Mining District File Summary Sheet

DISTRICT	Tonopah
DIST_NO	4840 60000509
COUNTY If different from written on document	Nye
TITLE If not obvious	Hughs Tool Co.; Metallurgy; reports, data and correspondence on plant at Tonopah
AUTHOR	D.J Gobbin; W. Simmons; D.L. Pruett; P. Skinner; R.E Baker; W.D Mallison; P. Robertson; C.S. Ikehara; P.G. Reeve; R.S. Shoemaker; D.R. Paxton; D.K. Hamilton; SEE BACK
DATE OF DOC(S)	1974-1976
MULTI_DIST Y / (N?) Additional Dist_Nos:	
QUAD_NAME	Tonopah 7½'
P_M_C_NAME (mine, claim & company names)	Hughs Tool Co.; Summa Corp.; Universal Silver; URS/Hill, Ingman, Chase and Co, Bechtel Corp.; Husky Industries; Idaho Mining; Sitkin Smelting and Refining Inc.
COMMODITY If not obvious	gold; silver; copper; lead; zinc
NOTES	correspondence; reports; assays; test data; diagrams; equipment specifications; handwritten notes; chemical lists; receipts NOTE: Scan front of folders at dividers

Keep docs at about 250 pages if no oversized maps attached
(for every 1 oversized page (>11x17) with text reduce
the amount of pages by ~25)

Revised: 1/22/08

	Initials	Date
SS:	DD	2/25/08
DB:		
SCANNED:	T.M.	3/23/09

AUTHORS CON'T: D.C Norcross; L.W. Taggart; B. Walker; J.L. Wood; D.C. Mathews; F. Fillerup; P Ricks; H.J. Heinen

Met.
General - Pilot Plant 129

HUGHES TOOL CO. 129
METALLURGY
METALLURGY - PILOT PLANT

60000509
4840

DATE: February 5, 1975

TO: D. J. Gribbin and Walt Simmons

FROM: Dave Pruett

SUBJECT: Pilot Plant Functions

1. Use as a Sampling Plant: Sampling can be done much more accurately when done in bulk. The larger the sample, the better statistical base. The finer the sampled material is crushed, the more accurate the results of sampling. No sampling is completely unbiased but the mechanical type, if set upright can correct and average most error. The fine bin discharge from this pilot plant may easily be routed outside to a sampler if desired. Capacity: 3 tons per hour minus ¼".

The auto samplers provided will give good head and tail information of the ground and processed pulps.

2. Crushing and Grinding Evaluations: Observations of time, power requirements, size distribution, particle shapes may be correlated directly to requirements of full size machinery.

3. Carbon in Pulp-Cyanide Plant -- Gold and Silver Ores: This pilot plant is a direct copy of the most current gold-milling trend in gold extraction. The newest American plant using this technique is the Homestake Mill, Lead, South Dakota. Three new major plants in South Africa built within the last four years have been carbon in pulp. Ores not generally adaptable to heap leaching or of value too high to risk the losses inherent in heap leaching are often treated economically with carbon in pulp.

Advantages compared to conventional cyanide:

(1) Minimum amount of equipment.
(2) Ores with difficult settling or filtration character may be treated.
(3) Free cyanide need not be maintained in final

stages, a major problem in environmental waste.
(4) Fouled solutions do not retard precipitation.
(5) Low demands on operational personnel, technically and in time required.

This process is not experimental but a well proven method.

4. Flotation: Ores other than gold and silver may be run. Silver ores often treat better in flotation than cyanide. Methods are well established and operational costs low. Also, fine carbon can be treated by flotation and recovered. Cheaper carbons may be tested in conjunction with the cyanide plant. Flotation of carbon plant tails check carbon loss and abrasion property.

5. In-House Test Work: Although outside consultants may be necessary often, the advantages of test work by existing personnel are readily apparent. If a full-scale operation is envisioned many of the bugs can be eliminated by pilot operation. Experiments altering the flow of existing plant machinery can be run first in pilot.

6. Adaptable Ores: The carbon in pulp plant represents the probable flow sheet and treatment of Tonopah ore and the Tonopah Belmont Tailings. Checks can be run on Manhattan ore and others. The flotation plant could process the type of ore from the Lida area and the properties in Mineral County or any flotable ore. By acid proofing the agitators and pumps copper ore could be leach tested. Any bulk samples could be crushed and split.

7. Rates of Treatment:

Sampling	3 tons per hour	(-¼ inch)
Carbon in pulp	15 tons per day	(-35 mesh)
Flotation	10 tons per day	(-65 mesh)

(Higher for softer ores and coarser grinds.)

8. Labor Requirements:

1 crusher man	1 shift/24 hours
1 mill operator	1 per shift

Dave Pruett

SUMMA CORPORATION: TONOPAH, NEVADA

PILOT PLANT COSTS: FLOTATION, CARBON IN PULP

	Material	Labor
Feed Hopper 5' x 9'	$ 600.	$ 200.
Elevator Repair	100.	100.
2' x 4' Screen	600.	250.
6" x 16" rolls	1,500.	250.
Fine Ore Bin	750.	300
Feeder Installation		150.
Screw Classifier	1,500.	200.
Pump Installation with catch	125.	100.
Reagent Feeders:		
2 Clarkson Wet	500.	30.
1 Dry	150.	80.
Auto Sampler	350.	150.
Conditioners Tanks (3)	1,800.	300.
Conditioners Motors	300.	100.
Carbon Screens (3)	1,500.	250.
Carbon and Pulp Air Lifts	250.	200.
Compressed Air -- 30 cfm.	1,750.	200.
Carbon Feeder	200.	50.
Auto Sampler	350.	150.
Tail Pump Installation	150.	100.
Pan Filter	300.	250.
Pan Filter Vacuum Blower	400.	150.
Flotation Machines	1,750.	200.
Supervision, Engineering, 40 days to complete		6,000.
	$14,925.	$9,860.
Set Amalgamation Unit	250.	250.
Set furnace	400.	250.
Carbon Bin	400.	250.
Drum Deck Wood	600.	300.
	$16,575.	$11,010.
Estimate plus Contingency	$27,575. + 2,425.	
TOTAL ESTIMATE		$30,000.

URS/HILL, INGMAN, CHASE & CO.
Consulting Engineers and Analysts . . . since 1891
2909 Third Avenue, Seattle, Washington 98121
(206) 623-6000

URS
A URS COMPANY

February 20, 1975

Mr. D. J. Gribbin
General Manager
SUMMA Corporation
Land Exploration and Mining Division
5700-B South Haven
Las Vegas, Nevada 89119

Re: Methanol Recovery

Dear Mr. Gribbin:

The attached is a more detailed material cost summary of the methanol recovery system proposed in our interim report. This will give a better comparison of the packaged system that Paul Skinner is pricing.

We are ready to complete the project when the choice of a system is made.

