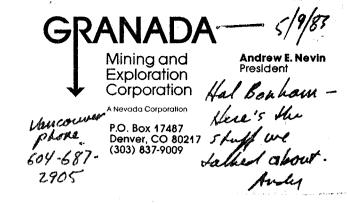
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METALLURGICAL REPORT
ON ORE FROM
SUNLIGHT MINE, NEVADA

ITEM 8

SEPTEMBER 1978

PREPARED FOR

PACIFIC EXPLORATION

BY



Ford, Bacon & Pavis Utah Inc.
The energy engineers.

375 CHIPETA WAY SALT LAKE CITY UTAH 84108 - 801 - 583-4773

SINCE 1894

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CHAPTER 1 INTRODUCTION

1.0 INTRODUCTION

1.1 Objectives of this Report

Pacific Exploration is engaged in a program to determine the economic viability and engineering tractability of processing ore from the Sunlight Mining Property near Fallon, Nevada for precious metals. The objectives of this report are to:

- (a) Present a concise summary of the historical aspects and previous assays.
- (b) Present and compare results of recent assays after pre-treatment of the ore to enhance recovery.
- (c) Discuss possible scenarios for commercial gold recovery processes with respect to bench-scale extraction techniques.
- (d) Provide working estimates for pilot and commercial plant capital and operating costs.

From this information a business plan can be generated which, to a first approximation, will allow cash flows to be estimated.

1.2 Sunlight Mining Co. Property

The property consists of 5 unpatented Placer claims comprising 160 acres each. It is located 28 mi north of Fallon, Nevada on U.S. Highway 95. The Southern Pacific Co. has rail-road tracks accompanied by power lines which are parallel to Highway 95, approximately 500 yards from the property.

There is more than adaquate water for processing purposes located on the property in as much as the water table is located 12 to 15 ft from the surface. The water is not potable; however, potable water can be found near the edge of the property.

The claims were located by John Peterson of Fallon, but are presently held under a lease-purchase contract by E. Anderson of Pacific Exploration.

1.3 Geology

The property is located in the "Carson Sink" area of Nevada. It is a dry lake bed dating from the late Pleistocene era (3-5,000 years ago). The entire area is underlain by alluvial deposits of material eroded from the higher mountain ranges of tertiary times. The overburden varies in depth from 2-20 ft and consists of a tough, semi-concreted conglomerate.

The water table is located 5 to 14 ft below the surface of the lake bed. As a result, the majority of the ore in the lake bed contains 40% water and can be mined by pumping it as a slurry. Because the ore is finely divided alluvial material it requires no crushing or grinding before treatment. The extent of the ore body has been estimated by core drilling at 40 million tons proven with an additional 20 million for reserve.

1.4 History of Claims and Previous Assays

The ore body has been core drilled in at least eighteen places and over 50 assay samples have been tested for gold and silver values. These early tests showed that gold was present in the range between about 0.4 and 30 oz/ton. Silver values were obtained up to 19 oz/ton.

These early tests summarized in table 1-1, were made by fire assays (F.A.), "Controlled and corrected" (C&C) fire assays, and cyanide methods (c) performed by the Nevada Assay office in Fallon, Nevada. (Copies of assays are in Appendix).

The higher values were initially <u>regarded</u> "hot spots", but later work suggests these values may be more representative than the lower ones.

The feasibility of an extraction process developed by John Peterson was studied during August and early September, 1978. The ore presented two impediments to successful recovery:

- (1) The absorptive guality of the clay prevents satisfactory separation of the pregnant cyanide solution, and
- (2) The gold appears to be very finely divided and encapsulated in alkali metal salts passivating them from direct attack by cyanide.

These problems were addressed during the study, with at least moderate success. The cyanide and amalgamation recovery tests used and the results obtained are shown in table 1-2.

From these tests it was concluded that at least commercial quantities of gold could be consistently extracted. Further work was curtailed in favor of methods proposed by R. Beyer and J. Hendersen of Provo, Utah which employ pretreatment to agglomerate the gold into larger particles which can then be recovered reliably by conventional methods.

It is interesting to note that this pretreatment technique was developed simultaneously by the two men working entirely independently and unknown to each other.

Mr. Jerry Hendersen was the first chemist to work on this ore. His report of August 10, 1978, (a copy of which is shown as Exhibit 1-1 indicated approximately 8 and 35 oz./ton (gold and silver) on two different samples. This report was greeted by most profound skepticism. The values seemed far too high, even though similar results had occasionally been obtained, they were not included in previous reports and were dismissed as a "hot spot". The dore beads extracted by Mr. Hendersen were sent to C.N. Parker & Sons, Denver, for assaying. Their assay confirmed that sample #1 had 6 oz/ton Au and 2 oz/ton Ag and sample #2 assayed 100% Ag.

Mr. Andersen of Pacific Exploration then contacted Mr. Rod Beyer, a chemical engineer at Ford, Bacon & Davis Utah, Inc. (FBDU) in Salt Lake City who had considerable experience in dealing with difficult ores. Mr. Beyer suggested that the ore be treated by a new technique he had been working on. At the same time a similar sample was to be given to Mr. Hendersen to evaluate.

A second set of samples were then delivered to R. Beyer and J. Hendersen. Four samples were taken from the same test hole from two different depths. The samples were then split and four samples then sent to each lab. The purpose of this series of tests was to determine whether the testing procedures were reliable and consistent. Work done by E. Andersen and J. Peterson indicated that values were distributed in a reasonably homogenous manner throughout all four samples. Also, a "hot spot" was deliberately chosen to give the lab every opportunity to pick up the values.

The "hot spot" turned out to be quite hot indeed. As per Exhibits 1-2 and 1-3, both direct and indirect average values reported by R. Beyer and Rogers Research utilizing a pretreatment process and subsequent analysis by X-Ray were 6.53 ounces gold/ton and 10.65 ounces silver per ton of ore. A subsequent report by Rogers Research dated 9-7-78 also indicated .075 ounces Platinum/ton, which may or may not be recoverable.

To confirm his initial test, Mr. Hendersen then performed a far more exhaustive test on another composite sample and recovered 11.553 oz Au and 55.765 Ag/ton of ore as shown in Exhibit 1-4. These results were indicative that further development of these recovery methods on this ore is definitely warranted.

Table 1-1 Assays from Nevada Assay Office Fallon, Nevada

Sample No.	Method		Au, oz/ton
6550 (concentrates 6550 (concentrates 6550 (tails) 3653 3653 3653 3842 6867 6869 6550	5) F.A.		.069 .565 5.22 .034 .197 2.62 22.72 .534 .462 .768
6551 6071	C&C F.A.	•	.405 29.89
		Avq.	5.29

Note: Copies of the actual reports are shown in the Appendix

TABLE 1-2
Results of Cyanidation Assays by J. Peterson, Fallon, Nevada

Test No.	Type	Result,	oz/ton of Au
4-12-40-(2)	Cyanide		0.145
4-12-40-(3)	Cyanide/Charcoal		0.251
4-12-40-(4)	Cyanide/NaBr		0.227
4-12-40-(5)	Cyanide/Dow		0.235
4-12-40-(6)	Cyanide/Calgonite		0.287
4-12-40	Hg Amalgamation	•	1.17
4-15 Mill Feed	Hg Amalgamation		0.312
		Avg.	0.375



651 Calambia Lane - Perco, Unit 84601 - 375-6960 P. O. Box 361 - American Fork, Unit 84003

Pacific ExploRation
1143 Mariposa
San FRANCISCO, Ca. 9410.

att. Exic and ERECN

San1 P/E # 1=

Code # 8-1-40 SunLight MI

Two (2) assay tens of ore Fused - Hi Caustie Leached with acid & Filtered. Solution & Filter WERE RERUN and assayed Total wt. of Metal 15,30 mg. CR 7.66 3/TON

Sample #2

Brights are Leach with agua Regis for J4h.

Filtered and Solution Elaparated To 50Ml.

His added To 400Ml. Warned & Run Throis

Special Resin. Resin Dried-Bured & Resider

assay ed.

Total Metal 35.00 Mg. OR 35 03/TON.

Have one more test would like to fun bu Do Not have the time light Now. Will Try to Run it Next WEEK. Test Takes 5 days.

Jerry & Hondon

Ford, Bacon & Davis Utah Inc.

ENGINEERS - CONSTRUCTORS

A SUBSIDIARY OF

Ford, Wacon & Davis

375 Chipeta Way P. O. Box 8009 Salt Lake City, Utah 84108 801-583-3773

SUNLIGHT MINE ORE

Assay Report from Clare Rogers (Rogers Research Inc, SLC)

7 September 1978

PROCEDURE FOR ASSAY

- A. Sample is ground first in a steel crucible to 75-100 mesh (if required).
- B. Sample is further reduced in size to +375 -200 by motorized agate pestle and mortar.
- C. A 1-2 gram sample is placed in a cup 1" dia. \times 1/8" deep and subjected to X-Ray beam from Molybdeneum target.
- D. Flourescence pattern from NaCl crystal is obtained. (NaCl is known crystal structure with known delta spacing)
- E. Equation used to obtain N λ (wavelength):

$N\lambda = 2d \sin \theta$

F. $\lambda_1, \lambda_2, \lambda_3$, etc. compared to known values of elements. Peak height of signal compared to known standard in same range of concentration to get quantitate amount.

PROCEDURE FOR TREATMENT OF ORE

- A. Ore ground, if required, to 75-100 mesh.
- B.

 THESE ITEMS DELETED TO SAFEGUARD PROPRIETARY TECHNOLOGY
- D. Ore mix removed and ground to powder if necessary.

TEST RESULTS

Results of tests on Ore supplied by Jack Jutzy, Geologist.

Sam	ple_Number	Gold	Silver
1.	untreated	0.811	4.23
1.	treated	2.89	8.11
4.	(Sunlight) untreated	2.13	5.13
4.	(Sunlight) treated	9.52	12.52
4.	(Sunlight) treated	12+	20+
5.	untreated (, ,)	2.02	5.06
5.	treated (deleted)	5.12	8.12
5.	supertreated .	7.18	11.32
	2 hours)		

DISCUSSION OF RESULTS

The sample treated deletid substantially higher values in both gold and silver, indicating that (deleted) for these sample pretreatments. This sample will be redone to obtain a more definitive assay.

It is of interest that the values of the untreated ore are not unsimilar to values reported previously for these same materials using other techniques. It is believed that the values reported represent conservative figures. (Data reported by J. Hendersom, 8-30 values from Sunlight Mine of 11.5 oz Au and 55.7 oz Ag. from physical extraction.) Sample #1 was obtained near the surface, the other two were from the ore taken at a greater depth.

R. B. Rever. Metalluroist

Note: All of the above values, except for the last "#4 (Sunlight) treated", are low by a factor of 2.50 due to a weight correction not included in the calculation at the time of assay.

MAIN OFFICE GB SOUTH MAIN STREET SALT LAKE CITY LITAH RAID! PHONE 18011 ACCOUNTS \$ \$1013 1

HOME OFFICE 551NORTH 1100 EAST BOUNTIFUL UTAH H4010 PHONE 18011 795 4402

RESEARCH & ANALYSIS INC.

CLAIR W. ROGERS, President

September 7, 1978

Ford Bacon & Davis C/O Rod Byer Salt Lake City, Utah

Customers Identification: Quantitative Analysis of Gold & Silver

Sunlite Mine 1. #5 Supertreated	Gold 7.18 O/T	Silver 11.32 O/T
2. #5 Untreated	2.02 O/T	5.06 o/T
3. #5 Treated	5.12 O/T	8.12 O/T
4. #4 Untreated	2.13 o/T	5.13 O/T
5. #4 Treated	9.52 O/T	12.12 O/T
6. #1 Untreated	.811 o/T	4.23 o/T
7. #1 Treated	2.89 O/T	8.11 o/T

These values are low by a factor of 2.50 because a weight correction was not made.

RBB

651 Colombia Lane - Perco, Unit 84601 - 375-6960
 P.O. Box 361 - American Fock, Unit 84003

August 30, 1973

Pacific Exploration 1143 Mariposa San Francisco, CA 94107

SUBJECT: Ore Sample From Sunlight Mine - Sec. 8, Hole #1 Millfeed 35' to 40'

Assay Results of One-Pound Test: (Total metals recovery after in-quart taken out)

Silver (Ag) - 55.765 oz. per ton Gold (Au) - 11.553 oz. per ton

ACTUAL METAL BUTTONS

Smelting atused as a collector.

Lead was

Slag ground and leached.

Re-smelted at 2100° F. with lead as collector. Inquart 2 grams silver.

1st and 2nd smelt of lead collector remelted, poured into bar.

Electrolytic process was used for breaking down lead bar.

Anode and cathode, plus solution, was re-worked through electrolytic three (3) times.

Total metals recovery after inquart was taken out shown above in "Assay Results".

Jerry C. Henderson

NOTE: Silver and gold recovered in this test is enclosed.

Brokers of New and Used Equipment For All Mining Needs

CHAPTER 2 RECENT ASSAYS AND RECOVERY METHODS

2.0 RECENT ASSAYS AND RECOVERY METHODS

2.1 Gold Assay Techniques and Limitations (Monograph by R. Beyer)

2.1.1 Introduction

A-sample of ore from the Sunlight Mine was recently submitted for analysis, and we found our experiences with this material worthy of extended comment. The ore is unusual in that it is actually a pleistocene lake bottom mud, which according to present theory and past experience is a most unlikely place to find high gold values. It is challenging in that heretofore it has been extremely difficult to assay.

