, T. D., 1985, Mineral investigation of the Sierra Ancha Wilderness and Salome Study Area, Gila County, Arizona, 141 p., scale 1:48,000.

U.S. Geological Survey

Bulletins

1646 Tooker, E. W., ed., 1985, Geologic characteristics of sediment- and volcanic-hosted disseminated gold deposits—search for an occurrence model, 150 p.

1654 Tabor, R. W., Mark, R. K., and Wilson, R. H., 1985, Reproducibility of the K-Ar ages of rocks and minerals—an empirical approach, 5 p.

Maps

I-1310-C Drewes, Harald, Houser, B. B., Hedlund, D. C., Richter, D. H., Thorman, C. H., and Finnell, T. L., 1985, Geologic map of the Silver City 1° x 2° quadrangle, New Mexico and Arizona, scale 1:250,000.

I-1570 Drewes, Harald, 1985, Geologic map and structure sections of the Dos Cabezas quadrangle, Cochise County, Arizona, scale 1:24,000.

MF-1412-D Drewes, Harald, Moss, C. K., Watts, K. C., Jr., Forn, C. L., and Bigsby, P. R., 1983, Mineral resource potential map of the North End Roadless Area, Chiricahua Mountains, Cochise County, Arizona, 9 p., scale 1:50,000.

MF-1567-A Karlstrom, T. N. V., Billingsley, G. H., and McColly, Robert, 1983, Mineral resource potential and geologic map of the Rattlesnake Roadless Area, Coconino and Yavapai Counties, Arizona, 9 p., scale 1:24,000.

MF-1614-C Bankey, Viki, and Kleinkopf, M. D., 1985, Geophysical maps of the Whetstone Roadless Area, Cochise and Pima Counties, Arizona, scale 1:48,000.

MF-1644-A Conway, C. M., McColly, R. A., Marsh, S. P., Kulik, D. M., Martin, R. A., and Kilburn, J. E., 1983, Mineral resource potential map of the Hells Gate Roadless Area, Gila County, Arizona, 10 p., scale 1:48,000.

MF-1681 Gray, Floyd, Miller, R. J., Peterson, D. W., May, D. J., Tosdal, R. M., and Kahle, Katherine, 1985, Geologic map of the Growler Mountains, Pima and Maricopa Counties, Arizona, scale 1:62,500.

MF-1783 Brooks, W. E., 1985, Reconnaissance geologic map of part of McLendon Volcano, Yavapai County, Arizona, scale 1:24,000.

Open-File Reports

Studies of geology and hydrology in the Basin and Range Province, southwestern United States, for isolation of high-level radioactive waste [84-738 to 84-745]:

84-738 Bedinger, M. S., Sargent, K. A., and others, 1985, Basis of characterization and evaluation, 189 p., scale 1:2,500,000.

84-739 Bedinger, M. S., Sargent, K. A., and Langer, W. H., eds, 1985, Characterization of the Trans-Pecos region, Texas, 122 p., scale 1:1,000,000, 1:500,000, 7 plates.

84-740 Bedinger, M. S., Sargent, K. A., and Langer, W. H., eds., 1985, Characterization of the Rio Grande region, New Mexico and Texas, 148 p., scale 1:500,000, 7 plates.

84-741 Bedinger, M. S., Sargent, K. A., and Langer, W. H., eds., 1985, Characterization of the Sonoran region, Arizona, 141 p., scale 1:500,000, 5 plates.

84-742 Bedinger, M. S., Sargent, K. A., and Langer, W. H., eds., 1985, Characterization of the Sonoran region, California, 103 p., scale 1:250,000, 1:500,000, 6 plates.

84-743 Bedinger, M. S., Sargent, K. A., and Langer, W. H., eds., 1985, Characterization of the Death Valley region, Nevada and California, 173 p., 7 plates.

84-744 Bedinger, M. S., Sargent, K. A., and Langer, W. H., eds., 1985, Characterization of the Bonneville region, Utah and Nevada, 139 p., scale 1:500,000, 6 plates.

84-745 Bedinger, M. S., Sargent, K. A., and Langer, W. H., 1985, Evaluation of the regions, 195 p., 2 sheets.

84-830 Wilt, J. C., Keith, S. B., and Theodore, T. G., 1984, [1985], A listing and map showing molybdenum occurrences in Arizona, 62 p., scale 1:1,000,000.

85-28 Ege, J. R., 1985, Maps showing distribution, thickness, and depth of salt deposits of the United States, 11 p., 4 plates.

85-113 Chaffee, M. A., 1985, Geochemical evaluation of the Winchester Roadless Area, Cochise County, Arizona, 8 p.