Very truly yours,

Cal Patterson

Calvin Patterson, P.E., Ph.D.
Director, Environmental Engineering Design

CP/mk

cc: Paul Skinner
SUMMA Corporation
Box 1126
Tonapah, Nevada 89049

RECEIVED
SUMMA CORPORATION
MINING DIVISION

FEB 24 1975

SYSTEM C
MATERIALS COSTS
(NOT INCLUDING INSTALLATION)

SUMMA CORP.
TONAPAH, NEVADA
2/18/75

Vaporizer Body (Fabricated as shown on Sketch #1, 2/13/75 Report)	
Metal	$ 1,140.00
Insulation (120 SF 2" Th Foamglass)	70.00
Vaporizer Body Total Cost	$ 1,210.00
Vaporizer Piping (Fabricated as shown on Sketch #2, 2/13/75 Report)	$ 740.00
Vaporizer Pump (Bell & Gossett 1-1/4" AA)	155.00
Vaporizer Heaters (Chromalox NWH 64015)	$ 2,430.00
Vaporizer Piping & Heating Total Cost	$ 3,325.00
Condenser (Bell & Gossett SU-85-4)	$ 510.00
Condenser Support	500.00
Cooling Tower (Marley Aquatower 4617)	725.00
Cooling Water Pump (Bell & Gossett 1-1/4"AA)	155.00
Condenser & Cooling Tower Piping	310.00
Condenser & Cooling Tower Total	$ 2,200.00
TOTAL COST	$ 6,735.00

walt

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Consulting Engineers and Analysts . . . since 1891
2909 Third Avenue, Seattle, Washington 98121
(206) 623-6000

A URS Systems Affiliate

February 3, 1975

Mr. D. L. Gribbin
General Manager
SUMMA Corporation
Land Exploration and Mining Division
5700-B South Haven
Las Vegas, Nevada 89119

Re: Methanol Recovery System

Dear Mr. Gribbin:

This letter will outline the scope of the engineering services we propose to provide in accordance with our discussion of the system with you and Paul Skinner on January 31, 1975. We will select the best of four alternate feasible systems for boiling and condensing methanol on the basis of capital and operating costs, space requirements, and time required to get the system on line. After we have agreed on the system we will prepare layout, piping, and wiring schematics for the system installation and specifications for purchase of equipment and material.

The following is a list of systems that will be studied and steps that will occur in the course of the project.

1. Determine size, material, power input, cost, delivery time, reliability, etc., for

   a) Package solvent distillation system

   b) Tailor made combination of electric boiler, vapor system and a water-cooled shell and tube condenser.

   c) Same as b) with an air-cooled condenser

   d) Heat pump for refrigerated condenser and heat recovery in the boiler system

2. Summarize results of Step 1 and present to SUMMA Corporation, select system.

3. Prepare layout, piping, and electrical schematics and specifications for the selected system.

This work will be performed by Messrs. Clyde Crawford and Norman Walton, resumes enclosed, under my supervision. The work will be done on a time and materials basis.

Individual billing rates are:

Mr. Crawford	$24.00/hr.
Mr. Walton	$30.00/hr.
Mr. Patterson	$41.50/hr.

We have started work on the project and we anticipate completion of Steps 1 and 2 in two weeks, and Step 3 in one additional week.

We will appreciate you signature below on one copy of this agreement and its return to us. I wish to express my personal appreciation of your hospitality on the 31st and of the opportunity to be of service.

Very truly yours,

Calvin Patterson

Calvin Patterson, P.E., Ph.D.
Director, Environmental Engineering Design

CP/cw

Encl.

Chester J. Woods — 2-5-75

For URS/Hill, Ingman, Chase
Chester J. Woods — Date

D. L. Gribbin — 2-5-75

For SUMMA Corporation
D. L. Gribbin — Date

cc: Elzie Arnold, URS Nevada

PILOT CYANIDATION DATA SHEET

NOMINAL CAPACITY: one ton of ore per twenty-four hours.

CAPACITY LIMITATION: thickener throughputs (10 ft^2/ton/24 hrs).

EQUIPMENT COSTS:(1)

Process Equipment	$20,000
Process Instruments	2,800
Laboratory Equipment	4,500
Shop Equipment	1,500
	$28,800

EQUIPMENT SOURCES: 4% used, 39% fabricated in-house, 11% fabricated outside, 46% purchased ready-made.

BUILDING REQUIREMENT:

SURFACE: Process: 1,000 ft^2
Services: 500 "
1,500 "

HEADROOM: 13'6" general requirement; ability to go to 17'6" in one spot.

LOCATION: anywhere in the world.

GENERAL DESCRIPTION: This facility employs countercurrent decantation to evaluate the processability of gold and silver ores. The grinding area is designed to operate six hours per day and uses air classification; the rest of the plant is to operate twenty-four hours per day. The plant can be shut down on weekends.

PROCESS CONTROL: The plant is designed to operate with minimal operator attention. No part of the process requires constant attention.

OPERATING PERSONNEL:

Wageroll: three shift operators, one mechanic/laboratorian
Salary: one chemist, one engineer

---

(1) Bare equipment; ex installation and engineering (also excluding piping and electrical work).

SERVICE REQUIREMENTS:

Water:	almost none (depends on ore character)
Air:	low (air compressor is included)
Nat. Gas:	low (for assay and laboratory)
Inert Gas:	low (one N_2 cylinder every 4 days)
Steam:	none
Electricity:	low (plan for 15hp, 110VAC installed)
Polution Treatment:	infrequent (chlorination of effluent)

ERECTION:

Time Under Favorable Conditions: 45 days
Time Under Unfavorable Conditions: 90 days

Personnel: one welder (temporary), one mechanic (permanent), one design engineer (temporary)

mansfield process engineering

– plant design, economic analysis, trouble shooting –

305 Edison Way
Reno, Nevada
89502

(702) 322-6784

October 18, 1973

[redacted], Inc.
[redacted]
[redacted], Nevada 89701

Attn: [redacted]

Dear Mr. [redacted]

Is cyanidation feasible? This is a question that usually is never answered satisfactorily. Most attempts to answer the question are laboratory tests done in beakers. These tell whether cyanidation is possible, not whether it is feasible. A study of the ore in an actual cyanidation process is what needs to be done: watch coagulation in the thickeners, observe the effect of CN concentration, record how often barren solution must be dumped.

Building a full-scale process based on beaker tests is a mistake. The way to evaluate cyanidation is to run the ore on a pilot plant for several months.

Attached are drawings and data sheets for an inexpensive pilot cyanidation plant. Nominal capacity is one ton per day. Equipment cost is about $30,000. Shown here is only a small portion of the information developed at this time. The Preliminary Design (equipment sizing, etc.) is complete. What remains to be completed is the Final Design (tank drawings, piping drawings, etc.). In most cases, specific pieces of equipment have been chosen and are ready to order.

A considerable amount of effort has been spent arriving at a design that is inexpensive, yet semi-automated. For example, complex sealing devices have been eliminated in the zinc precipitation tank (P-200) by using a nitrogen blanket on the contents. Also high quality components have in many cases been substituted for by mediocre quality components: this plant doesn't have to last for ten years.

Perhaps the pilot plant is of interest to you. I will be happy to discuss this with you further. In case you are wondering what my stake in this might be, I suggest you examine the sheet entitled "Pilot Cyanidation Plant Design Cost."

Very truly yours,

Scott Mansfield

Scott Mansfield

SM/c

Randy

mansfield process engineering

– plant design, economic analysis, trouble shooting –

305 Edison Way
Reno, Nevada
89502

(702) 322-6784

February 19, 1974

SUMMA CORPORATION
Mining Division
5700 Haven
Las Vegas, Nevada

SUMMA
MINING DIVISION
FEB 22 1974
RECEIVED

Attn: D.J. Gribbin

Dear Mr. Gribbin:

I recently spoke to an ex-consultant of yours called Dan Kappes. I told him about the design I have prepared of a pilot cyanidation facility. He said you would probably be interested in learning about it.