Methods for the analysis for gold (and other precious metals) basically fall into two categories - direct and indirect. The direct approach is to physically, or chemically, remove the gold from the sample of ore by one of several well-known techniques, such as fire assay. The indirect method is to ascertain the amount of gold instrumentally, using primary standards for comparison. In these methods a measure of the amount of gold is determined from a spectrograph or crystalograph, such as "AA" (atomic absorption) and X-Ray fluorescence or diffraction.

The fire assay, because of its incontrovertable result, namely a piece of metal that can be felt, weighed, tested, etc. has been the favorite method for years of many metallurgists and prospectors. However, the fire assay technique, in which gold is first "inquarted" into a base metal, usually lead, and then "purified" by absorbing the lead into a bone-ash cupel, is not reliable for many ores now being processed commercially.

Much controversy has arisen over the various methods now employed - principally because they don't all agree with each other. Which method, if any, is right? Or are they all wrong? These are basic questions which must be resolved. The author has spent many hours assaying gold and silver ores himself, and has enlisted Drs. Lynn Kimball and Jack Ruckman of Provo and Clair W. Rogers of Salt Lake City to assist in the evaluation of the various techniques. Kimball and Ruckman are inorganic chemists well versed in AA spectroscopy and similar "wet chemical" assay methods. Rogers has specialized for over 20 years in X-Ray spectroscopy both diffraction and fluorescence. This paper is a first attempt at trying to explain some of the observations and inconsistencies that we are finding among the several assay techniques.

2.1.2 Case Histories

1. A sample of clay ore was analyzed by Dr. Ruckman using a standard atomic absorption method; i.e., dissolve in aquaregia, partition with MIBK and run in the spectrograph. The results were negative. Similar samples were run by Dr. Kimball

by a fire assay method. He also showed nothing. Mr. Rogers found 5 oz/ton with his X-Ray analysis. Kimball turned to a chemical method and eventually showed gold to be present, Ruckman was experimenting with his agua regia - MIBK sample, and after mixing with some additional organic liquids to see if he could get a phase separation, he observed that gold precipitated as a black colloidal solid at the interface. He pipetted off the gold as it formed, redissolved it in aqua regia and treated it again with fresh MIBK. This time he measured about 2.5 oz/ton of gold by atomic absorption. He doesn't know why this happened, but there was actually gold there, either in the acid or organic fraction, that he could not detect initially with the He doesn't yet know why the gold precipitated but suspects it reduced from an organic complex of some type by reaction with one of the chemicals present.

j

- 2. A second sample of ore was assayed by Kimball, Ruckman, and Rogers. This time the sample was separated into five fractions: original (as-mined); concentrates from a table; heads, tails, and middlings. Rogers obtained high values of gold in all samples. Kimball and Ruckman, on the other hand, found only good values in the concentrates, small quantities in the heads, and near zero values in the middlings and tails. No one could explain why this occurred.
- 3. Ruckman was boiling an ore sample in aqua regia in the hood on a hot plate. He observed gold being plated out on the side of an aluminum pan nearby. He became concerned as to how much gold was being lost from the solution in this manner. He collected the gold from the aluminum and weighed it. There was a substantial amount.
- 4. T. K. Rose, in his book "Metallurgy of Gold" (1894), states that many an assayer's hood and chimney have been cleaned and found to contain much gold among the soot.
- 5. Rogers reports that a group of mining people hired him to analyze some ore they were investigating. Others were also enlisted to give comparable assays. After a comprehensive series of assays and about \$3,000 of work by Rogers, the miners decided not to pay Rogers for his work stating that his analyses had to be "way too high" compared with the other analyses that were made. Rogers told them that he wouldn't sue for his charges, but put them on their honor to pay him, if in retrospect, they found out he was correct. A year and a half later he received a check from them for \$3,000. When Rogers inquired as to why they paid him, he was told that the amount of gold recovered from the commercial process exceeded even the value Rogers had reported. In fact, Rogers' value had been conservative, just as he had predicted earlier.
- 6. Rogers observed some interesting phenomena while roasting a sample of ore at moderate temperatures (up to $1500^{
 m OF}$). Samples withdrawn from the furnace were broken into

small fragments and examined under a microscope. Whereas no visible evidence of gold could be found prior to roasting, afterward, small rivulets or beads of the yellow metal appeared agglomerated into visible proof that gold was present. Much care had to be taken not to heat the ore too rapidly to a high temperature or the gold would be lost by vaporization.

7- Although the mechanism is unclear, some assayers, including Mr. Jerry Henderson, claim that in the presence of certain metallic elements such as selenium, gold is absorbed into the bone-ash cupel during a fire assay. It can be recovered to some extent with difficulty by the so-called "controlled and corrected" method wherein the cupel itself is subjected to assay.

Other examples similar to those above could be cited, but the point is clear: Fire assays and chemical assays don't always give the correct amount of gold in the sample. Whether all of the gold assayed by X-Ray techniques can be recovered economically in a commercial plant is another question. We will examine that aspect in a section 4.

2.1.3 The Occurrence of Gold in Nature

Clues to the assay mystery may be found by examining how gold is dispersed in its natural state. We know that gold occurs in several forms: as free metal, as alloys with other metals, as compounds such as telurides, and as so-called "complex" ores which are thought to be compounds with organic substances. A chelate may be one possibility.

Free gold has been observed to occur over a wide range of sizes from large masses weighing a number of pounds to exceedingly fine particles. The amount occuring in each size fraction cannot be accurately determined, of course, but it supposed that it occurs somewhat like the distribution curve shown in Figure 2-1.

This curve represents that small amounts occur naturally as large nuggets and the quantity at progressively smaller sizes increases. Obviously, the size cannot decrease below the size of the atom (10^{-4} m) . Although the subject needs more research, it is thought that very finely divided gold may have been deposited in some ores from solution or from the vapor state, and in many instances has combined with organic matter to form single-atom (chelated) compounds. Unreacted or free gold is believed to occur as ultra-thin colloidal films. In many deposits micro-sized gold has been found encapsulated in alkaline salts.

2.1.4 Hypothesis

After reviewing the experiences of Kimball, Ruckman and Rogers, the following hypothesis is put forward:

- 1. Gold in relatively large sizes, as alloys, and as compounds with high melting points are readily assayed by standard fire assays and most chemical methods. In these cases, both the chemical and physical assays agree with the X-Ray method: Gold in this form, can be separated and recovered commercially by most of the standard methods, i.e., tabling, flotation, cyanidation, amalgamation, etc.
- 2. Gold in smaller sizes, i.e., colloidal particles of 20-1000 microns, for example, is sufficiently small that it is easily vaporized. Rogers reports that he has observed thermal loss of gold at temperatures as low as 300°C. It may go lower. When contained as part of an organic molecule the oxidizing condition of a fire assay treatment would enhance the vaporization. Most of the very finely divided gold is not picked up in a fire assay, because within seconds after an ore sample is introduced into the 2000+°F furnace, the gold flash vaporizes before the inquarting agent has an opportunity to alloy with it. Chemical analyses appear to be more accurate if the ore is first ground or milled sufficiently fine, to free the gold and make it available to acid dissolution.
- 3. As gold becomes even more finely divided (i.e., as colloidal films, encapsulated colloidal particles, or organic chelates) it more easily vaporizes, even from solution (as an organic specie with high vapor pressure). In the encapsulated form it may even have a tendency to combine with the slag. The X-Ray method cannot detect gold in this finely divided form. Rogers reports that colloidal films are too thin to reflect the X-Rays for reliable quantitative treatment.
- 4. If the gold in the finely divided-state is agglomerated by a suitable process then the gold can apparently be detected by X-Ray methods first, chemical methods second, finally by fire assay as the agglomerates become larger. Thus far, only a limited number of comparisons between methods have been obtained after agglomeration. These look very encouraging. Even after agglomeration there may still be chemical interferences that mask the presence of gold in the fire and chemical assays of some ores.

2.1.5 Conclusions

It can certainly be stated that much research is required for a complete understanding causing the differences between the gold assay methods. However, in reviewing in detail the X-Ray spectroscopy method, there are a number of advantages of this method over the others:

- (a) Sample is used without treatment other than sizing (-200+300 mesh).
- (b) Good separation between elements are obtained. Even isotopes can be detected and identified accurately.
- (c) No interfering species mask read out.
- (d) Can accurately read-out to very small concentrations (0.002% or 0.6 oz/ton).
- (e) Can read accurately even with very fine particle sizes (above 10 microns).

It does have some limitations:

- (a) Extremely small (submicroscopic) particles such as colloidal films don't reflect X-Rays quantitatively.
- (b) In heavy metal ores, it is necessary to add an "enhancer" (i.e., to lower the density) before quantitative measurements can be made.

It is clear that in many cases the fire assay and the chemical assay don't measure the gold present in a reliable manner. Often, gold has been extracted from the bone-ash cupels, which after firing has shown no results. The gold was absorbed into the cupel along with the lead. Sometimes over 100 mg has been recovered per assay ton after successive extraction from ten or more cupels.

In summary, the X-Ray fluorescence and diffraction technique appears to be significantly more reliable and not subject to chemical interferences, as are both wet chemical and fire assay methods. For micro-sized particles (below 10 microns) even X-Ray techniques are limited. The best approach is to agglomerate the fine particles by a thermal treatment to obtain sizes which the X-Ray method can detect. It is believed that economical commercial methods may also take the same approach.

2.2 Sample Preparation

Three samples of Sunlight ore were submitted for testing and analysis on August 28, 1978. On September 6, 1978 thirteen additional samples were submitted for evaluation.

These samples were first ground, when necessary to approximately 100 mesh. Many of the samples were quite damp when received. They were reported to have had up to 40% moisture. No attempt was made to dry them before analysis. Eight ounces of the ore was weighed out and mixed thoroughly with 12 ounces of pre-treatment mix. The samples were then roasted at various temperatures for various lengths of time between 1 and 3 hours. The samples were then submitted for assays by X-ray fluorescence X-ray diffraction, atomic absorption, acid leach, and cyanide extraction.

2.3 X-ray Fluorescence

Samples of both treated and untreated ore were submitted to Rogers Research in Salt Lake City. Other than sizing the sample in a pestle and morter to +300-200 mesh, no further treatment was required. The samples were placed in a shallow metal container and subjected to a beam of X-ray from a molybdenum target diffracted through salt (NaCl). The characteristic wave lengths of the elements in the unknown sample ore calculated from the Bragg equation viz: $=2d\sin$ where d is the lattice spacing distance of the salt crystal, and represents the angle of diffraction. The harmonics of the major frequency are mutiples of , or n . Excellent separation between the characteristic frequencies of the elements are obtained. Even isotopes can be easily identified and there are no interfering elements to mask anothers presence.

The results of the analyses are presented in Table 2.1.

At first glance, the analyses of the treated samples appear to vary from 1.28 oz/ton up to a high of 25 oz/ton. However, the temperature for the last 13 samples never reached the maximum that the first batch did, and apparently incomplete conversion occurred. Some of these will be selected to undergo further thermal treatment and be reassayed.

On the other hand, even the untreated samples show remarkably high gold values. The results indicate that the thermal reaction enhances the assay results by a factor of between 3 and 5 times. This is not surprising on the basis of similar assays on other ore bodies.

2.4 Atomic Absorption

These tests were conducted by Dr. Jack Ruckman of Provo. Utah. After first dissolving duplicate 1 gm samples of ore in concentrated aqua regia and digesting them at near boiling for

24 or more hours, the solution was treated with MIBK to strip off the resulting gold chloride and tested on the AA spectograph. Results of these tests are shown in Table 2-2.

While these results are not at all similar to the X-ray assays, other samples of a similar type ore, gave nearly identical results to the values shown in Table 2-2 on all but one of the tests. This one sample gave a reading of zero on one replicate and its' duplicate was 22.8 oz/ton. An X-ray analysis of the same sample previously showed gold at 21.3 oz/ton. While Dr. Ruckman has no explanation for this discrepancy at the moment, it is possible that gold forms an insoluble complex that prevents it from reliably transferring to the organic solvent phase, as discussed in Section 2.1. Further analysis is continuing with these samples.

2.5 Fire Assays

Several of the 13 semi-treated and untreated samples were submitted to Union Assay Office in Salt Lake City for a standard fire assay. The results of these tests are presented in Exhibit 2-1.

The negative results obtained from these fire assays are not at all surprising. First of all, the gold in this ore, according to J. Henderson, exhibits definite complexing qualities and therefore reports principally in the vapor and slag.

Mr. Henderson ran both a standard and a special fire assay on samples No. C-2, C-3, C-4, and C-5. He reported, as shown in Exhibit 2-2, that the standard assay yielded nothing, but the special assay, in which he used silver instead of lead as an inquartering agent, yielded 2.2 oz/ton of gold -- a decided improvement, but far from the total amount shown to be present by his other analysis and by X-ray analysis.

2.6 X-ray Diffaction

Rogers Research conducted crystalographic examination by X-ray diffraction of Sample R #1 (SM #5). Results of the X-ray diffraction pattern, in which the d-spacing of the lattices experimentally found are compared to a known standard, are shown in Exhibit 2-3 for the untreated ore and in Exhibit 2-4 for the treated ore. Several items are noted: first, water of hydration is lost from the three minerals originally present; second, the carbonate is decomposed (to CO₂); the Calcium Silicate hydrate decomposed to alpha quartz (SiO₂); and lastly, the percentage of noble metals detected increased markedly, particularly the gold. This material is typical lake bed clay except for the exceptionally high values of noble metals.