85-394 Reimer, G. M., 1985, Helium soil gas survey of a collapse feature on the Hualapai Indian Reservation, Arizona, 12 p.

85-399 Gilmore, T. D., and Elliott, M. R., 1985, Sequentially and alternatively developed heights for two representative bench marks—near Palmdale, California and along the Bill Williams River, Arizona, 50 p.

85-400 Senterfit, R. M., Mohr, P., and Horton, R., 1985, Geophysical studies of breccia pipe locations on the Hualapai Indian Reservation, Arizona, 30 p.

85-410 Annual summary of ground-water conditions in Arizona, spring 1983 to spring 1984, 2 sheets.

85-462 Harms, T. F., Bradley, L. A., Tidball, R. R., Motooka, J. M., and Conklin, N. M., 1985, Analytical results and sample locality maps of stream sediments, heavy-mineral concentrates, and plant samples from Black Rock, Fishhooks, and Needles Eye Wilderness Study Areas, Graham and Gila Counties, Arizona, 49 p., scale 1:24,000, 3 plates.

85-469 Wenrich, K. J., Van Gosen, B. S., Balcer, R. A., Scott, J. H., Mascarenas, J. F., Bedinger, G. M., and Burmaster, Betsi, 1985, A mineralized breccia pipe in Mohawk Canyon, lithologic and geophysical logs, 72 p.

85-527 Peterson, J. A., Cox, D. P., and Gray, Floyd, 1985, Mineral resource assessment of the Ajo and Lukeville 1° by 2° quadrangles, Arizona, 77 p., scale 1:250,000, 3 plates.

Water-Supply Papers

2258 Hollett, K. J., 1985, Geohydrology and water resources of the Papago Farms-Great Plain area, Papago Indian Reservation, Arizona, and the upper Rio Sonoyta area, Sonora, Mexico, 44 p.

2271 Gilliom, R. J., Alexander, R. B., and Smith, R. A., 1985, Pesticides in the Nation's rivers, 1975-1980, and implications for future monitoring, 26 p.

Other Publications

Arthur D. Little, Inc., and Jones, Day, Reavis & Pogue, 1985, Impact of the President's tax reform proposals on the United States mining industry: 72 p.

Cather, S. M., and Johnson, B. D., 1984, Eocene tectonics and depositional setting of west-central New Mexico and eastern Arizona: New Mexico Bureau of Mines and Mineral Resources, Circular 192, 33 p.

Davis, G. M., 1985, Geology of the southern Plomosa Mountains: Tempe, Arizona State University, M.S. Thesis, 158 p., scale 1:24,000.

Dietz, David, and Williams, Charles, 1985, Geophysical investigations of Butler Valley, Arizona: University of Arizona Water Resources Research Center, 22 p.

Goodlin, T. C., and Mark, R. A., 1985, Geologic map of and cross sections through the Hot Springs Canyon area, Cochise County, Arizona: Tucson, University of Arizona, M.S. Thesis maps, scale 1:24,000, 2 sheets.

Myers, I. A., 1984, Geology and mineralization at the Cyclopic mine, Mohave County, Arizona: Las Vegas, University of Nevada, M.S. Thesis, 64 p., 6 sheets.

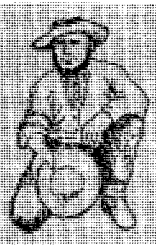
Owen-Joyce, S. J., 1984, Hydrology of a stream-aquifer system in the Camp Verde area, Yavapai County, Arizona: Arizona Department of Water Resources Bulletin 3, 60 p., scale 1:24,000, 3 plates.

Smith, R. C., 1984, Mineralogic and fluid-inclusion studies of epithermal gold-quartz veins in the Oatman district, northwestern Arizona: Tucson, University of Arizona, M.S. Thesis, 232 p., 1 sheet.

Stoneman, D. A., 1985, Structural geology of the Plomosa Pass area, northern Plomosa Mountains, La Paz County, Arizona: Tucson, University of Arizona, M.S. Thesis, 99 p., 4 plates.

U.S. Bureau of Land Management, 1985, Yuma district resource management plan and environmental impact statement: 310 p., scale 1:250,000.

Yeats, K. J., 1985, Geology and structure of the northern Dome Rock Mountains, La Paz County, Arizona: Tucson, University of Arizona, M.S. Thesis, 123 p., 3 plates.