As a consequence, I am sending you (1) the specification sheet on this pilot plant and (2) a folder that generally describes my business.

If you are interested in more details regarding the cyanidation facility, I am prepared to send you blueprints and samples of the equipment list pertaining to the Preliminary Design.

Even though this plant is described as being plain countercurrent decantation, the techniques of electro-oxidation or adsorption on charcoal can be incorporated if so desired.

Regarding the folder and plant design in general, I think it is sufficient to say that welders and mechanics work best when guided by blueprints. Also equipment sized by firm calculations will fit any scheme much better (and usually at less cost) than equipment sized in a helter-skelter fashion with the accompanying huge safety factors.

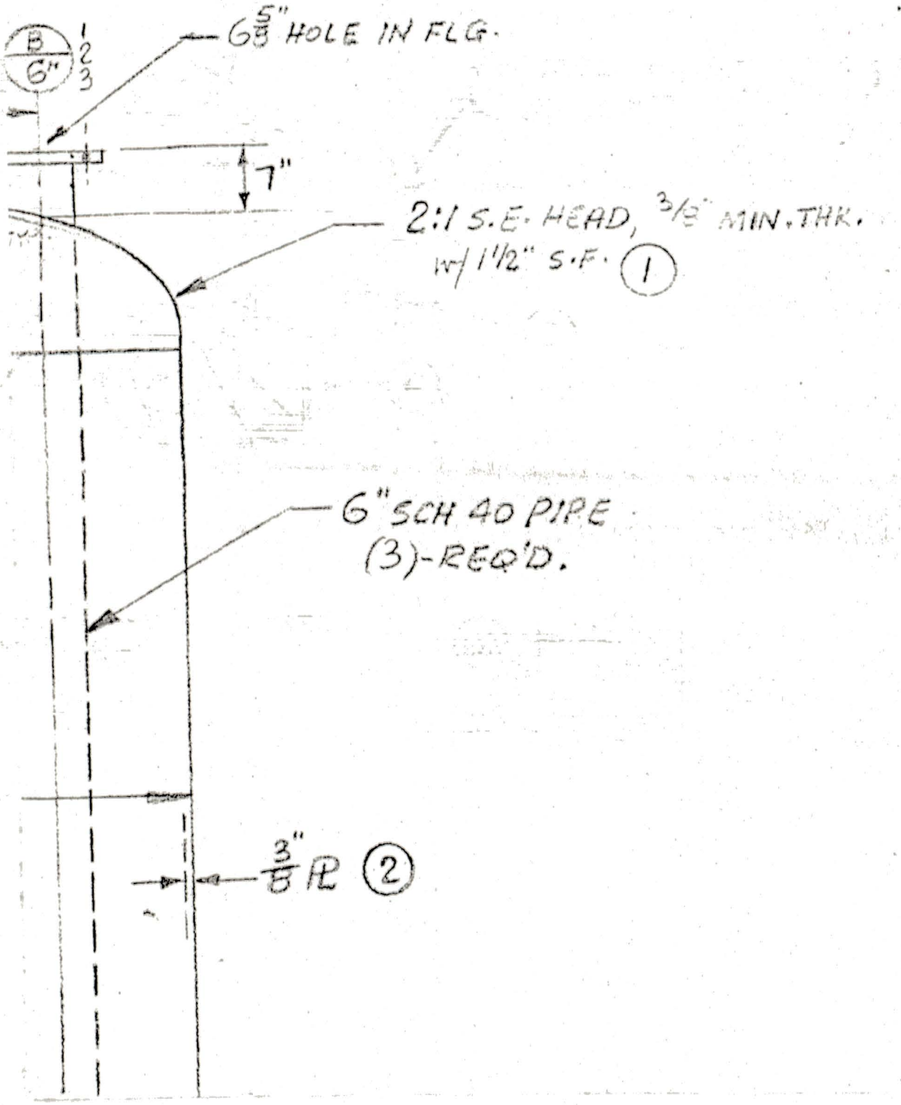
Very truly yours,

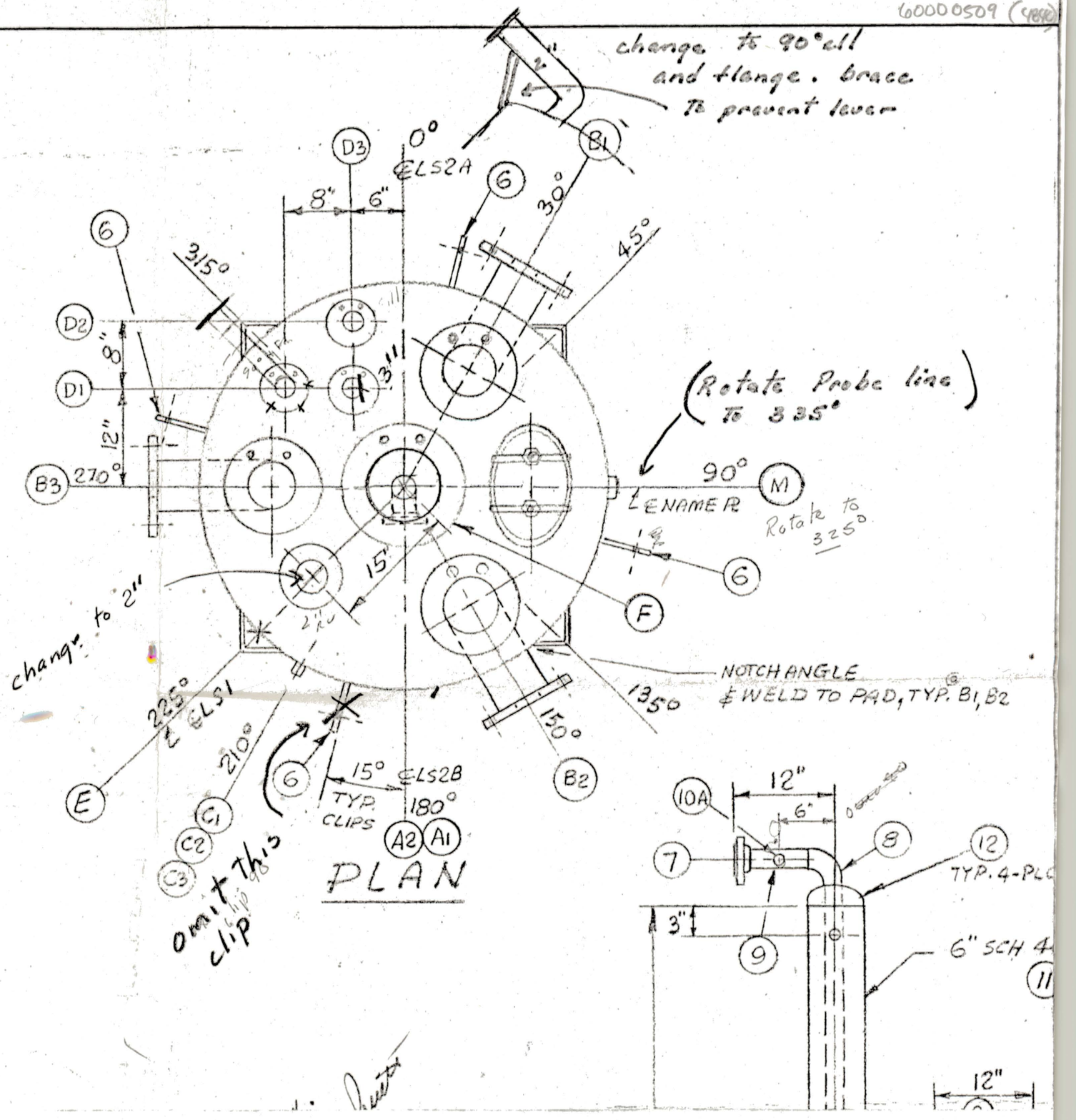
Scott Mansfield

Scott Mansfield

SM/cm

D.