2.7 ACTUAL RECOVERY METHODS

Assays, at best, are but a crude indication of the values that are recoverable from any given ore. An ore can assay 25 oz/ton or 1000 oz/ton in any given element, but if it is economically or physically possible to recover only 0.25 oz/ton then that is the basis on which financial and engineering considerations must be made.

On the other hand, when the precious metals are actually recovered by a process similar to that contemplated for commercial extraction, then there can no longer be a question as to how much can be recovered. This is the reason for performing the chemical leach analyses at this point in time. It is a potential candidate as an extraction method. The results from a dilute aqua regia leach are presented in Exhibit 2-2.

Although results from samples #2 and #3 are lower than the X-ray values obtained from these samples, results from #4 and #5 are closer to the average X-ray values obtained from these samples. (8.7 oz/ton.) Even though the acid leach method is a substantially more expensive process than a cyanide leach, it certainly emerges as an acceptable alternative to the cyanide method.

2.8 SUMMARY OF ASSAYS

There have been several sets of assays reported for the Sunlight Mines. Early assays for gold, as reported in Chapter 1, are summarize as follows:

1.	Fire assays and cyanidation, 12 tests (Nevada Assay Office)	5.29
2.	Cyanidation and Amalgamation, 7 tests (J. Peterson and E. Anderson)	0.375
3.	Caustic Fusion - Acid leach, 1 test (J. Henderson)	6.0
4.	Reductive Roast - silver inquart, electroplate, l test (J. Henderson)	11.55
5.	Thermal pretreatment, X-ray fluorescence, 3 tests (Rogers Research, Inc.)	17.13
	Straight Average of previous tests Weighted Average	8.07 5.62

The values of silver in the ore varied from a low of 3 to a high of 55 oz per ton with the average being slightly higher than that for the gold. Platinum is also present at values between about 0.1 and 0.3 oz/ton. Some of the above samples were deliberately selected from known "hot spots" to obtain the

best showing possible.

The latest set of assays reported in this chapter were obtained to determine an over-all assessment of the entire ore body and the values that could be reasonably expected from actual operation of the mine. Care was taken to not single out any "hot spot".

The sampling procedure included taking a core sample from the center of each of the 13 mill feed ponds on the property. The weight of each core sample was about 10 pounds. Each ore sample was well mixed and great care was taken to obtain a representative sample. Mr. Jack Jutzy, the geologist who took the samples, reports that he is "confident that the samples were correct and give a good approximation of the overall values present."(1)

The average of the assays reported for the 13 millfeed samples are summarized as follows:

Method	Au, oz/ton	Ag, oz/ton	Pt, oz/ton
Pretreatment - X-ray fluorescence	5.12 <u>+</u> 3.6	5.03	0.067
Atomic Absorption	0-0.1	un	
Fire Assay	trace	0.2	the ma
Ag inquart - fire assay	2.12 <u>+</u> 0.07	en en	~~
Agua Regia leach	4.35 <u>+</u> 3.1		
X-ray diffraction (treated)	11+01	7.5	0.5 + .
X-ray florescence (untreated)	1.2 <u>+</u> 0.7	1.64	0.015

Disregarding the fire assays and AA results due to their inherent inability to pick up fine gold as per previous discussions, the overall average of the other results is 5.65 oz/ton for gold for the thermally pretreated samples. The silver values are about the same. It is interesting to note that this average is almost exactly identical to the weighted average shown above for all of the previous assays (5.62 oz/ton) (which minimizes the effect of "hot spots").

⁽¹⁾ Jack J. Jutzy - Personal communication to Eric Anderson, September 25, 1978.

It should also be pointed out that many of the samples thermally pretreated did not reach the maximum treatment temperature and may not have converted completely. Therefore, the above assay results for X-ray fluorescence should be conservative.

Inasmuch as both the silver inquart fire assay and the agua regia leach ended up with a button of bullion gold at the end of the test, we feel confident that values in this magnitude should be recoverable by a commercial process. Methods to prove this recovery are discussed in the next chapter.

TABLE 2-1
X-RAY FLUORESCENCE RESULTS (1)

RRA#82178 -	1 -	17;	1700	OF,	2	Hrs.
-------------	-----	-----	------	-----	---	------

	mple No.	Au, oz,	(u)	Ag, o (t)	z/ton (u)	Pt, 0:	z/ton (u)
R#1 (S	SM#5)	18.84	5.30	29.72	13.28	.283	
R#4 (5	SM#1)	7.58	2.12	21.28	11.10	.060	****
R#5 (S	5M#4)	24.99	5.59	31.82	13.47	.294	-

RRA #9978 - 1 - 13; 1000 OF

				· ·			
C#2, 1 hr 50 min	7.62		6.85	**	.097	-	
C#3, 1 hr 50 min	8.95		7.65		.105		
C#4, 1 hr 50 min	12.13	··· is	5.42		.125		
C#5, 1 hr 50 min	6.15	two state	4.12		.095		
C#6, 1 hr 45 min	4.25	***	3.25		.084		
C#9, 1 hr 40 min	5.12	****	6.14	Mark prope	.065		
C#10, 1 hr 35 min	1.28		5.11	-			
C#11, 1 hr 30 min	2.18		4.03				
C#12, 1 hr 20 min	1.66		4.42			***	
C#13, 1 hr 10 min	1.81	*** ***	3.31				
,							

Unreacted (low heat)

C#14,	2 hr 3	15 min	 2.03	 2.11	***	.025
C#15,	2 hr	15 min	.855			
C#16,	2 hr .	15 min	 .715	 .912		.009

t = treated

u = untreated

⁽¹⁾ Copies of assay reports are in the Appendix.

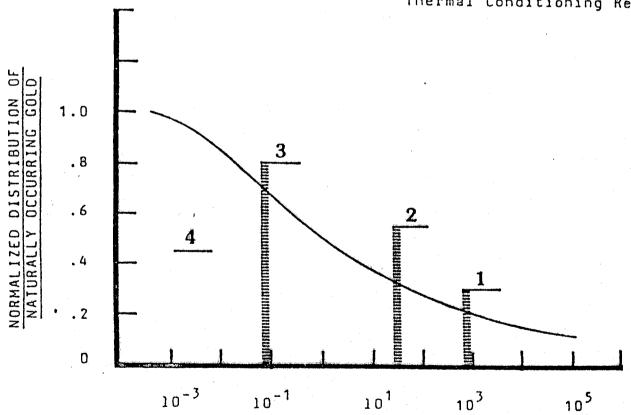
TABLE 2-2
ATOMIC ABSORPTION RESULTS
(treated samples only)

Samp	le No.	<u>Au, oz</u> (a)	/ton (b)
R#,1	(SM#5)	0	0
R#4	(SM#1)	0.009	0.099
R#5	(SM#4)	0	O

figure 2-1

Zone

- FIRE ASSAY LIMIT
 Physical Recovery Methods
- 2 CHEMICAL ASSAY LIMIT Chemical Recovery Methods
- X-RAY ASSAY LIMIT
 Special Recovery Methods
- 4 SUB-COLLOIDAL OR MICROFINE Thermal Conditioning Required



PARTICLE SIZE IN MICRONS

Exhibit 2-1

Hand Sample Serial....13436-13442

Telephone 363-3302

Marges \$ 42.00

ASSAY REPORT

UNION ASSAY OFFICE, Inc. Beyer Corp W. C. WANLASS, President 1824 Pine Lane G. P. WILLIAMS, Vice President Provo. UT RESULTS PER TON OF 2000 POUNDS A. S. JOLLIFFE, Treasurer GERALDINE A. WANLASS, Secretary P. O. Box 1528 Salt Lake City, Utah 84110 Sept 15, 1978 GOLD SILVER LEAD NUMBER COPPER INSOL. ZINC SULPHUR IRON LIME Ozs. per Ton Per Cent Ozs. Per Ton Per Cent Per C. Per Cent Per Cent Per Cent Per Cent Per Cent Per Cont 14 Trace none 15 Trace 0.3 16 none none C-2 Trace 0.1 Trace none C-4 Trace 0.1 Trace none



651 Colombia Lanc - Provo, Unb 84601 - 375-6960 P.O. Box 361 - American Fork, Unb 84003

eptember 21, 1978

Pacific Exploration 1143 Mariposa San Francisco, CA 94107

SUBJECT: Ure Samples Nos. 2, 3, 4, & 5

Millfeed Delivered by R. B. Beyer

Assay results as follows:

	Fire Assay Ag	. <u>Au,</u>
#2 Millfeed	0.2	0 Trace
#3 Millfeed	Tr.	cc Nil
#4 Millfeed	0.0	5 Trace
#5 Millfeed	0.1	5 Trace

Same samples run fire assay with silver inquart of two (2) grams: (Gold only) 40

	•
#2 Killfeed	2.20 oz./ton
//3 Millfeed	2.10 oz./ton
#4 Millfeed	2.03/oz./ton
#5 Millfeed	2.15 oz./tcn

Four (4) 30-gram samples run with 3-to-1 aqua regis for 24 hours. Filtered and washed. Dilute solution to 1000 ml. to Ph 1.00 to 2.50. Run through selective resin for 30 minutes. Dried resin and burned in assay furnace. Results: (Gold only) Au.

oz./ton
oz./ton
oz./ton
oz./ton

Tails and filtrate of this series were dried and assayed. Results total metal:

			AU.
Tails Filtrate Fee - \$400.00.	o	4.35 oz./ton 2.70 oz./ton	Trace 0.08 oz./ton

Research Chemist

cc: Mr. R. B. Beyer (hand delivered)

Brokers of New and Used Equipment For All Mining Needs

ROGERS RESEARCH & ANALYSIS INC.

HOME OFFICE 551NORTH 1100 EAST BOUNTIFUL U12H 84910 PHONE 18011 295 4402

CLAIR W. ROGERS, President

September 19, 1978

Mr. Rod-Byer Salt Lake City, Utah.

RRA# 82178
Diffraction Analysis of #5 Untreated Ore:

I.	Alpha Quartz or Silicon Dioxide SiO ₂	28,23 \$
II.	Afwillite or Calcium Silicate Hydrate Ca ₃ (Si ₃ OH) ₂ • 2H ₂ O	42.16 %
III.	High Temperature Calcium Silicate Ca ₂ SiO ₄ • .3H ₂ O	10.12 \$
lv.	Hydrate of Calcium Aluminum Silicate Ca ₂ Al ₂ Si ₂ O ₉ · 3H ₂ O	14.98 \$
٧	Calcite or Calcium Carbonate Ca CO3	4.10 %
VI.	Noble Metals: Gold 1.0 - 2.5 O/T Silver 5.0 - 7.0 O/T Platinum Grp= .5 - 1.5 O/T	

Clair W. Rogers M.S.

ROGERSC RESEARCH & ANALYSIS INC.

HOME OFFICE 551NORTH 1100 EAST BOUNTIFUL, UTAH 84010 PHONE (801) 295-4402

CLAIR W. ROGERS, President

. September 19, 1978

Mr. Rod Byer Salt Lake City, Utah.

Diffraction Analysis of #5 Treated: (PRA# 82178)

I.	Alpha Quartz or Silicon Dioxide SiO ₂	92.41 \$
n.	High Temperature Calcium Silicate Ca ₂ SiO ₄	3.13 \$
III.	Hematite or Iron Oxide (Alpha) Fe ₂ O ₃	2.20 \$
IV.	High Temperature Aluminum Silicate Al ₂ SiO ₅	2.06 \$
v.	Noble Metals: Gold 10 12.0/T Silver 5 10.0/T Platinum Grp= 5 5.5 0/T	

Clair W. Rogers M.S.

CHAPTER 3
PRETREATMENT OF ORE

3.1 Historical Gold Recovery Processes (1,2)

3

In the mid-sixteenth century, Agricola comprehensively described man's early methods for recovery of gold from its He first cited handsorting, then went on to describe size reduction by hand, by hammer, by fire, by stamps, and by millstones. He collected the gold with the help of gravity by pan, sluice, table, launder, basket, or tub, with riffles, cloth, or fleece to catch the fines, all aided or not, as the case may be, by amalgamation. Skills in using amalgamation for both gold and silver were developed when gold was first recovered from the quartz veins of California. High losses in the tails were reduced to some extent when chlorination was introduced, but this process with its incovenient reagent has a very limited success. The next great advance in gold metallurgy came with the milling of gold ores in cyanide solution. This was first used in Africa in 1892, though success was not complete, for then there was no method for recovering gold from the slimes. Cyanidation was introduced commercially in New Zealand in 1897, and in South Dakota in 1899.

Starting 1877, gold metallurgy in the Black Hills of South Dakota and at the Homestake Mine followed Agricola's precepts and traditions. After crushing, the ore was pulverized by stamps and the gold recovered by gravity and amalgamation. However, it was not long before the Black Hills became a center for the next great leap in the technology of gold recovery. genius of Charles W. Merrill and inventiveness of J.V.N. Door took the original English cyanide process as a start and improved it tremendously by inventing and perfecting ancillary processes and equipment for separating sand from slimes, for dewatering, aerating, separate cyanide leaching of sands, and pressure leaching of slimes, for removing the treated filter press, and for zinc dust precipitating of the gold from the cyanide solution. The simplicity of design and operation of the new processes and new equipment led to great improvement in recovery, and permitted substantial expansion in the size of equipment. Taken together, these changes sharply reduced the operating costs per ounce of gold recovery. More recently, investigations by H. B. Salisbury and G. M. Potter have greatly increased the understanding of carbon loading and stripping.