Spring Symposium on Geology and Ore Deposits

On March 20-21, 1986 at the University of Arizona, Tucson, the Arizona Geological Society will host a symposium entitled, "Frontiers in Geology and Ore Deposits of Arizona and the Southwest." Fourteen separate preconference and postconference field trips, to be held on March 17-19, 22, and 23, will round out the conference.

The symposium will highlight recent advances in tectonics and ore deposits of the Southwest. Papers with a regional focus include overviews of Precambrian tectonics and ore deposits, Mesozoic structure and tectonics in western Arizona and the Southwest, Laramide magmatism and metallogeny, the regional nature of detachment faults in western Arizona, potassium metasomatism in the Southwest, mineralization related to detachment faults, preliminary CALCRUST results, and ore deposits of contiguous Mexico. Papers with a specific focus include mineralization at Picacho mine, Imperial County, Calif.; mineralization of the McCabe-Gladstone deposit, Yavapai County, Ariz.; and mineral deposits associated with topaz rhyolites in the Southwest.

Field trips include topics such as Precambrian evolution of the Payson-Mazatzal area, Mesozoic to mid-Tertiary tectonics and mineralization of west-central Arizona, stratigraphy and petroleum potential of the Pedregosa basin in southeastern Arizona, and geologic hazards and hydrology of southern Arizona. The fourteen field trips will give participants an opportunity for "hands-on" exposure to the geology of the Southwest.

Activities for spouses and guests will feature views of Arizona, from desert flora and fauna to the culture of its people.

For more information, contact AGS Conference, University Conference Dept., 1717 E. Speedway, Suite 3201, Tucson, AZ 85719; (602) 621-1232.

New Bureau Publications

The following open-file reports may be purchased over the counter or by mail from the Bureau offices at 845 N. Park Ave., Tucson, AZ 85719. Orders are shipped via UPS; street address is required for fastest delivery. All orders must be prepaid by check or money order made out to the Arizona Bureau of Geology and Mineral Technology. Shipping and handling charges are listed below. If your total order is

$1.01 to $5.00, add $1.75	40.01 to 50.00, add 7.75
5.01 to 10.00, add 2.25	50.01 to 100.00, add 10.00
10.01 to 20.00, add 4.25	More than 100.00, add 10%
20.01 to 30.00, add 5.50	Foreign mail, add 40%
30.01 to 40.00, add 6.25	

Chenoweth, W. L., and Learned, E. A., 1985, Historical review of uranium-vanadium production in the northern and western Carrizo Mountains, Apache County, Arizona: Open-File Report 85-13, 35 p., scale 1:126,000; text: $5.75; map: $2.00.

The camotite deposits of the northern and western Carrizo Mountains have been mined for their radium, vanadium, and uranium content since 1920. This report summarizes uranium and vanadium production in these mountains and provides hard data for a previously confused chapter in the history of uranium mining in Arizona. The report was made possible by the release of heretofore confidential data prepared by the Indian Trust Accounting Division of the General Services Administration for a court hearing in 1983 (Navajo Tribe vs. United States).

Capps, R. C., Reynolds, S. J., Kortemeier, C. P., Stimac, J. A., Scott, E. A., and Allen, G. B., 1985, Preliminary geologic maps of the eastern Big Horn and Belmont Mountains, west-central Arizona: Open-File Report 85-14, 25 p., scale 1:24,000, 2 sheets; text: $4.00; maps: $3.25 each.

This report presents preliminary 1:24,000-scale geologic maps of the eastern Big Horn and Belmont Mountains in west-central Arizona. The mapping, completed between January and April 1985, was jointly funded by the U.S. Geological Survey and the Arizona Bureau of Geology and Mineral Technology as part of the cost-sharing Cooperative Geologic Mapping Program (COGEOMAP). The Big Horn and Belmont Mountains were chosen because neither range had been previously mapped, except in broad reconnaissance for previous State geologic maps, and because both ranges have substantial mineralization and exploration activity.

The Big Horn and Belmont Mountains are composed of a metamorphic-plutonic basement that is overlain by middle Tertiary volcanic and sedimentary rocks. Volcanism was accompanied by low- to high-angle, normal faulting and rotation of the older volcanic units and subjacent crystalline basement. Slight to moderate angular unconformities within the volcanic sequence attest to synvolcanic tilting and faulting. The area contains a number of distinctive types of precious- and base-metal mineralization. Many mineral deposits in this area, including significant occurrences of gold, manganese, and barite-fluorite mineralization, are associated with middle Tertiary faults and intrusive-volcanic centers.

Chenoweth, W. L., 1985, Early vanadium-uranium mining in Monument Valley, Apache and Navajo Counties, Arizona and San Juan County, Utah: Open-File Report 85-15, 13 p.; $2.00.