B 6" 1 2 3
6 5/8" HOLE IN FLG.
7"
2:1 S.E. HEAD, 3/8" MIN. THK. w/ 1 1/2" S.F. (1)
6" SCH 40 PIPE (3)-REQ'D.
3/8" PL (2)


change to 90° all and flange. brace to prevent lever
D3
0°
℄LS2A
B1
8"
6"
6
30°
45°
315°
D2
D1
B3
270°
12"
8"
3"
(Rotate Probe line To 325°)
90°
M
℄ NAME R
Rotate to 325°
15°
F
2 1/2"
change to 2"
225°
℄LS1
210°
E
C1
C2
C3
omit this clip
15°
TYP. CLIPS
℄LS2B
180°
A2
A1
B2
135°
150°
NOTCH ANGLE & WELD TO PAD, TYP. B1, B2
PLAN
10A
12"
6"
7
8
12
TYP. 4-PLC
3"
9
6" SCH 4
11
12"

Met.
General - Correspondence &
Reports
127

HUGHES TOOL CO. 127
METALLURGY
METALLURGY - CORRES & REPORTS

Metallurgical Reports

August 7, 1975

TO: D. J. Gribbin

FROM: Robert E. Baker

SUBJECT: Gold Refining

Relative to the various discussions held this morning I would like to suggest that a thorough sampling, weighing and assaying be made of all silver and miscellaneous sludges, related to the stripping and electrolytic circuit.

Judging from a few random assays it appears that these residues are carrying a substantial amount of gold. It is possible that the interest on the money represented by this material would more than offset any additional refining cost for removing the silver from the circuit.

R. E. Baker

Distribution: Walt Simmons
Bill Mollison
Bill Robertson
Clarence Sikkenga
R. E. Baker
Tonopah files (2) ✓

Metallurgical Reports

August 8, 1975

TO: Mr. D. J. Gribbin

FROM: R. E. Baker

SUBJECT: Melting Furnace

Since my arrival on the scene it has been obvious that one of the bottlenecks in the refinery is the furnacing of the gold precipitates. At the high temperatures required the heatup time is prolonged and the electric muffle being used is in need of constant upkeep and repair.

As an alternative it is suggested that the possible use of an induction furnace be investigated. The possible advantage is that an induction furnace is a rapid means of heating to high temperature. In an induction furnace there are no heating elements to burn out. The source of heat is a high frequency current passing through water-cooled coils.

I have never used one of these furnaces for melting gold but have used one for production of eutectic tungsten carbide which requires a far higher temperature to melt than gold.

The cost will be high but maintenance will be extremely low. I would suggest looking into the subject and giving it careful consideration. It should not be gone into lightly but on the other hand, in the long run might do the job more satisfactorily than the present procedure.

Respectfully submitted,

Robert E. Baker

Distribution:

Walt Simmons
Bill Mollison

August 29, 1975

TEST M.S. #1 (MAGNETIC SEPARATION)

Procedure: 1000 grams of Manhattan Placer were taken and magnetically separated with a hand magnet. Each product (magnetic and non-magnetic) was retreated for the purpose of getting a good separation. The non-magnetic portion (288 grams) was placed in a bottle with 500 mls of water and 5 grams of slaked lime ($CaOH_2$) and 30 grams of mercury. The bottle was placed on the bottle roll machine for 20 hours. At the end of the rolling period the mercury was then separated from the pulp and dissolved in nitric acid to recover dore'. This gave an extraction of 90.246% of the dore' in the non-magnetic fraction.

Magnetic fraction (712 grams) was treated on the bottle rolls for gold extraction by cyanidation for a period of 24 hours. The extraction by cyanidation was 95.63%. This gave an overall extraction of 94.1%.

Refer to: Test #1 (Determination of Gold Extraction from Magnetic Portion of Placer Concentrates 8/8/75.)

Test #MS #1 (To Determine Distribution of Gold Between Magnetics and Non-magnetics in Manhattan Placer: 8/6/75)

MST = Non-magnetics
MSC = Magnetics

Bill Robertson

Dist: D. J. Gribbin
Manhattan Metallurgy office file
BR rf
Lab

lab -

July 7, 1975

TO: Walt Simmons

FROM: R. E. Baker

SUBJECT: Comments

While it is impossible to come up with many conclusions after a period of only three days of observation it is felt that the following comments are worth consideration:

1. The possibility of fouling the heap is rather remote if the sample that was brought to the lab is representative. There are soluble calcium, soluble sulfates and some soluble sulfides (very minor). The soluble calcium and sulfates will have a tendency to form calcium carbonate as a result of the carbon dioxide picked up from the atmosphere when solution is sprayed on the heap. This should be minimized by using caustic soda NaOH rather than lime CaO for control of alkalinity.

2. The sample of tailings, representing the pad material for the heap contained some soluble sulfide. In small amounts this should not be deleterious. The tendency of its effect might be to inhibit silver dissolution and to consume a little cyanide.

3. Due to the possibility of its creating foul solutions by the buildup of soluble sulfides, it is felt the barren solutions from the carbon desorption circuit should be discarded AFTER BEING ANALYZED and proven substantially barren.

4. The laboratory is apparently highly contaminated with gold values. This was shown when a series of tests was attempted on a low-grade tailings sample. The products, solution plus residue, were assayed and the total content was computed and compared to a head

assay. The computed assay from the products contained three or four times as much gold as was indicated by the head assay. Subsequent examination indicated that the equipment contained gold that was dissolved by the test solutions.

It is recommended that the assay office and laboratory be completely isolated from the recovery portion of the plant, particularly any section where carbon is used.

It is also recommended that all laboratory equipment be soaked with cyanide solution, water washed and then rinsed with weak hydrochloric acid, then rinsed with water. Hopefully this will prevent the possibility of "salting" and allow tests to be duplicated and allow a metallurgical balance to be obtained.

5. If a series of tests is planned, which seems to be indicated, it is further suggested that a set of laboratory bottle rolls be obtained so that more than one or two tests can be in progress at the same time.

Respectfully submitted,

Robert E. Baker

REB:sfm

Distribution:
D. J. Gribbin
R. E. Baker
Clarence Sikkenga
Bill Mollison
Tonopah files ✓

lab

July 9, 1975

TO: Walt Simmons

FROM: R. E. Baker

SUBJECT: Dissolution Rate of Au + Au/Ag Alloys

The following information was submitted by George Potter of the U. S. Bureau of Mines, Salt Lake Station.

Reference: American Cyanimid Co.
CHEMISTRY OF CYANIDATION
page 7

Cyanide strength 2 lb./ton of solution (constant stirring)

%Au	%Ag	Dissolution Rate mg/cm^2/hr
100	0	2.99
79.8	20.2	2.44
57.6	42.4	1.94
0.0	100.0	1.54

The Bureau of Mines in Salt Lake confirmed these results.