The operation of roasting as a preliminary to chlorination expells sulphur, arsenic, antimony, and other volatile substances existing in the ore. Nothing but metallic gold is left behind to combine with the chlorine to form gold chloride when

⁽¹⁾ Henshaw, P.C. Dec 26, 1974

⁽²⁾ Rose, T.K., "The Metallurgy of Gold" 1896, London

treated in an aqueous solution. For this purpose, the ore was heated in a furnace through which a current of air was passed, salt (NaCl) being added if copper, calcium, or magnesium oxides etc. were present. When large quantities of iron pyrites were present in the ore, dust chambers were added to collect the gold vaporized in the oxidizing roast. At one California chlorination mill in 1882 it was found that nearly 50 percent of the gold was being lost by volatilization.

3.2 Recent Technological Developments

It has been amply illustrated in Section 2.1 that many high quality ore bodies have undoubtedly been passed up because of erratic or poor assays values obtained from fire or chemical assays. Colloidal gold films and organically held gold are subject to rapid volatilization upon oxidative heating.

Gold particles encapsulated in alkali metal salts are refractory to most commonly used solvents. Crushing and grinding to the size of colloidal gold particles (10 microns or less) to expose the metal is not economically feasible in commercial practice. Although large quantities of gold are suspected as being present in many types of ores, no reliable, cost effective method has been available to recover it. A promising new method came to light serendipitously in the course of testing a thermal process to recover oil from a Utah shale. A chance testing of the residual sands showed the presence of a quantity of gold. Investigations were started which led to the discovery that the presence of gold in otherwise undetectable form, could be detected in copious quantities by first pre-treating the ore in a carefully controlled thermal roasting process. Instead of volatilizing, the gold agglomerates into larger particle sizes, lending itself more readily to common recovery methods. Commercially proven equipment is available which can be used with only minimum modifications in carrying out this pretreatment of the ore.

Only a relatively crude approximation of the commercial process has been achieved at the laboratory scale thus far, in quantities of 10 pounds of ore or less, by batch processing. Further development is now required which will undoubtedly be a function of the composition of each ore, at least to some extent. This development work must be conducted under precisely controlled environmental conditions to obtain reaction rates, retention times and conversion levels (size enlargement) and optimum operating conditions. The procedures for obtaining this data are discussed in the following three sections.

3.2.1 Tube Furnace Testing

The most convenient laboratory apparatus to investigate the basic aspects of the pretreatment reaction is the tube furnace. This is essentially a cydindrical oven capable of achieving and maintaining temperatures up to 2000° F with

good control. An Inconel tubular reactor with inlet and outlet parts located at opposite ends is vertically suspended in the furnace from a triple-beam balance or other suitable load detector. Gas of appropriate composition is passed through the reactor, in which a weighed quantity of ore has been previously placed. The temperature is gradually increased and the reaction is monitored by weight changes, if such occurs in the particular sample being evaluated.

By selecting heating rates, gas compositions, maximum exposure temperatures, and residence times, basic information of the process can be determined from the resulting weight and analysis measurements. It is estimated that approximately three tube furnace tests will be required to define the essential properties of each ore. Naturally, more resolution would be obtained by further testing, but initial characterization of the ore should require only three tests.

3.2.2 Kiln Similator Testing

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Allis-Chalmers has developed a research and development facility on a scale between bench-type experimentation and a continuous pilot plant operation.

Their Controlled Atmosphere Furnace (CAF) embodies the principles of grate-rotary kiln technology while retaining the ported kiln design aspects.

Figure 3-1 shows the CAF schematically. Figure 3-2 is a photograph of the unit. In the vertical position drying and preheating of the charge takes place, simulating a static bed or traveling grate condition. The pot section is removable and contains a slotted grate in its bottom which supports the charge while allowing the passage of down-drafted hot gases. The furnace pivots about the exhaust trunnion to the horizontal position. The material charge spills into the central section which is rotated to simulate a kiln operation.

Approximately 350 experiments were conducted between 1965-1968 for the purpose of testing various mixtures of gases, process retention times and temperatures.

* A major breakthrough was achieved during the third quarter of 1967. A specific gas was used for the first time as the reactant. This fuel subsequently replaced other mixtures previously used.

At least one test of each type of ore to be evaluated should be conducted in the CAF. Each test requires 200 pounds of ore and takes about a day to complete. Samples are removed periodically throughout the roast for evaluation. A plot of reaction as a function of time indicates the progress of the test as shown in Figure 3-3. After completion of the roast, the treated ore will be shipped to a laboratory set up at BYU to

recover the precious metals through cyanide leaching. The samples will also be sent to an assay lab in a sealed container for X-Ray fluorescence analysis. Laboratory recovery of gold and silver will be compared to results of the X-Ray analysis to obtain the degree of recovery of the process.

The data from the samples taken out during the roast will give additional information on roast temperatures, reaction rates and retention times required. In addition, the data will permit evaluation of the size of the equipment needed for the commercial plant.

3.2.3 Pilot Plant Testing

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s

Prior to constructing a full-size plant, it is highly recommended to identify process flaws on a smaller, less expensive scale. The pilot plant, or process development unit (PDU), is an ideal way to do this. Sometimes it is desireable to construct the pilot plant sufficiently large to break even after operating costs are included.

There are a couple of avenues open to obtain the necessary process design data via a PDU route. One route would utilize equipment already set up and operating, another requires that the plant be constructed on-site. Advantages of the former include: less time to get processing data, no capital cost, experienced personnel and, hence, shorter learning time. Advantages of the latter include: captive facility, longer-term operation, long-term profitability potential, customized facility, etc. The more complex the equipment is, the more alternate one is favored especially if time is a crucial factor.

At the present time, there are two different types of commercially available units to perform the thermal pretreatment of the ore. One method is based on the use of a rotary kiln, such as are available from Allis-Chalmers; the other is based upon the fixed-bed gasifier principle. Foster Wheelers' Stoic gasifier is an example of this type of unit. These units are described in the sections that follow. The final recovery stages of the pilot plant, the cyanidation/electrowinning/refining steps are all expected to be standard methods. Of course, the lab work in the next few weeks will prove this assumption beyond doubt, but it is anticipated that these steps will not be a problem.

1. Rotary Kiln Reactor

At this writing, this approach is favored due to the inherent simplicity of the treatment, including minimal ore preparation. Although the AC rotary kilns used in iron ore reduction and gasification processes are very complex and expensive, we believe that a basic kiln can be fabricated relatively inexpensively.

While it would be highly desireable to use the pilot plant already in operation at AC, we have been informed that scheduling an extended run on this plant would be very difficult in less than a year. The best approach would seem to be to purchase an available second-hand kiln if one can be found of the right size, or to fabricate one, however simple it may be.

The basic concept of the rotary kiln reactor is shown in Figure 3-4.

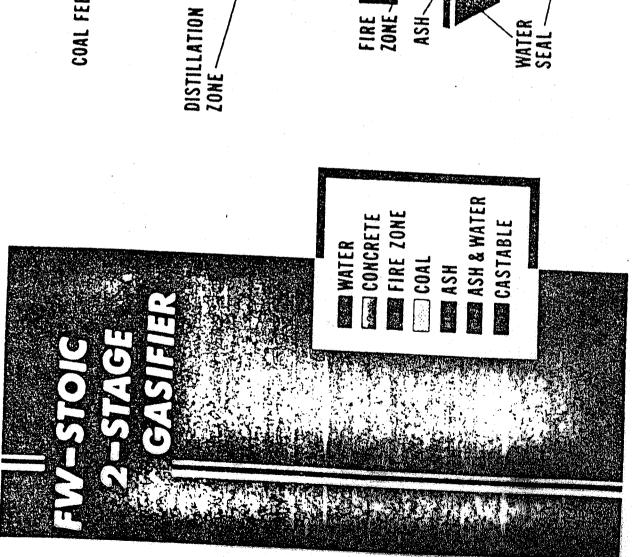
Fixed-Bed Reactor

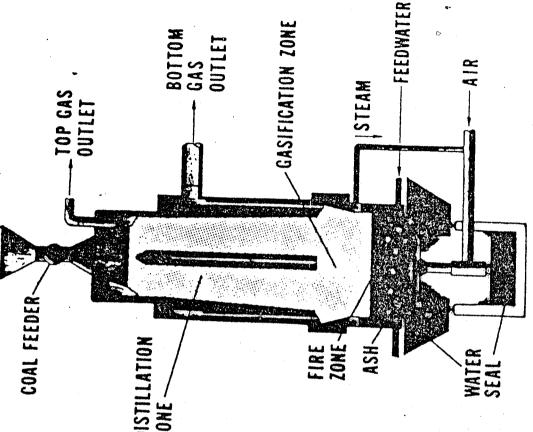
3

Fixed-bed reactors have been around for 20 or 30 years and their technology is well developed. Because of the relatively complicated design of even the simplest of these, it would be preferable to purchase a commercial unit sized for the throughput needed. Pilot-scale units are also available, but it would probably be less expensive to use one already in operation at Foster Wheelers' New Jersey plant. Their unit can be used within a few weeks of notification and their charges would run about \$1000 per ton of ore. The principle disadvantage of this approach is shipment of the ore back and forth across the country. On the other hand, this may be preferable to the delay in time and additional expense in setting up a captive plant that may only be used for a short time.

Another disadvantage of the fixed-bed reactor stems from the fact that the ore and fuel must be carefully sized, or improper gas flow through the bed will result. Channeling or blockage have been principle problems. Figure 3-5 shows the basic Stoic design and Figure 3-6shows a process flow diagram for a typical gassification plant. The ore and coal are top loaded through a rotary As the mass travels down through the bed it is gradually brought up in temperature to the combustion point of the coal. The oils and tars distill off first and are collected in the cyclones and electrostatic precipitator. Gasification occures in the next zone and oxidation occurs in the burning zone. spent coal and ore are ejected out the bottom as a wet ash/ore mud.

In both of the two process methods described, the ratio of fuel to ore will need to be determined empirically and will depend on both the fuel and ore properties.





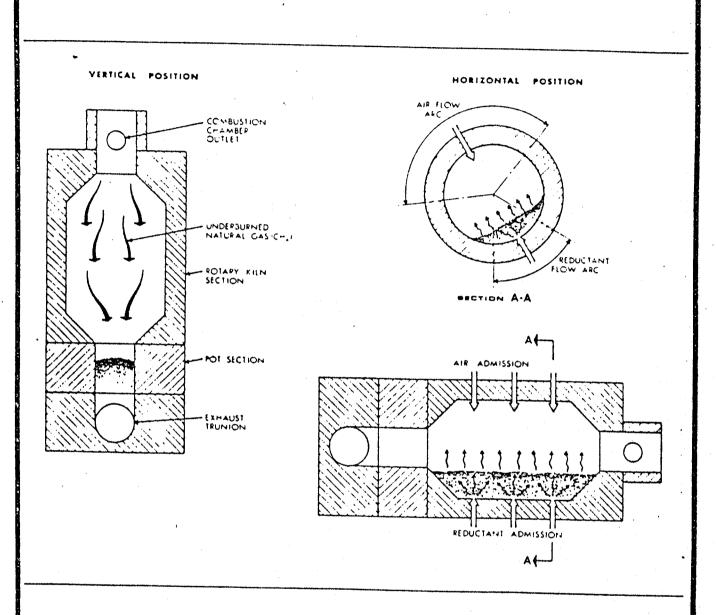


FIGURE 3-1 SCHEMATIC VIEW, CAF (CONTROLLED ATMOSPHERE FURNACE)

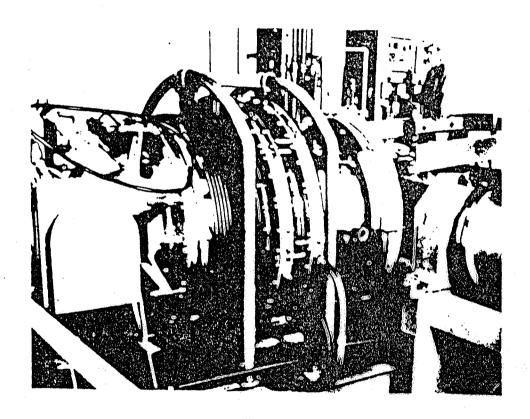


FIGURE 3-2 CAF AT ALLIS-CHALMERS PROCESS AND RESEARCH CENTER

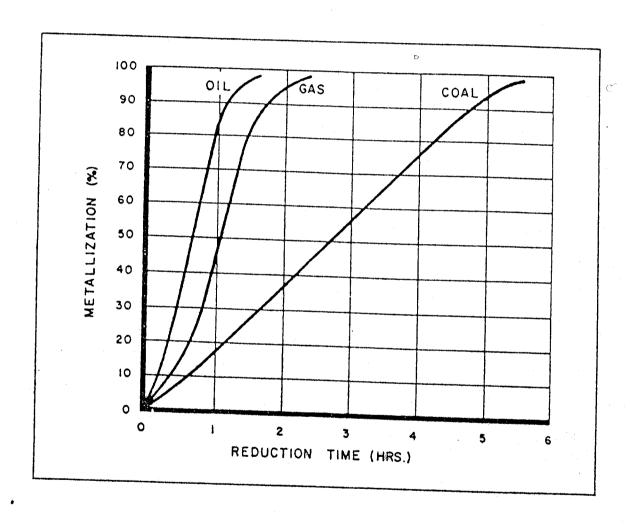


FIGURE 3-3 METALLIZATION & VS. REDUCTION TIME FOR VARIOUS FUELS

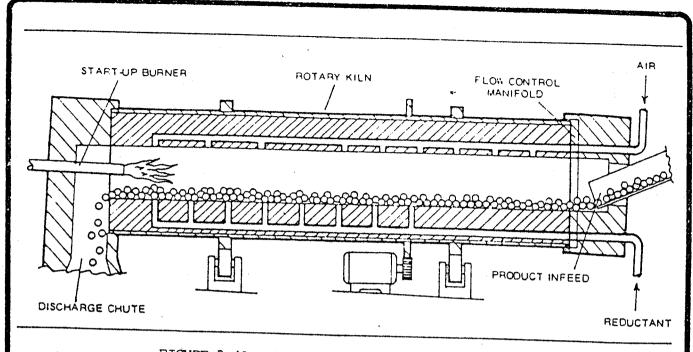


FIGURE 3-4A FARLY CONCEPT OF PORTED ROTARY KILN

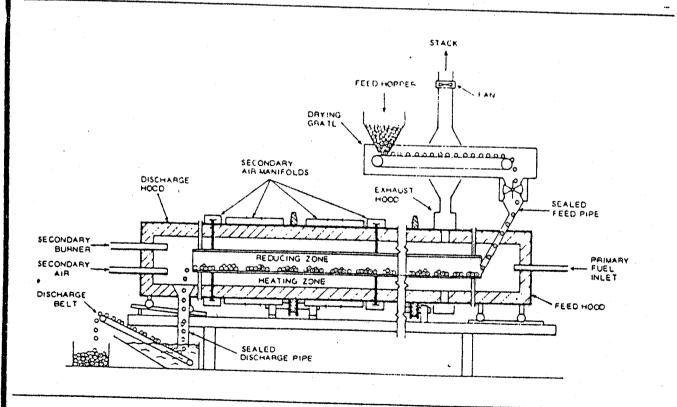


FIGURE 3-4B ACAR (ALLIS-CHALMERS AGGLOMERATION REDUCTION) SYSTEM SCHEMATIC

CHAPTER 4 METALLIC EXTRACTION

4.0 METALLIC EXTRACTION

The second part of the recovery process, after the thermal pretreatment step, is the extraction of the metal from the ore. There are several techniques that could be used, including: cyanidation, acid leach, flotation, jigging, amalgamation, and others.