This report summarizes the history of vanadium and uranium production in the Monument Valley area. The report contains previously confidential data prepared by the Indian Trust Accounting Division that are similar to data released in Open-File Report 85-13.

Cooperative Geologic Mapping

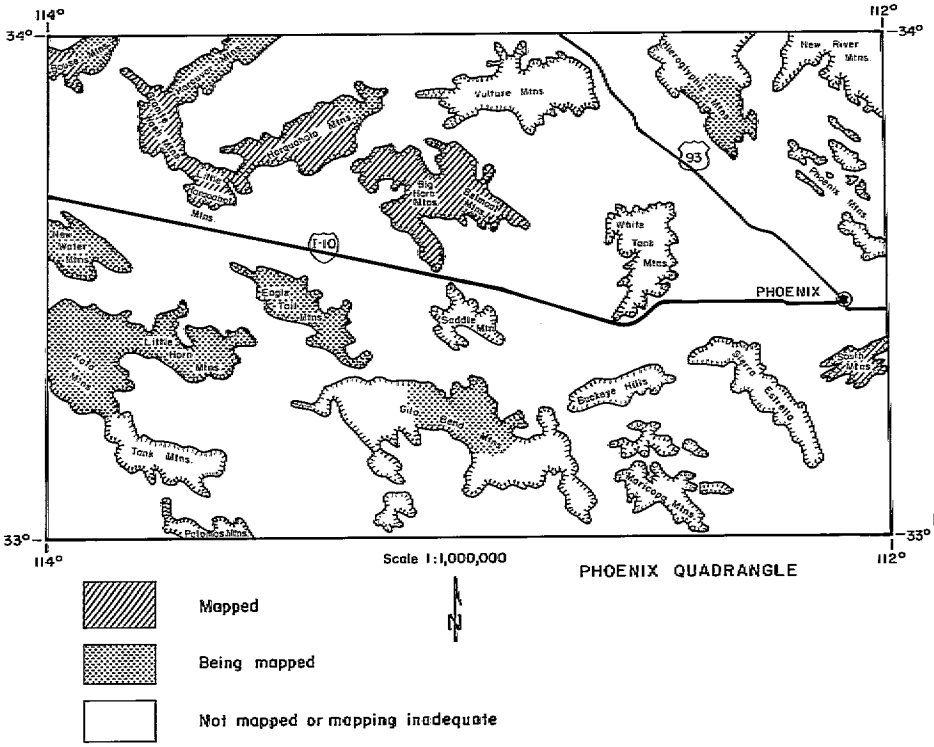
The Geological Survey Branch (GSB) of the Arizona Bureau of Geology and Mineral Technology has been awarded $63,204 from the U.S. Geological Survey (USGS) to support geologic mapping in the Hieroglyphic Mountains during fiscal year 1985-86. This award is the USGS's contribution to the Cooperative Geologic Mapping Program (COGEOMAP), a jointly funded effort with the State geological surveys. The GSB will provide an equal amount of funding through the mapping efforts of its own staff.

Dr. Stephen J. Reynolds (GSB) is principal investigator and project leader. Dr. David M. Miller serves as the COGEOMAP coordinator for the USGS, having succeeded Dr. Juergen Reinhardt. COGEOMAP is administered through the USGS Office of Regional Geology, under the direction of Dr. Eugene H. Roseboom.

This is the GSB's second year as a participant in the COGEOMAP program. During fiscal year 1984-85, GSB staff mapped in the Big Horn, Belmont, and Granite Wash Mountains and in the Bouse Hills, and USGS geologists mapped in the Kofa, New Water, and western Big Horn Mountains. A geologic map and descriptive summary of the Big Horn and Belmont Mountains, resulting from the 1984-85 COGEOMAP project, was recently released as Bureau Open-File Report 85-14 and is announced on page 11.

Dr. Stephen J. Reynolds.

Dr. Eugene H. Roseboom and Dr. David M. Miller.



Status of geologic mapping in the Phoenix quadrangle.

GEOSCIENCE DAZE COLLOQUIUM

The students of the Department of Geosciences, University of Arizona, will be holding their 14th annual *Geoscience Daze* on March 11-13, 1986 in the Gallagher Theater, Student Union Building. More than 50 papers will be presented on various aspects of geosciences. The public is cordially invited; admission is free. For more information, contact Mae Gustin, Geoscience Daze Committee, Dept. of Geosciences, Univ. of Arizona, Tucson, AZ 85721; (602) 621-6024.

Fieldnotes

Vol. 15, No. 4 — Winter 1985

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