REB:sfm

R. E. Baker

lab

July 9, 1975

TO: Walt Simmons

FROM: R. E. Baker

SUBJECT: Estimation of Wet Carbon in Strip Tank
Sample taken July 9, 1975

S. G. = 1.20 Carbon and solution at same level

Tank Volume $\cong$ 30 cu. ft.

so, 30 x 62.4 x 1.20 = 2247 lbs.

This figure will vary depending upon the length of time the carbon is allowed to settle.

It was computed on the basis of having the carbon and the solution at the same level.

REB:sfm

R. E. Baker

July 15, 1975

TO: Walt Simmons

FROM: R. E. Baker

SUBJECT: Laboratory Equipment -- Bottle Rolls

It is felt that bottle roll testing in cyanidation work offers the most rapid method of obtaining preliminary and necessary information.

A set-up whereby a multiplicity of tests can be run at the same time has the following advantages:

1. Several samples can be cut at one mixing. If irregular free gold content is a problem, it will be indicated by irregular assays, as well as non-conforming metallurgical balances (computed head assays). Take averages.

2. Assuming no free gold problems, then a series of tests can be run varying time of leach.

3. Another series should be run varying cyanide strengths at optimum time.

4. Another series should next be run in which the caustic (or lime) strength is varied at previously determined optimum conditions.

5. Only after the conditions have been determined should column tests be started.

6. The leaching of coarser ore, in a column will involve a longer time to effect extraction; so, while this is being done, tests on another sample of the same

ore from a different section of the deposit should be run under optimum conditions which have been previously determined.

7. Screen analyses of residues from the above tests will determine the nature of the ore and the gold occurrence. Titrations on each test will show reagent consumption.

8. It should be possible for one man to manipulate six bottle roll tests during the course of a day, so it is felt that a set of bottle rolls to accommodate six bottles should be obtained. The labor cost and time to obtain information will easily justify the cost of the equipment.

9. The laboratory staff is endeavoring to maintain cleanliness. However, there is still some of the equipment that should be cleaned of residual carbon that might or might not have precious metal values adhering to it. This can be gone over as time allows. Laboratory equipment should be cleaned both before and after use. The ore samples are, in general, low grade and it does not take very much gold to make a salt that would render the test unreliable.

It is only with tedious and repetitive work that design criteria can be established.

As time goes on, more testing equipment might be suggested, but it is felt that at present ample bottle roll facilities are of paramount importance.

Respectfully submitted,

Robert E. Baker

REB:sfm

Dist: DJG
WOM
CS
Files: Lab Reports, R. E. Baker

Metallurgical Reports

August 29, 1975

TO: Summa Corporation
Mining Division
Distribution

FROM: Robert E. Baker
Consulting Metallurgical Engineer

SUBJECT: Comments and Discussion of Visit to Tonopah

First, it should be stated that the cordial and cooperative manner in which thoughts and suggestions were received is beyond reproach. I wish to thank all of the staff for their assistance.

A. The very first and most obvious condition to be overcome was the general housekeeping. One or two bottle roll cyanidation tests indicated that the laboratory and most of the laboratory equipment was horribly "salted". It was, therefore, felt imperative to have a general clean-up. In making small scale tests it is necessary to have a metal balance between the amount of metal started with and the sum of the metal contained in the products. If this metallurgical balance is not achieved the test data is considered unreliable. Metallurgical balances are difficult to obtain when there are varying amounts of relatively coarse free gold and it sometimes becomes necessary to run tests and assays in duplicate or triplicate in order to confirm results. It was, therefore, felt necessary to establish cleanliness in order to eliminate one source of potential error.

It has been observed over a long period of time that sloppy working conditions make for sloppy work.

B. Leach Pad Material:

It was requested that some tests be run in order to determine whether or not the old mill tailings, which were proposed for use as pad material for the heap leach, contained any material that would be chemically deleterious to the extraction of gold and silver from the ore. Two bottle roll cyanidation tests indicated that, while there was practically

no gold or silver extracted, there was very little or no cyanide or lime consumption and no indication of foul solutions. It was, therefore, felt that these old tailings would be suitable material for use in constructing the heap leach pad.

C. Manhattan Mine:

Several bottle roll leach tests were run on different Manhattan ore types and grades. The main purpose of these tests was to show the techniques used in making such tests as well as to indicate potential extractions by leaching. These tests were all 24-hour bottle roll tests. Depending upon grade of head material, these tests indicated that an extraction of about 50% of the gold might be extracted. In order to further delineate time and recovery some column tests on coarse ore must be made. Most of the material needed for the set-up is on hand and only the necessary time and availability of personnel is required to do this work.

Some of the Manhattan ore was crushed, screened and panned. The panned concentrates were subjected to bottle roll cyanidation and the solution was removed by filtration each day.

Days	% Au Dissolved
1	14.97
2	62.87
4	2.99
Total:	80.83

This test shows that at ± 10 mesh only 80% of the gold is liberated and available for cyanide extraction. The panned concentrates contained 28.66% of the original gold and the panned tailings gave an extraction of 61.73% of their contained gold in 24 hours.

D. Mary Mine Ore:

Preliminary bottle roll tests on Mary Mine ore indicated that the tree types of ore samples which were submitted are amenable to cyanidation treatment.

Comments and Discussion -- 3

The three types of ore that were submitted are:

1. Sulfide: 0.278 oz/t Au, 0.404 oz/t Ag
2. Ore from ore shoot: 0.085 oz/t Au, 0.236 oz/t Ag
3. Dump Sample: 0.031 oz/t Au, 0.244 oz/t Ag

The three types of ore were subjected to both a 24-hour and a 48-hour bottle roll test on the pulverized ore. Results are as follows:

Sample	24-hr. Extraction	48-hr. Extraction
Sulfide	75.38%	88.30%
Ore Shoot	86.21%	75.46%
Dump	73.19%	Lost

The next series of tests was run on samples that were crushed to approximately ¼-inch and given a 24- and 48-hour bottle roll treatment. Results are as follows:

Sample	24-hr. Extraction	48-hr. Extraction
Sulfide	49.21%	67.04%
Ore Shoot	25.84%	29.23%
Dump	58.76%	49.98%

The results of this series of tests seem to indicate that there is carbonaceous material in the dump ore. This is shown by the fact that extractions of 48 hours are lower than those obtained during a 24-hour test period. These tests should be repeated for confirmation.

Cyanide and caustic consumption are quite high for the sulfide ore. They would probably increase considerably under heap leach conditions because the period of oxidation would be

increased, thereby liberating more sulfate ion which is an acid radical that would take more alkali to neutralize. It is probable that a more sophisticated system of solution purification will be required during operation.

It is therefore suggested that, if this ore is treated, a thorough and intensive test program be initiated.

E. Recovery of values from sodium sulfide precipitate:

The sodium sulfide precipitation of silver from the caustic-cyanide stripping of values from the pregnant carbon contains varying amounts of gold and silver that should be recovered. Tests during the past few days have shown that the metal can be recovered by roasting to remove the sulfur from the silver sulfide and then fluxing and melting the roasted material. Further work is in progress and indications are that a very substantial amount of both gold and silver can be recovered, as Dore' metal, by this means. The equipment is small and production of this material will be slow but substantial.