The heavier or coarser metal particles are often extracted by jigs or shaking tables, where the heavies are concentrated in riffles or launder traps and subsequently removed by amalgama-Flotation is often used to separate the gangue from metallics for smaller particle sizes. Flotation or wetting agents are added to the slurry and frothed in a battery of cells. Depending upon the ore and the reagents used the gold and silver may be in either the overflow or the underflow. pregnant solutions are thickened in settling tanks, filtered, and then refined by smelting. The final stage recovery usually for the very smallest free gold particles or sulfides, is carried out in oxidizing cyanide solutions which complex the gold as the Au(CN)2 ion. Gold is then either precipitated by metallic replacement or electroplated from solution. description of various ore treatments are provided in the A brief following sections by McQuiston and Showmaker. (3)

The properties of gold in ores from the standpoint of recovery are its extremely high specific gravity (15.5 to 19.3 depending upon amount of alloying metal admixed); the fact that mercury wets it readily in the presence of water (amalgamation); its solubility in dilute aqueous solutions of alkaline cyanides to form relatively stable compounds of the form NaAu(CN)2; and its response, particularly as naturally alloyed, to flotation collectors.

1. Native Gold Ores

Free milling lode ores are those in which the gold is relatively coarse and amalgamable, the sulfide content is low and nonarsenical, oxidized compounds of bismuth and antimony are absent, and the gangue is substantially free from talc, clay and graphitic constituents. With these ares, there are advantages in extracting as much free gold as possible in the grinding circuit by gravity concentration. Concentration of free gold by gravity is a relatively simple method of recovery and when used in cyanide plants is applied ahead of cyanidation. On lode gold ores, launder traps, hydraulic traps or pulsating jigs are sometimes used in the grinding circuits for recovery of as much as 60 percent of the total gold in the mill feed. The jig hutch product may be continuously discharged onto a shaking riffled table with the concentrate fed in batches to barrel amalgama-Homestake recovers 20 to 25 percent of the gold in launder traps. Other recovery methods have not been successful because of cable splinters, blasting wire, etc., in the ore. Woolen blankets have long been used for trapping fine gold

particles and particularly for tellurides. Blankets are generally laid overlapping on wide inclined tables. From this practice of using blankets came the development of corduroy to entrap gold and a South African version of corduroy is sheet rubber having "V" shaped riffles molded into its surface. The Johnson concentrator, as inclined rotating cylinder, the plane tables, and belt concentrators are lined with this material.

Amalgamation depends upon the wetting and alloying of metallic gold with mercury. Direct amalgamation in which the entire ore stream flows over mercury-covered copper plates has now been generally abandoned to prevent stream pollution. has been replaced by a concentration step which subjects only a relatively small quantity of high grade concentrate to barrel amalgamation. This method eliminates the tedious cleaning and recoating of the copper plates and reduces the chances for loss through theft. The gravity concentrate is ground for several hours in a small mill or barrel with steel balls or rods before the mercury is added. This form of amalgamation is the simplest and most common method of treating an enriched gold-bearing concentrate. Examples of free gold concentration and amalgamation are shown in the flowsheets of Dome, Homestake, Itogon-Suyoc Palidan, Kalgoorlie, Campbell Red Lake, Blyvooruitzicht, and Vaal Reefs.

Following the recovery of the coarser free fold particles by gravity and barrel amalgamation, the grinding of the ore in cyanide solution with ball or pebble mills is generally practiced. Separate cyanidation of sand and slimes has diminished with the development of closed-circuit fine grinding for "all-slime" treatment by agitation.

Other Free Milling Ores

Gold mineralization in these ores may occur in a limey siltstone containing intermittent shale beds. Sulfides are seldom seen, but pyrite, galena, sphalerite, chalcopyrite, antimony, mercury and arsenic occur in minute amounts. Gold occuring in micron size is readily amenable to cyanidation as at Carlin and Cortez. Other free milling ores are Benguet, Camflo, Kinross, Kloof, and the new Pueblo Viejo.

Gold with Pyrite, and Marcasite

In this ore classification, the gold occurs both in the free state and disseminated in the sulfides. (Pyrite is found to some degree in most of the world's gold deposits). Sulfides tend to decompose in cyanide solutions. Pyrite is the most stable but when pyrrhotite is present trouble is usually experienced both in regard to cyanide consumption and gold extraction. Pyritic flotation concentrates are often reground for gold liberation before cyanidation as at Itogon-Suyoc Itogon, and Pamour. After fine grinding, long periods of agitatfon are often required to dissolve the gold. Goldbearing pyrite

concentrates are sometimes roasted and cyanided in separate circuits as at Kerr Addison. Also, high grade gold-pyrite flotation concentrates can be shipped to the smelter as is Knob Hill practice. Pyrite and pyrrhotite often occur together creating an overlap in treatment methods.

4. Gold with Pyrrhotite

Pyrrhotite readily reacts with cyanide to form cyanates and thiocyanates and it readily consumes oxygen. Aeration with lime ahead of cyanidation is usually used on ores in this classification. Aeration for preconditioning is used at Dome, Homestake, Kerr-Addison, and Pamour.

5. Gold with Arsenopyrite - Arsenic Minerals

Gold is occasionally associated with arsenic minerals as well as pyrite, stibnite, chalcopyrite, etc. Direct cyanidation in these cases is seldom possible. Additionally, when gold is associated with readily soluble arsenic compounds, there is the hazard, in precipitation, of forming arsine, AsH3. In plants where this extremely toxic gas is evolved, special ventilation techniques are required.

Giant Yellowknife produces a refractory flotation concentrate carrying gold in association with arsenopyrite, stibnite, and sulphantimonides of copper, lead and iron. Roasting liberates the sulfide-enclosed gold allowing the calcine to undergo conventional cyanidation. Campbell Red Lake roasts a flotation concentrate ahead of cyanidation.

6. Gold Tellurides

Following the native metal, the tellurides are the most important gold minerals. The tellurides include calaverite and krennerite which contain about 40 percent gold, and sylvanite and hessite with about 25 percent. The Kalgoorlie, Australian, ores contain free gold and tellurides which occur in Pre-Cambrian rock consisting essentially of schists and quartz-dolerite greenstones. Auriferous pyrite is also present and the gold is occasionally associated with chalcopyrite, tetragedrite and arsenopyrite. The gold in the pyrite is finely divided and requires grinding to about 75 percent passing 200 mesh. The ground product is floated and the concentrate, after cyanidation and filtration, is roasted and recyanided. Flotation tailings are also cyanided.

The ores of the Emperor Mine in Fiji contain gold associated with the telluride minerals sylvanite and hessite. A chemical oxidation step is used in place of roasting to liberate the gold for cyanidation.

7. Gold with Copper Minerals

Gold is often associated with chalcopyrite in porphyry deposits. When recovered into the copper concentrate, it travels through the smelter and to the refinery where it reports with the anode slimes from electrolytic refining and is subsequently recovered as gold bullion. Gold losses in copper concentrating are about the same as for copper, but are neglicible in smelting and refining. Gold occuring in pyrite associated with chalcopyrite can sometimes be separated by flotation into an auriferous pyrite concentrate for cyanidation as at Benguet Exploration. At the Itogon Suyoc, Palidan Mill, the auriferous pyrite and chalcopyrite are recovered into a bulk flotation concentrate which is then separated into two flotation products; a pyrite concentrate for cyanidation of the gold and a copper concentrate for shipment to a smelter.

At San Manuel, Arizona, the gold follows the molybdenite and this concentrate is treated by a standard type of cyanidation flow-sheet.

8. Gold with Lead and Zinc Minerals

Gold occurring with lead-zinc sulfide ores or copper-lead-zinc ores usually is recovered into the flotation concentrates and shipped to a smelter where gold recovery is high, particularly at lead smelters. Occasionally, free gold may be recovered by amalgamating the concentrate from a jig in the grinding circuit. Gold contained in the flotation tailing is recovered by cyanidation as any residual galena or sphalerite is not harmfull to cyanidation.

9. Refractory, Carbonaceous, and Graphitic Ores

Carbonaceous is a term loosely applied to those ores containing black graphitic material which causes dissolved gold to adsorb on the carbon thus causing premature precipitation. The gold adsorbed on the carbon is lost with the tailings. Refractory carbonaceous material in gold ores has presented metallurgical problems since cyanidation was adopted in the late 1800's. Some carbonaceous material (unactivated) may not be an adsorbant for gold. Many schemes have been tried such as blanking the carbon with kerosene or fuel oil thereby employs this practice.

Carbonaceous gold ores in the State of Nevada are essentially hydrothermally altered silty dolomitic limestones. The carbonaceous materials are graphitic or activated carbon and long chain organic compounds similar to humic acids. Oxidation by roasting at 500 to 500°C is effective, but costly. Another treatment is chlorine oxidation in pulp as used at Carlin.

Table 4-1 presents an overview of the many treatments used in commercial practice today.

It is premature to try to select the optimum extraction method until more is known about the ore under investigation. For this reason several laboratory-scale test series are underway. These are briefly discussed below.

4.1 Cyanidation

This method is the most likely candidate for the extraction process and will receive greatest initial attention. A laboratory program has been initiated at Brigham Young University in Provo, Utah to investigate this extraction technique.

The approach that is being taken initially is to prepare an oxidizing alkaline cyanide solution. An air lift bath (pachuca tank) is used to thoroughly agitate the slurry during the leaching period. The concentrations of reagents and leach times will be principle parameters to evaluate. Zinc dust and lead acetate will be used to precipitate out the gold from solution. X-Ray analysis of a resultant melt will yield the percentage of the various metallic constituents.

4.2 Acid Leach

If direct cyanidation methods refuse to completely recover the gold, more stringent methods may be required. One of these employs the use of a dilute aqua-region solution which dissolves all gold, silver, and platinum. The reaction is very fast compared to the cyanide leach, but is much more costly.

Silver is first separated by dropping out as the insoluble chloride by addition of salt, and the remaining metallic ions (Au and Pt) are abosrbed on a special resin, which can contain 0.78 lbs metal per pound of resin. The loaded resin is subsequently burned and the metals are recovered as a gold-platinum nugget which can then be conveniently assayed.

4.3 Other

If either of the above approaches are not entirely satisfactory, other methods may be used to accelerate dissolution of the gold. As mentioned earlier; natural gold is occasionally found encapsulated in alkaline or alkaline earth salts. Alcompletely we should pretreatment step should remove this barrier lems. Many of these difficulties are easily overcome by incorporating cyanide or solvent ball milling, or by treatment to increase leach rates. The dissolution reaction rate can normally be increased many fold by several physical or thermal processes. These are not expensive steps and may find application with this ore.