A test was run in which it was hoped to remove the sulfur by treating with hydrochloric acid. The reaction was satisfactory but the hazard incurred by the liberation of hydrogen sulfide gas makes this approach unfeasible.

F. Sodium Sulfide precipitation:

An attempt was made to lower the amount of sodium sulfide used for precipitating the silver to stoichiometric amounts. This failed, with a resulting lower grade gold in the bullion.

It is possible that clarification was not complete and some finely dispersed silver sulfide escaped precipitation. This step should be studied further.

G. Manhattan Placer:

A test was run on a sample of Manhattan Placer Concentrates. The material was first subjected to magnetic separation with a hand magnet. The non-magnetics, consisting of about 25% of the weight, were then amalgamated on the bottle rolls, with the addition of hydrated lime. The material was rolled for 16

hours. The resulting material was then panned for the recovery of amalgam.

The recovered amalgam was then parted with nitric acid and weighed to determine the recovery. The magnetic fraction was treated by bottle roll cyanidation for recovery of gold. Results were:

	Percent Recovery
Magnetics	71.2
Non-magnetics	28.8
Overall	94.1*

* Based on weighted average.

It is suggested that this material, (non-magnetics) be amalgamated in an amalgam barrel with no grinding media. Grinding would tend to flour the mercury and thus make it more difficult to separate by gravity treatment methods.

H. Assaying:

The Chiddy Method of solution assaying was instituted. This is a standard method used for assaying gold and silver solutions. It has the advantage of being relatively rapid and of allowing a larger and therefore more representative sample to be used. Thus, the effect of any discrepancy is minimized. The assaying, in general, appears to be in good hands. Duplicate assays are checking reasonably well. The presence of an occasional bit of coarse free gold is obvious. On important samples when such a condition is suspected, triplicate assays might be indicated. If one of the three seems out of line that assay should be eliminated or rechecked.

I. Testwork:

Due to the fact that heap leaching is now in vogue, and probably will be for some time to come, it is felt that two or three test columns be set up on a permanent basis. In this way, tests can be in conduct at all times, either for the purpose of running new samples or monitoring heaps that are in production.

A set of bottle rolls is now in operation so preliminary cyanidation tests can be run. These tests will give an indication of the amenability of a particular ore sample to cyanidation as well as to give an estimate of reagent consumptions.

It is hoped that the above comments will be of assistance in the future operation and to the ultimate success of the Summa operation.

Respectfully submitted,

Robert E. Baker

Robert E. Baker
Consulting Metallurgical Engineer

Distribution: Dave Gribbin
Walt Simmons
Clarence Sikkenga
Bill Robertson
Bill Mollison
Baker file
Tonopah file

~~Technical Info~~ Metallurgical Dept.

summa

Internal Communication

Date: September 17, 1975

To: WOM, DJG, WS

From: Susie Mollison

Subject: Spectrographic Analysis "Skinner #1 03/28/75" by USBM Reno Metallurgy Research Center (Sample of cracked Summa Bar)

Dr. Howard Heady, Director of the Analytical Group for the above center, telephoned the following to me this afternoon. (I had spoken to his assistant this morning requesting the information.) He believed the gold sample had come to him from Harold Heinen with a request to identify contaminants.

The elements he detected in the sample are in the following percentages:

Aluminum	.01
Copper	.06
Iron	.06
Magnesium	.002
Manganese	.2
Nickel	.01
Lead	.5
Silica	.0005
Tin	.004
Zinc less than	.1
Silver	4.0
Gold	major (the remainder of the sample)
Platinum	.003

Bechtel Corporation
Engineers—Constructors
Fifty Beale Street
San Francisco, California 94119

June 23, 1975

Mr. Walt Simmons
Summa Corporation
P. O. Box 1126
Tonopah, Nevada 89049

Dear Walt:

Briefly, this is a summary of what I discussed with you and Clarence today on the telephone. I would suggest first taking a sample of the tailings you are using as a pad under the leach heap, agitating this sample in a laboratory flotation cell with a cyanide solution, and then turning on the air to see if you can develop a froth similar to that you are getting in your carbon column. Samples of this froth could be compared to samples of the froth from the carbon column by a laboratory such as Metallurgical Laboratories here in San Francisco to determine if they are the same.

If you are able to generate a froth from the tailings in a laboratory float cell I would suggest washing some tailings with alcohol and then acetone. This should remove the organic. Then dry the tailings to remove the acetone and treat them in the float cell with a cyanide solution. There should be no foaming after the acetone wash. It is also possible that some of the desert plants or bushes are contributing to the problem. There are undoubtedly some of these plants getting into the leach heap and since desert plants contain high percentages of creosotes, these may be the source of the foaming. Since there is no possible way of removing the tailings from under the leach heap, I would suggest treating some of the pregnant solution from the heaps in the laboratory flotation cell to see if the organics can be removed by the introduction of air bubbles much the same as they are removed from either the raffinate or the feed to the electrolytic plant from a solvent copper extraction process. At Ranchers, for instance, both the raffinate and the electrolytic feed are treated in this manner to remove last traces of the organic.

It would be easy enough to make a flotation cell in which to treat your carbon column feed. All you need is a long trough with a canvas false bottom in it under which air would be introduced at low pressure. This would be nothing more than an old Callow flotation cell which was used many years ago. In fact, Kennecott Copper at Ely, has just thrown out their last ones. Another type which would be easy to make would be similar to the Forrester flotation cell. The Callow cell is shown on Page 12-55 of Taggart's Handbook of Ore Dressing, and the Forrester cell is shown on Page 12-58.

If laboratory tests give you any indication that the foaming agent is coming from old tailings, or even if they don't, I would be more than pleased to help you out by taking samples of both the ore and the tailings to Martin Quist, at Metallurgical Laboratories here in San Francisco, to see if we can determine what the problem is.

So far this afternoon (June 23) I have been unable to locate Bob Baker who was in my office this morning. As I mentioned over the telephone, he is now living at 215 Nihell Street, Nevada City, California, (916) 265-5913. I have written him a note, however, asking him to call you as soon as he gets home.

Very truly yours,

R. S. Shoemaker
Consulting Metallurgist
Mining & Metals Division

RSS/nk

Metallurgical Reports

Universal Silver

649-1810

A DIVISION OF PAXTON ENTERPRISES, BOX AZ, MAIN P.O., VENTURA, CALIF. 93001, (805) 643-7447

October 21, 1975

Summa Corp.
Mining Division
P. O. Box 1126
Tonopah, Nevada 89049

Attention: Mr. William Mollison

Gentlemen:

Here are the results of refining tests run on samples of sludges received from you on September 23, 1975. We also enclose the gold Dore' button and silver derivative from these samples, and a factor sheet of smelting and refining costs.

Wet Sludge

Moisture	71.270%
Assay	.037 au. .295 ag.

Dry Sludge

Moisture	15.40%
Assay	.041 au. .088 ag.

Samples run for plant recovery, 100 grams each.

Wet sludge derivative - 3 samples. Dry sludge derivative - 6 samples. Total: 9 samples, totaling 900 grams.

Results

Wet	11.10 grams au. 88.50 grams ag.

Dry 24.600 grams au.
52.800 grams ag.

Totals

11.10	88.500
24.600	52.800

35.700 grams gold 141.300 grams silver

Total: 177 grams

Average Assay

Au. .040

Ag. .157

Smelting and Refining Costs

Smelting

.900 kilo x .000366 equals .000329 kilo gold

Dore' - 180.2 grams

.198 au.
.784 ag.