TREATMENT METHOD BY ORE CLASSIFICATION

Ore Classification	Mine Where Used	Treatment Method
Free Milling Lode Ores w/Native Gold	Blyvooruitzicht Homestake ¹ Dome Vaal Reefs	Gravity concentration and amalgamation followed by direct cyanidation.
Other Ores w/Native Gold	Campbell Red Lake Itogon-Suyoc, Palidan Kalgoorlie Knob Hill	Gravity concentration and amalgamation, flotation-cyanidation.
Free Milling Ores	Benguet Camflo Carlin Cortez Kinross Kloof Pueblo Viejo	Direct cyanidationall slime circuit.
Gold w/Tellurides	Emperor	Bulk flotation of Te and Cu. Chemical oxidation of float concentrate, cyanidation; float tails to cyanidation.
	Kalgoorlie ²	Bulk flotation, cyanidation, roasting, re-cyanidation.
Gold in Pyrite	Knob Hill	Bulk flotation concentrate to smelter, cyanidation of tailing.
	Kerr-Addison ²	Cyanidation, flotation, roast- ing of concentrate and re- cyanidation.
	Itogon Suyoc - Itogon	Flotation, concentrate re- ground, cyanidation.
•	Pamour	Flotation, concentrate to cyanidation.

l Carbon-In-Pulp

² Plants using roasting

³ Both Carbon-In-Pulp and Roasting

Ore Classification	Mine Where Used	Treatment Method
Gold w/Pyrrohotite	Dome Homestake Kerr Addison Pamour	Aeration with lime, direct cyanidation.
Gold w/Arsenopyrite	Campbell Red Lake ³	Flotation with roasting of concentrate, separate cyanidation of calcine and tailing.
	Giant Yellowknife 3	Flotation with roasting of concentrate, calcines reground and cyanided with float tailings.
Gold w/Copper complex ore	Magma, San Manuel	Flotation separation of moly concentrate from copper concentrate, cyanidation of moly concentrate.
	Benguet Exploration	Flotation of copper with concentrate to smeltertailings to cyanidation.
	Itogon Suyoc - Palidan	Bulk flotation amalgamation of copper concentrate before smelting; tailings to cyanidation.
Gold in Carbonaceous Ore	Carlin	Chemical oxidation followed by direct cyanidation
	Kerr-Addison	Kerosene used to blank out graphite ahead of cyanidation.

Carbon-In-Pulp
Plants using roasting
Both Carbon-In-Pulp and Roasting 1 2 3

CHAPTER 5 PILOT PLANT DEVELOPMENT

5.0 PILOT PLANT DEVELOPMENT

Before constructing a commercial scale plant it is prudent to first de-bug the process by a pilot plant operation. Normally, this phase of process development should take no longer than a few months of operation - long enough to obtain operating parameters and economic data.

5.1 Basic Design

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Assuming that much of the equipment to be used can be obtained "second-hand", a simple cyanide circuit can be set up in a matter of several weeks. Commercially available refining and electrolytic equipment can be readily obtained at reasonable cost. Certain foundry equipment, now idle, can be secured. The principle effort will go into the design and construction of the thermal reactor, the heart of the process.

The most desirable scenario is to design the unit to process 200 lb/hr (l ton/day). Since operating data are not available, a firm cost estimate is difficult to make. Starting with the assumption that a rotating kiln reactor of 2' by 20' nominal dimensions will give the desired throughput of ore, the other basic equipment required are listed below:

5.1.1 Reactor and Associated Equipment

- 1. Reactor body, 2' x 20' steel or refractory tube
- 2. AP Green castable refractory, 2000 lbs
- 3. Two sets of bearing supports and drive rollers
- 4. Roller drive motor, 15 H.P.
- 5. Plenum housing, 2 ends
- 6. Fans, 4 ea
- 7. Burner system
- 8. Flare
- 9. Feed bin, 1 yd capacity
- Rotary feeder and motor
- 11. Screens, vibrating, 2 ea 2 level
- 12. Belt conveyor, 2 ea, 25 ft 5 H.P.
- 13. Ball mill, 25 H.P., 3' x 4'

5.1.2 Leaching Circuit

- 1. 50 gallon drums, with mixers, 5 ea
- 2. Sweco vibrating screens, 3 ea
- 3. Air lift columns, 4 ea
- 4. Circulation pump, 20 gpm
- 5. Slurry pump, 10 gpm
- 6. Miscellaneous pumps, 10 pgm 3 ea
- 7. Laval separator
- 8. Settling pond
- 9. Cyclone or sonic classifier

5.1.3 Refining System

- 1. Electroplating equipment
- 2. Fitters, 2 ea cartridge type
- 3. Spare cathodes, 2 ea
- 4. Crucible furnace and accessories
- 5. Ingot molds
- 6. Scales

5.1.4 Miscellaneous Support Equipment

- 1. Hoist
- 2. Lab equipment for assays, etc.
- 3. Wiring and Plumbing
- 4. Structural Material
- 5. Wilfley Table, or equivalent

This estimate can be refined as additional test data becomes available. When a flow scheme is fully worked out on the laboratory scale, we will then have some definitive figures.

5.2 Schedule

Assuming that the laboratory phase of work will take three

months to complete to obtain our basic design data, and that much of the standard equipment can be secured or purchased simultaneously with the lab work, very little lost time will be incurred. Two months after the lab work is finished the pilot plant should be operable.

Naturally, this will depend upon how many people can be effectively applied to the operation. The optimum approach will be to start slowly and build up the personnel to a maximum of about eight or nine by the time the plant is completed. It will require the following technical help:

1	Chemical/Process Engineer		5	mo.	
1	Mechanical Engineer		4	mo.	•
1	Welder and Senior Technician	-	3	mo.	
2	Mill men		3	mo.	
1	Foundry man		2	mo.	
1	Electrician		2	mo.	
1	Plumber		2	mo.	
1	Extra	_	3	mo.	
	Total		24	man	months

By starting the pilot plant design early in October, it should be in operation by February 15 (4 1/2 months elapsed time).

5.3 Estimated Cost

Included in the Pilot plant costs are the costs for setting up the laboratory and performing the process evaluation tests required. This is estimated as a three phase effort:

Phase I - Initial lab tests - tube furnace

Phase II - Simulator test and leach circuits designed

Phase III - Pilot Plant Design and Construction

5.3.1 Costs for Phase I

Direct Labor and Labor O.H.
Engineering and Technical assistance
(6 weeks) \$ 7,500

Direct Material and Charges 12 tube furnace tests Assays attendant with above Travel and miscellaneous	5,500 1,200 800
Sub Total-Phase I	\$15,000
Legal Administrative Total-Phase I	5,000 1,500 \$21,500
5.3.2 Cost for Phase II	
Direct labor and labor O.H. Engineering and Technical Assistance (6 weeks)	7,500
Direct Material and Charges 3 Simulator Tests (A/C) Lab equipment and Rental (BYU) Assays attendant with above Travel and miscellaneous Legal	15,000 6,200 3,800 2,500 \$35,000
General and Administrative (Insurance, Accounting, and Office	5,000 2,000
Total Phase II 5.3.3 Costs for Phase III	\$42,000
Direct Labor and Labor O.H. Plant Design, technical assistance (20 weeks)	10,000
Technical labor 24 man-months @ \$12.00 Consulting Chemist Engineering Assistance	49,536 12,000 6,000
Total Labor Direct Materials and Charges	\$77,536
Thermal Reactor and Assoc. equipment Leaching Circuit Refining Systems Miscellaneous Supporting Equipment	\$26,050 8,000 17,750 5,000
Total Mat'l sub total Contingency (11.66%) Sub Total Phase III	56,800 \$134,336 15,664 \$150,000

General and Administrative	5,000
Total-Phase III	\$155,000
5.3.4 Development Fund	
Legal Fund Raising Expenses	\$10,000
Fund Raising Expenses (For Production Facility)	25,000
Contingencies	15,000
Project Total	\$268,500

5.3.5 Operating Costs

The operating costs for the pilot plant are estimated as follows:

		Monthly Cost
l.	Rent - Lab and Office space,	
_	utilities	\$ 300
2.	Office Expenses	
	Business Mgr, Sec'ty	1,200
	Mat'l and Phone	150
3.	Transportation, van for pick-up	
	and delivery	250
4.	Freight (30 tons of ore)	350
5.	Mining cost, labor	350
6.	Plant costs	330
	labor	8,750
	material	2,500
7.	Travel expenses	•
	Per Diem expenses	1,000
		1,200
	Total Operating costs	\$16,000/mo
	(for 2 mo. operation)	\$32,000
	the state of the s	432,000

5.3.6 Recovered Costs

Assuming a recovery rate of 3 oz/ton of Au from the ore* at \$180/oz as a fair market price, and assuming that we can process 30 tons of ore over the first two months of operation, the resulting income will be \$16,200. In the third month of operation and beyond we should process 25 tons/mo and produce an income of \$13,500, which is nearly a break even cost. The plant would be operated as many months as required to obtain the necessary design data for the commercial plant at very nominal or break-even cost.

^{*1/3} of the lowest value assayed by Rogers Research and an average of recovery levels obtained from chemical leach.

CHAPTER 6 COMMERCIAL PLANT

6.0 COMMERCIAL PLANT

6.1 Equipment and Machinery Needed for Mining, Refining and Support Operations - 500 Ton Per Day Plant

Although the process flow sheets cannot be completely determined until the laboratory tests are conducted and the major unit operations are established, a representative schematic, showing typical process equipment and flows is presented for a commercial scale plant. The schematic, shown in two sections as Figures 6-1 and 6-2, represents the pretreatment stage and the chemical extraction stage.

In the pretreatment stage, ore is pumped to the reactor (a rotary kiln) where it is dried and pretreated at temperatures up to 2000° F. As the ore emerges, it is cooled and conveyed to a bucket elevator where it is introduced into the chemical extraction circuit.

In the chemical extraction circuit the ore is screened, if required, and sent to a tank where it is mixed with a cyanide solutions. Slurry pump introduces the ore-cyanide mixture into a classifier stage (sonic classifier) where fine gold is quickly dissolved. Heavier gold is collected from the slimes in a gold trap. The fines and cyanide solution flows to aggitator tanks where residual gold is removed. The fines pass counter current to incoming activated carbon which collects the gold from the solution. The carbon is dissolved by hot cyanide to get a concentrated gold solution from which the gold is electrostripped slimes are washed and discarded. SO2 columns destroy pond for settling and clarifying.

If the cyanide recovery technique proves satisfactory for this ore, this recovery method, or one similar, would be designed.

The first option presented is a plant capable of processing up to 500 tons per day of ore. As a comparison, costs for both 75 and 1000 tons per day plant are estimated by scaling up or down according to the 0.6 power rule common in industrial calculations.

6.1.1 Mining, Pumping, and Pretreatment of Ore

- a. 2 150 H.P. Slurry Pumps
- b. 1 30' diameter holding tank
- c. 1 24' joost drill
- d. assorted pipe and casing

- e. thermal reactor and associated hardware and equipment (see fig. 6-1).
- f. feed bin
- g. D-8 Class Caterpillar Tractor
- b. feeders and conveyors

6.1.2 Desliming and Sonic Classification Stage

This equipment will have to be fabricated to our specifications, and it will include the following:

- a. 500 gallon tank with Lightnin mixer.
- b. Solvent metering tank with associated metering devices and sensors.
- c. 100 gallon per minute slurry pump.
- d. 30 ft sonic classifier columns (3 each).
- e. Cyanide mix tank with metering equipment and sensors.
- f. 25 horsepower air compressor with 100 gallon surge tank.

6.1.3 Final Recovery Stage

Approximately 25 tons per hour of ore (dry basis) will have to be handled in this phase of the operation. Accordingly, the following equipment will be required:

- a. Gold trap, fabricated to our specification
- b. 4 ft diameter by 5 ft ball mill with 50 horsepower motor.
- c. Cyclone separator.
- d. 50 gallon per minute sand pump.
- e. Metering and concentration sensing equipment.

6.1.4 Absorption Stage

- a. 500 gallon agitator tank with Lightin mixer (3 each).
- b. Sweco vibrating screens (3 each).
- c. Air lift columns (4 each).
- d. Sensors and metering devices.

6.1.5 Desorption and Reactivator Stage

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- a. 1,000 gallon batch desorber tank.
- b. 250 gallon reactivator tank for carbon.
- c. 50 gallon hopper for feeding fresh carbon with vibrator screens.
- d. 30 ft conveyor belt.
- e. Wash screens with vibrator and deluge system.

6.1.6 Electrolytic Recovery and Refining

- a. 30 gallon electrolytic cells (10 each)
- 500 amp. regulated power supply.
- c. Control circuitry and sensors for above.
- d. Spare cathodes
- e. 5 kilowatt electric furnace, induction type
- f. No. 70 crucible furnace, gas fired (4 each).
- Ingot molds in various sizes.
- h. 2-ton hoist and trolley system.
- Miscellaneous equipment associated with smelting and refining.
- j. Electric fork-lift (2 each).
- k. Electric sander and grinders (2 each).

6.1.7 Waste and Recycle System

- a. Thickener, washing classifier (fabricated to our specification)
- b. 150 gpm slurry pump.
- c. 50 gallon per minute centrifugal pump.
- d. 20 gallon per minute centrifugal pump (2 each)
- e. Gas bubbler columns, 50 ft (2 each)
- f. Sulfur dioxide supply system
- g. Sensors and controls

6.2 Support System

6.2.1 Maintenance Shop

- a. Pipe threading machinery
- b. 60 inch engine lathe
- c. Vertical mill and attachments
- d. Drill press and assorted power tools
- e. Arc welding, heli-arc and acetylene cutting equipment
- f. Hand tools
- g. Janitorial supplies

6.2.2 Assay and Wet Chemical Laboratory

- a. pH meter
- b. Recording titrator and potentiometer
- c. Analytical balance (Mettler type)
- d. Associated chemistry hardware, glassware and chemicals
- e. X-Ray spectrographic analysis equipment

6.2.3 Office Supplies

- a. Desks
- b. Tables and chairs
- c. Filing cabinents
- d. Typewriters and dictating equipment
- e. Drafting equipment
- f. Other appurtenances and equipment as needed

6.3 Cost Estimates

6.3.1 Engineering and Design

At the outset of the project, a pilot plant unit needs to be set up and operated for between two and six months to obtain design data to optimize the plant operations with the ore. This unit was discussed in Section 5. In addition, several trips to the plant site will need to be made for measurements, and

utility assessment. Power, water, and access roads will be evaluated.

A minimum of six months will be required to complete the engineering and design of the plant regardless of size, but construction in several areas of the operation can begin at once. The engineering and field supervision would be provided by FB&DU with support by subcontract labor. The costs below reflect overhead expenses but no profit. The profit will be negotiated at the time of the contract.

l. Labor Cost

Project Manager	6 mo.	9	6,000	\$36,000
Process Engineer	12 mo.	6	5,000	60,000
Mechanical Engineer	12 mo.	6	5,000	60,000
Electrical/Civil	12 mo.	9	5,000	60,000
Engineer				
Designer/Piper	18 mo.	e	3,000	54,000
Drafting	24 mo.	9	2,000	48,000
Pilot	6 mo.	6	2,000	12,000
Reproduction, etc.	6 mo.	@	1,000	6,000
Total Eng.	Labor			\$336,000

2. Travel and Phone

Air travel	,	6 R.T	. 6	480.00	+141 +	2,800
Auto		7,500	mi.	0.2		1,500
Per Diem	1	10 đa	ys @	50		500
Phone		•			******	1,000
· T	otal Er	ng. &	Desig	n Cost	\$3	41,880

6.3.2 Construction

Construction will begin with consideration being given to utilities and access roads.

Recovery and Refining

Tenatively, a plant large enough for expansion to 2,000

tons per day would seem desirable, finances permitting. A 40 foot high building with insulated steel walls and roof and a cement slab floor of 10,000 square feet should contain all of the operation except large tanks and initial sizing and classifying operation. This would afford maximum protection for refining. It would be sufficiently large for office and laboratory space. At the present time, it would seem desirable to limit the operations to basic refining, although minting may be a desirable option later on.

2. Construction Labor

Civil (Structural)	s	400,000
Equipment and Machinery Piping	•	140,000
Electrical		50,000
Instrumentation		75,000
Insulation		50,000
Painting		25,000
Clean up		25,000
Total Labor		25,000
TOCAL DADOL	\$	790,000

Equipment and Material

Thermal Reactor and Associated Hdw. Desliming Final Recovery Absorption Desorption Electrolytic Recovery, Refining Waste and Recycle System Instrumentation Maintenance Laboratory Office Piping and Plumbing Security Systems \$ 30,000 1,250,000 24,300 24,300 224,300 32,000 32,000 32,000 32,000 32,000 32,000 32,000 32,000 32,000
Desliming Final Recovery Absorption Desorption Electrolytic Recovery, Refining Waste and Recycle System Instrumentation Maintenance Laboratory Office Piping and Plumbing 106,400 24,300 220,000 2200,000 125,000 32,000
Final Recovery Absorption Desorption Electrolytic Recovery, Refining Waste and Recycle System Instrumentation Maintenance Laboratory Office Piping and Plumbing 54,000 106,400 24,300 22,000 200,000 32,000 32,000 32,000 32,000 32,000 32,000 32,000 32,000 325,000
Absorption Desorption Electrolytic Recovery, Refining Waste and Recycle System Instrumentation Maintenance Laboratory Office Piping and Plumbing 106,400 24,300 32,000 32,000 100,000 100,000 50,000 350,000
Desorption Electrolytic Recovery, Refining Waste and Recycle System Instrumentation Maintenance Laboratory Office Piping and Plumbing 24,300 32,000 32,000 100,000 20
Electrolytic Recovery, Refining Waste and Recycle System Instrumentation Maintenance Laboratory Office Piping and Plumbing 32,000 200,000 200,000 25,000 350,000 25,000
Waste and Recycle System Instrumentation Maintenance Laboratory Office Piping and Plumbing 200,000 125,000 125,000 350,000
Instrumentation Maintenance Laboratory Office Piping and Plumbing 125,000 100,000 50,000 350,000 25,000
Maintenance Laboratory Office Piping and Plumbing 100,000 50,000 25,000
Laboratory Office Piping and Plumbing 50,000 25,000
Office 350,000 Piping and Plumbing 25,000
Piping and Plumbing 25,000
Fring and riumbilly
30.000
Water 85,000
Road 30,000
10.000
berdetar (steer, cement, etc.)
or one
• Total Equipment and Materials \(\frac{75,000}{\\$2,726,700}\)

Indirect Field Costs

Project Manager Supervisor Field Engineer Travel Costs Material

			Total		\$ 300,000
5.	Total	Construction	Cost	o o	\$3,816,700

6. Total Engineering and Construction \$4,158,580

6.3.3 Operating Costs

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1. Operating Labor (330 Day Annual Basis*)

Mining Labor 500 tpd @	0.80 \$132,	000
Plant Labor	Number of	Personnel
size reduction, classi	ification	3
extraction		3
refining		3
chemist	•	1
clerical		2
security		8
helpers		6
Total personnel	26 @ \$27,040=\$703,	040

2. Materials

Chemicals Office Fuel - Based on \$20/ton Water	\$142,000 18,000 3,300,000
Electrical	90,000
Total	\$3,550,000

- 3. Indirect at 40% of Labor \$ 281,216
- 4. Total Annual Operating Costs \$4,666,256
- 5. Operating Cost per Ton of Ore Produced (500 tpd) \$28.28

This is equivalent to a break-even recovery level of 0.16 oz./ton of gold at \$180 per troy oz. Shipping costs must be added.

At three (3) oz/ton of recoverable gold in the ore, the annual gross income, based on 330 working days and \$180/oz, would be 89.1 million. After expenses the operation would net over 84.1 million.

6.4 Construction Time Schedule

The engineering and design phase of the commercial plant will take six months but some construction could start almost immediately (roads, water sources, etc.). Plant construction could start by the end of the second month after the pilot plant is operating and results are available to allow us to finish process design. Plant construction may take nine to twelve months, depending upon contract labor availability and material ordering. Normally, eighteen months are required for the complete construction phase. However, we are trying to shorten

the time schedule as much as possible. We cannot be certain yet if we can mobilize an efficient crew that rapidly.

6.5 Cost Estimate - 75 ton per day plant

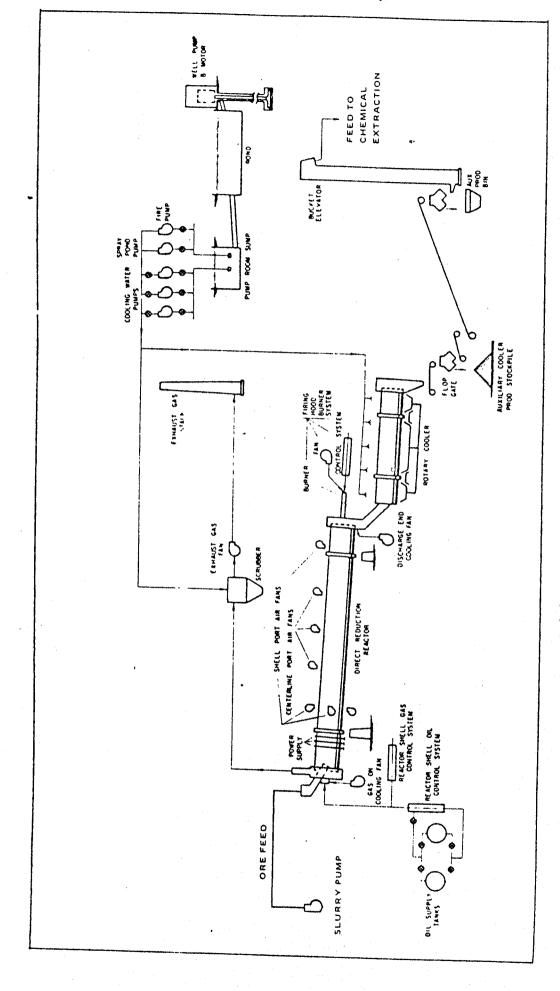
Assuming that the 0.6 factor applies to this scale down, the estimated cost for a 75 ton per day operation would be:

$$\frac{\$4,158,580}{(500/75)^6} = \frac{\$1,332,900}{}$$

6.6 Cost Estimate - 1000 ton per day plant

Assuming that the scale up factor applies in this case also, we obtain an estimated cost for a 1000 tpd plant of:

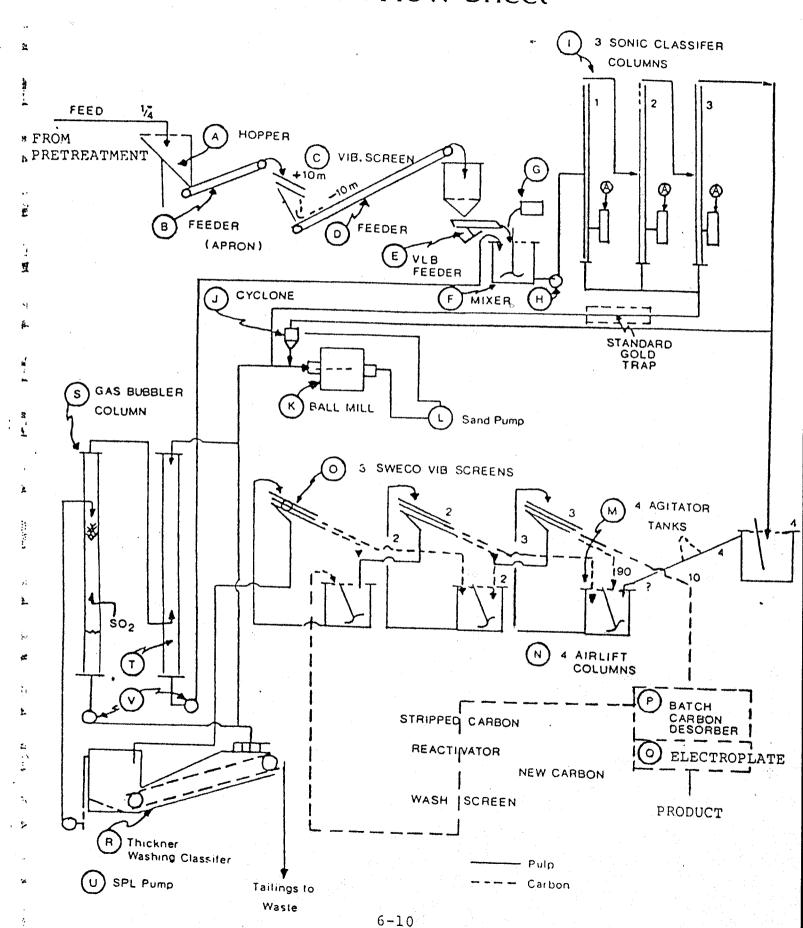
$$\$4,158,580 \quad (1000)^{6} = \$6,303,230$$