Total .982

Return on .198 au. .1820 (from factor sheet)

.1820 x 180.2 (Dore' wt.) dquals 32.796
Minus smelter costs .329

32.467 grams au.

Return on .784 ag. .6331 (from factor sheet)

.6331 x 180.2 (Dore' wt:) equals 114.084 grams ag.

Thank you for sending us your sludge samples. We look forward to hearing from you.

Very truly yours,

Douglas R. Paxton

DRP:jp
Enc.

Universal Silver

A DIVISION OF PAXTON ENTERPRISES, BOX AZ, MAIN P.O., VENTURA, CALIF. 93001, (805) 643-7447

649-1810

PROVISIONS APPLYING TO FABRICATION COST SCHEDULE

1. All prices will be fixed by contract or as quoted at time of placing order. Date and time of shipment will be established by contract or at time of acceptance of order. Shipping dates subject to modification by circumstances beyond the control of Universal Silver Refinery. Buyer will be notified of any deviation from schedule.

2. To determine price of fabricated silver, multiply form, size and quantity factor times Universal Silver Refinery quoted base (or spot) silver price. This price is for prepaid or contract c.o.d. orders.

With a 10% deposit and balance 10 days, add 1%.

With a 10% deposit and balance 30 days, add 2%.

3. Universal Silver Refinery's quoted base (or spot) silver price is predicated upon world silver prices as traded on the major exchanges of the world, minus the cost of shipping from refinery to exchange, and may be obtained at any hour by calling (805) 649-1810. The recording is regularly up-dated during trading hours in the United States, or upon any significant change in the world prices.

4. This schedule applies to silver of commodity or commercial purity (.999 pure), superior grade (.9995 pure) and super fine grade (.9999 pure). Universal Silver Refinery adheres rigidly to specifications for fine silver as established by the American Society for Testing & Materials (A.S.T.M. Designation B413-69). Assay reports from independent testing laboratories furnished with superior grade and super fine grade materials.

5. Other shapes, forms and sizes available, and prices will be quoted on request.

6. Prices are f.c.b. our plant, shipped freight and insurance collect by carrier designated by buyer. Special packaging and handling costs will be quoted at time of purchase.

GOLD REFINING ACTUAL COSTS
FROM DORE' ANODES TO →

ASSAY	FURNACE & REFINING LOSSES	REFINING COSTS	TOTAL COSTS & LOSSES	RETURN UNDER ASSAY	COMMODITY GRADE ASTM STANDARDS NO. B 562-73 .995 PURE ON CONTRACT	COMMODITY GRADE ASTM STANDARDS NO. B 562-73 .995 PURE JOB LOTS	SUPERIOR GRADE ASTM STANDARDS NO. B 562-73 .9995 PURE ON CONTRACT	SUPERIOR GRADE ASTM STANDARDS NO. B 562-73 .9995 PURE JOB LOTS	SUPERIOR GRADE ASTM STANDARDS NO. B 562-73 .9995 PURE LESS THAN 1 KILO	SUPERIOR GRADE ASTM STANDARDS NO. B 562-73 .9995 PURE 1 TO 10 KILOS
.999	.0006	.0133	.0139	.9861			.9851	.9784	.9203	.9453
.990	.0020	.0150	.0170	.9830	.9732	.9665	.9699	.9632	.9084	.9334
.950	.0023	.0167	.0190	.9810	.9320	.9253	.9287	.9220	.8672	.8923
.900	.0023	.0183	.0206	.9794	.8815	.8748	.8782	.8715	.8167	.8417
.800	.0025	.0200	.0225	.9775	.7820	.7753	.7787	.7720	.7172	.7422
.700	.0028	.0217	.0245	.9755	.6829	.6762	.6796	.6729	.6181	.6431
.600	.0034	.0233	.0267	.9733	.5840	.5773	.5807	.5740	.5192	.5442
.500	.0040	.0250	.0290	.9710	.4855	.4788	.4822	.4755	.4207	.4457
.400	.0050	.0267	.0317	.9683	.3873	.3806	.3840	.3773	.3225	.3475
.300	.0067	.0283	.0350	.9650	.2895	.2828	.2862	.2795	.2247	.2497
.200	.0100	.0300	.0400	.9600	.1920	.1853	.1887	.1820	.1272	.1522
.100	.0200	.0317	.0517	.9483	.0948	.0881	.0915	.0848	.0300	.0550
.050	.0400	.0333	.0733	.9267	.0463	.0396	.0430	.0363	Ø	.0065

	SUPER FINE GRADE ASTM STANDARDS NO. B 562-73 .9999 PURE ON CONTRACT	SUPER FINE GRADE ASTM STANDARDS NO. B 562-73 .9999 PURE JOB LOTS	ASTM STANDARDS NO. B 562-73 .99995 PURE ON CONTRACT	ASTM STANDARDS NO. B 562-73 .99995 PURE JOB LOTS
.999	.9718	.9651	.9452	.9385
.990	.9599	.9532	.9333	.9266
.950	.9187	.9120	.8921	.8854
.900	.8682	.8615	.8416	.8349
.800	.7687	.7620	.7421	.7354
.700	.6696	.6629	.6430	.6363
.600	.5707	.5640	.5441	.5374
.500	.4722	.4655	.4456	.4389
.400	.3740	.3673	.3474	.3407
.300	.2762	.2695	.2496	.2429
.200	.1787	.1720	.1521	.1454
.100	.0815	.0748	.0549	.0482
.050	.0330	.0263	.0064	.0003

SMELTER COSTS - GOLD

Primary Material Being Reduced to Dore' Bars

Job lots less than 1,000 kilo

First Melt - .000366 kilo gold per incoming kilo

Job lots over 1,000 kilo

First Melt - .000244 kilo gold per incoming kilo

On Contract

First Melt - .000179 kilo gold per incoming kilo

Dore' Material Poured into Anodes or Shot

Job Lots - .000122 kilo per incoming kilo

On Contract - .000116 kilo per incoming kilo

Assay	Average Smelter Loss	Assay	Average Smelter Loss	Assay	Average Smelter Loss
990	.0009	.500	.0062	.025	.0118
950	.0017	.400	.0071	.010	.0127
900	.0025	.300	.0081	.005	.0136
800	.0034	.200	.0090	.002	.0146
700	.0043	.100	.0099	.001	.0204
600	.0053	.050	.0108		

Sampling and Assay Costs

Reduction of primary material to Dore'	$25.00
Fire assay including sample preparation	
To .999	$20.00
To .9999	$35.00
Spectrographic analysis including sample preparation	
Semi-Quantitative	$35.00
Exact analysis by element	$15.00 add'l.

SMELTER COSTS - SILVER

Primary Material Being Reduced to Dore' Bars

Job lots less than 1,000 kilo

First Melt - .00927 kilo silver per incoming kilo

Job lots over 1,000 kilo

First Melt - .00618 kilo silver per incoming kilo

On Contract

First Melt - .00453 kilo silver per incoming kilo

Dore' Material Poured into Anodes or Shot

Job Lots - .00396 kilo per incoming kilo

On Contract - .00291 kilo per incoming kilo

Assay	Average Smelter Loss	Assay	Average Smelter Loss	Assay	Average Smelter Loss
.900	.0099	.500	.0597	.025	.1194
.950	.0149	.400	.0696	.010	.1291
.900	.0199	.300	.0796	.005	.1390
.800	.0298	.200	.0895	.002	.1489
.700	.0398	.100	.0995		
.600	.0497	.050	.1094		

SMALL LOT HANDLING CHARGE - $25.00 (TO COVER SAMPLING & ASSAYING COSTS)

Sampling and Assay Costs

Reduction of primary material to Dore'	$25.00
Fire assay including sample preparation	
To .999	$20.00
To .9999	$35.00
Spectrographic analysis including sample preparation	
Semi-Quantitative	$35.00
Exact analysis by element	$15.00 add'l.