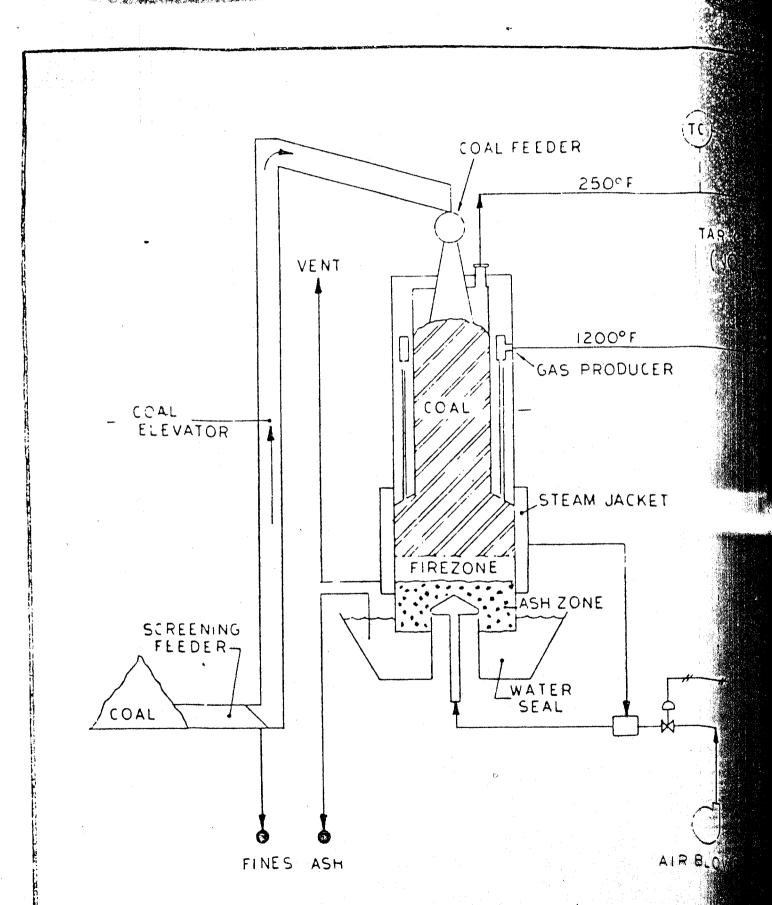


7

FIGURE 6-1 PROCESS FLOW DIAGRAM - PRETREATMENT

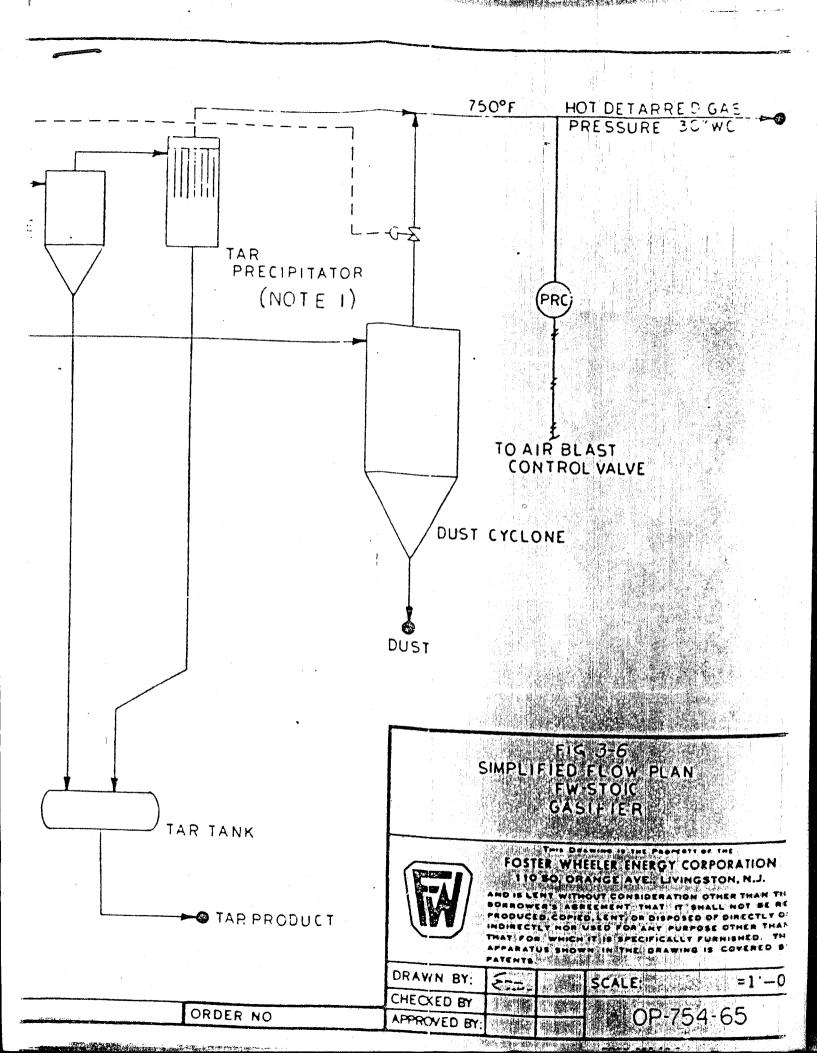
Chemical Extraction Flow Sheet





1. 18

NOTE I - FOR HUT PAN GAS PRODUCTION, ONLY TAR CYCLO



APPENDIX

Mr. Rod Byer C/O Ford Bacon & Davis Salt Lake City, Utah.

8 TO

SALT LATE CITY, UTAN 24121
Phones (201) 121-12244

DATE September 7, 1978

SUBJECT: Quar	restative Analy	rses RRA #82178-1-1	.7 Pt.
rous de Cust	omers Identifi	cation:	Platimum
Sunli	ite #5 Supertre	sated R fl	.108 o/T
•	#5 Treated	R #1	.058 o/T
•	# 4 Treated	R #5	.112 o/T
	#1 Treated	R #4	.023 O/T
Jadite	13 Treated	R #3	.026 o/T
*	#2A Treated	R #2	.109 ₀ 0/T

SIGNED

LOT # 87704

AVAILABLE FROM BUSINESS ENVELOPE MANUFACTURERS, INC. . PEARL RIVER, N.Y. . BRONX, N.Y. . CLINTON, TENN. . MELROSE PARK, ILL. . ANAHEIM, CALIF. PRINTED IN U.S. A.

Mr. Rod Eyer & S. Hunsaker 1924 Fine Lane Provo, Utah 34601

117 1712 CAV, CTV 11 24.07 (144.25 (271) 521 22 11

¥	(Conti	inusd)		
e ·	il	Quantitative Analyses PRA	#82178_1_17	Saptamber 7. 197
LOFE	~	Customers Identification:	Gold	Silver
; ;	7.	R #1 Supertreated	7.18 G/T	11.32 c/T
¥	8.	R #1 Untreated	2.02 o/T	5.06 n/T
•	9•	R #1 Treated	5.12 O/T	8.12 n/T
· ,	10.	R #5 Untreated	2.13 O/T	5.13 c/T
ý	11.	R #5 Treated	9.52 c/T	12.12 n/T
<i>,</i>	12.	R #4 Untreated	.811 c/T -	4.23 n/T
1	13.	R #4 Treated	2.89 n/T	8.11 0/T
i	14. F	R #3 Untreated	1.04 c/T	3.89 c/T -
2	15.	R#3 Treated	2.59 0/T	6.99 O/T
į	16.	R #2 Untreated	2.04 n/T	4.89 C/T
J. Water.	17.	R #2 Treated	8.11 o/T	9.81 n/T

AVAILABLE FROM BUSINESS ENVELOPE MANUFACTURERS, INC. . PEARL RIVER, N.Y. . BRONX, N.Y. . CLINTON, TENN. . MELROSE PARK, ILL. . ANAHEIM, CALIF.

Mr. Rod Byer C/O ford Bacon & Pavis Salt Lake City, Utah.

Diana:	(101)	521-22:	:	
				
		C		 •

ا د کونا	Sul Com	Quantitative Analyses RRA#9978-1	-13		September 11, 197
FOLI	*	Customers Identification;	Gold	Silver	Platinum
; ;	1.	C#2 1-Ar.50 min	7.62 c/T	6.85 n/T	.097 n/T
7	2.	C#3 1 Rr. 50 Min	8.95 n/T	7.65 c/T	•105 n/T
7	3.	C34 1 Er. 50 Min	12.13 n/T	5.42 n/T	.125 n/T
ř	4.	C35 1 Hr. 50 Min	6.15 n/T	4.12 O/T	.095 n/T
•	5.	C#6 1 Hr. 45 Min	4.25 C/T	3.25 c/T	.084 C/T
<u>ځ.</u> خ	6.	C#9 1 Hr. 40 Min	5.12 c/T	6.14 o/T	.065 n/T
₹.	7.	C#10 1 Rr 35 Min	1.28 n/T	5.11 o/T	.021 n/T
3	8.	C#11 1 Rr 30 Min	2.18 O/T	4.03 C/T	.030 n/T
3	9.	C312 1 Hr. 20 Min	1.66 c/≸	9.42 O/T	.022 n/T
	10.	C#13 1 Hr. 20 Min	1.81 n/T	3.31 o/T	.024 O/T
2.	u.	C\$14 2 Hr. 15 Min Lo Heat 300_400F	2.03 n/r	2.11 o/T	.025 n/T
•.	12	C315 2 Hr. 15 Min W H	.855 0/T	1.89 n/T	.010 n/T
ţ	13.	Cf16 1 Hr. 15 Min " "	.715 n/T	.912 n/T	.009 n/T

LOT # 877084

AVAILABLE FROM BUSINESS ENVELOPE MANUFACTURERS, INC. + PEARL RIVER, N.Y. + BRONX, N.Y. + CLINTON, TENN. + MELROSE PARK, ILL. + ANAHEIM, CALIF.

Ford, Bacon & Davis Atah Inc.

ENGINEERS - CONSTRUCTORS

A SUBSIDIARY OF

Sord, inacon a Davis

375 Chipeta Way P. C. Box 8009 Solt take City Utah 84129 861-583-3773

Sept. 11, 1978

To. Jack Jutzey From: R. B. Beyer

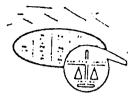
Subject: Assay Results- Samples from Sunlight Mine

Enclosed is the assay report from Rogers Research for the last camples I received from you.

The pretreatment on the ore was carried out for various lengths of time to help ascertain the optimum residence time. The results suggest that there is a pronounced reduction on conversion for exposure times less than 1 hr 40 min. The results of the "low temperature" samples and samples treated for less than 1 hr 40 min appear to be close to the "untreated" samples.

Trustment is effective for all samples assayed to date when the temperature is above 500° C and the residence time exceeds 2 hrs. Higher assays result as the temperature is increased to a maximum of 2100° (ca 1200° ,C).

Chemical Incheser



SKYLINE LABS, INC.

Hawley & Hawley, Assurers and Chiemists Division P.O. Box 50106 • 1700 West Grant Road Tucson, Arizona 85703

(602) 622-4836

REPORT OF SPECTROGRAPHIC ANALYSIS

C/ry fire Miling Hills H & H No. 742429

[117] Total Company Hills H & H No. 742429

| Company Company H & H No. 742429

Koradamex, Inc.
4014 Contral S. E.
Albuquerque, New Mexico 87108

Attention: Mr. Jim Smid

Values reported in parts per million, except where noted otherwise, to the nearest number in the series 1, 1.5, 2, 3, 5, 7, etc.

	Sample Number
Element	X 1542
Fo	51
Ca	78
Mg	20
λg	(i
λ¢	<500
В	مود
Ba	700
Be	7 4 4 4 2
B:	<10
Cd	<50
Co	10
Cr	50
Cu	30
Ga	15
Ga	<20
La	20
Nn.	500
110	10
ИЪ	<20
ит	20
Pb	10
Sb	<100
Sc	10
Sn	<10
Sr	1,000
Ti	
v	1,500 150
W	₹50
Y	20
Zn ·	<200
Zr	100

William L. Lehmbeck Manager

5800 RENO HIGHWAY FALLON, NEVADA 89406 Telephone 867-3678

Fallon, Nevada

RESULTS PER TON OF 2000 POUNDS

. HUMBER	COLD the per Ton	SILVER Dia per Ton	LEAD Wel on Ore	COPPER Per Cent	TUNGSIE: Per Cent	ZINC Per Cent	ANTIMON' Per Cent	IRON Per Cent	Cura Per Cent	Per Cent	Га
76550				•	·			•			
jons.	.069	3.10	std.	òre as	зау						
ons. C&C	.565	7.96	corre	ted an	contr	pled					
wails cyn. Fire wails via cyn.		.42 0	after 4 # p	cyanid r ton	e no	lime a	d e d				
Mill Tes	7						·				
1- FLOTATION	<u> </u>	en Tra	TAS S	NTro	ed FIX	e Ass Corre	cled				
3 - Final T 1 - Yalue re	21/5 COYO,	ed by	Trea	TING	FloTzT	Yor 7	71/5				

Rosails		·		
Charges 8		Ulation	oping.	
1975	Hand S aple Serial	112896	Newsca / co	

ina J & J Peterson

Fallon, Nevada

RESULTS PER TON OF 2000 POLINDS

FALLON, NEV. 2012 Yelophone 807-3078

RUDISER	COLD Pas. per Ton	Ous, per Yor	LEAD Wet on Ore	COPPER Per Cent	TUNGSTE:	ZINC Per Cent	A.7313503. 7.76376611		Carra ii Por Canti Per Centi
* 3653 * 3653 *	11	.723	stand: % HC r Ag		ивзау	- 1	Total gold	tota silve	1
ຳວ່ວວັງ3 Cyn. ຊື່	2.622	0						/ 5. / 2.	
u									

Remarks (III)

Mine	Cold Range Inc.	
	••	١.

5800 RENO HIGHWAY FALLON, NEVADA 89406 Telephone 867-3678

RESULTS PER TON OF 2000 POUNDS	February 1, 1974	

NUMBER	COLD	SILVER DIA PET TOT	LEAD Wel on Ore	COPPER Per Cent	TUNGSTES Per Cent	ZINC Per Cent	Yer Cent	IRON Per Cant	Per Cent	Per Cent
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	44	rEG.50	ua6l	e_	"hot	spot	1- h	owe	ver	
	13 9	11	41	M	11	15	ut 1	11	13	
	11+	mig	nt h	pt	DC.	50	far	off		EA
·										

Modified eyaride mill test	` _		
Remarks		212	
		1/	4)
Charges \$		44	

Mine _____ Gold Basin Mining Co.

Nevada Assay O

5800 RENO HIGHWAY FALLON, NEVADA 8940

Telephone 867-3678

_ NUMPER	COLD pus. per Ton	SILVER Das per Ton	LEAD Welon Ore	COPPER Per Cent	TUNGSTES Fer Cent	D ZINC Per Cent	ANTIHUNE Per Cent	IRON Per Cept	CaF2 Por Cent	PNF Ce
6867 *6869	.534 .462	1.05			·					
3										
v		-								

Mine		Hand Sump d Bosin		C			~	5800	RENO :	HIGHWAY	. *
manufact statement		lon, Kev PER TON (DUNDS						/ADA 8940 867-3678	o
NUMBER	GOLD	SILVER Dzs. per Tor	LEAD	COPPER	TUNGSTI Per Cen	ZINC	ANTIMON Per Cen	IRON Per Cen	CaF2 Por Ce	ण रिक्स्प्रे द्रन्	() = 0
. NORDON	l									n oz.	eŗ
_	Невів		为HC	74HC	⅓SC)%SC				560	
6550 ·	.760	}}	1.30	1.04	iii					.768	
6551	.18	1.26	.86	.90	.16	.56	Sec 3	16	427.	.405	
			,			٥					
			9	II						N	
Resnarks					7//	1/	/	07) /		
Charges \$			·	, , , , , , , , , , , , , , , , , , , ,	ill.	Celi.	erry J.	34	ley		
oct. 7, 1974	н	and Sample					Ne	-vada	 A cc:	ay Off	íce
1ine		Gold Ba:	sin Min	ing	· 4 NP (New or assuring my, 6-64)			5800 1	RENO H	IGHWAY	100
Fallon, Nevada							FALLON, NEVADA 89406 Telephone 867-3678				
		ER TON OF									
NUMBER	GOLD see, per Ton	SILVER Ora per Ton W	LEAD (OPPER er Cent	Per Cent	ZINC Per Cent	ANTIMON Per Cent	IRON Per Cont	CaF2 Per Cent	Per Cent	Per C
6071	29.89	2.90				. [
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