Universal Silver

A DIVISION OF PAXTON ENTERPRISES, BOX AZ, MAIN P.O., VENTURA, CALIF. 93001, (805) 643-7447

REFINING COSTS

Assay	Furnace & Refining Loss	Refining Costs	Total Costs	Return Under Assay	Commodity Grade Return % of Gross on Contract	Commodity Grade Return % of Gross on Job Lots	Superior Grade Return % of Gross on Contract	Superior Grade Return % of Gross on Job Lots
.990	.0098	.0318	.0416	.9584	.9488	.9238	.8908	.8538
.950	.0148	.0332	.0479	.9521	.9045	.8795	.8465	.8095
.900	.0197	.0350	.0547	.9453	.8507	.8257	.7927	.7557
.800	.0296	.0394	.0689	.9311	.7448	.7198	.6868	.6498
.700	.0395	.0450	.0845	.9155	.6408	.6158	.5828	.5458
.600	.0494	.0525	.1019	.8981	.5388	.5138	.4808	.4438
.500	.0592	.0630	.1222	.8778	.4389	.4139	.3809	.3439
.400	.0690	.0788	.1478	.8522	.3408	.3158	.2828	.2458
.300	.0789	.1050	.1839	.8161	.2448	.2198	.1868	.1498
.200	.0888	.1575	.2463	.7537	.1507	.1257	.0927	.0557
.100	.0987	.3150	.4137	.5863	.0586	.0336	.0006	Ø
.050	.1184	.6300	.7484	.2516	.0125	Ø	Ø	Ø

SMELTER COSTS

Job Lots - 6.18 grams of silver per incoming kilo

Contracts - 3.43 grams of silver per incoming kilo

Assay	Average Smelter Losses	Assay	Average Smelter Losses	Assay	Average Smelter Losses
.990	.0099	.500	.0597	.025	.1194
.950	.0149	.400	.0696	.010	.1291
.900	.0199	.300	.0796	.005	.1390
.800	.0298	.200	.0895	.002	.1489
.700	.0398	.100	.0995		
.600	.0497	.050	.1094		

SMALL LOT HANDLING CHARGE - $25.00 (TO COVER SAMPLING & ASSAYING COSTS)

This factor sheet is based upon operating costs of .0237 oz. per incoming oz.
Investment amortization of .00629 " " " "
Brokerage and commissions of .0015 " " " "

Total .03149 " " " "

Universal Silver

649-1810

A DIVISION OF PAXTON ENTERPRISES, BOX AZ, MAIN P.O., VENTURA, CALIF. 93001, (805) 643-7447

FABRICATION & HANDLING COSTS

(Universal Silver Refinery "Spot" or Base Price as Quoted X Factor Equals Cost)

	Over 100 Kilo Over 3,000 Troy		30 to 100 Kilo 1,000 to 3,000		10 to 30 Kilo 300 to 1,000 Troy	
	Contract	Job Lot	Contract	Job Lot	Contract	Job Lot
COMMODITY GRADE .999 PURE						
As Cast	.9990	1.000	.9995	1.004	.9998	1.009
1,000 Troy Bar	1.005	1.006	1.005	1.011	1.006	1.011
Mixed Shot	1.015	1.016	1.015	1.023	1.016	1.030
Lg. Nuggets-Popcorn Shot	1.022	1.026	1.023	1.032	1.022	1.039
Grain-Medium Shot	1.022	1.026	1.023	1.032	1.022	1.039
Grain-Small Shot	1.022	1.026	1.023	1.032	1.022	1.039
Flat Anodes-15x3x1/2	1.032	1.043	1.036	1.047	1.040	1.051
Cut Length Anodes	1.036	1.048	1.040	1.051	1.044	1.059
Dog Bone Anodes	1.038	1.059	1.042	1.046	1.054	N/R
Kilo Bars	1.034	1.046	1.037	1.050	1.042	1.057
100 Troy Bars	1.034	1.050	1.038	1.054	1.046	1.072
Planchets	1.070	1.090	1.074	N/R	1.082	N/R
SUPERIOR GRADE .9995 PURE						
As Cast	1.022	1.060	1.028	1.085	1.055	1.119
1,000 Troy Bar	1.028	1.067	1.034	1.092	1.061	1.121
Mixed Shot	1.027	1.066	1.033	1.091	1.061	1.130
Lg. Nuggets-Popcorn Shot	1.035	1.076	1.041	1.101	1.067	1.139
Grain-Medium Shot	1.035	1.076	1.041	1.101	1.067	1.139
Grain-Small Shot	1.035	1.076	1.041	1.101	1.067	1.139
Flat Anodes-15x3x1/2	1.045	1.093	1.054	1.118	1.085	1.151
Cut Length Anodes	1.049	1.098	1.058	1.123	1.089	1.159
Dog Bone Anodes	1.051	1.109	1.060	N/R	1.099	N/R
Kilo Bars	1.046	1.096	1.055	1.120	1.087	1.156
100 Troy Bars	1.047	1.100	1.056	1.125	1.091	1.172
Planchets	1.083	1.140	1.092	N/R	1.127	N/R
SUPER FINE GRADE .9999 PURE						
As Cast	1.036	1.110	1.045	1.139	1.100	1.168
1,000 Troy Bar	1.042	1.117	1.052	1.146	1.106	1.170
Mixed Shot	1.042	1.116	1.051	1.148	1.105	1.178
Lg. Nuggets-Popcorn Shot	1.049	1.126	1.059	1.157	1.112	1.188
Grain-Medium Shot	1.049	1.126	1.059	1.157	1.112	1.188
Grain-Small Shot	1.049	1.126	1.059	1.157	1.112	1.188
Flat Anodes-15x3x1/2	1.059	1.143	1.072	1.171	1.130	1.200
Cut Length Anodes	1.063	1.388	1.076	1.176	1.134	1.208
Dog Bone Anodes	1.065	1.159	1.078	N/R	1.144	N/R
Kilo Bars	1.061	1.145	1.073	1.174	1.132	1.205
100 Troy Bars	1.061	1.150	1.074	1.179	1.136	1.221
Planchets	1.097	1.190	1.110	N/R	1.172	N/R

Copies 10-28-75: WS
DJG
CS
Tono Office
(orig WOM)
Stan.

United States Department of the Interior

BUREAU OF MINES
1605 EVANS AVENUE
RENO, NEVADA 89505

October 23, 1975

Mr. Wm. D. Mollison
Summa Corporation
P. O. Box 1126
Tonopah, Nevada 89049

Dear Mr. Mollison:

This is in regard to the analyses of the three silver-gold products, from your carbon desorption unit, that you submitted to this Center. We thought you would like a copy of the results we phoned to you today. The products were analyzed for silver and gold by our normal fire assay procedure, carbon by the Leco method, and the other constituents by semi-quantitative spectrographic analysis. The results were:

Constituent	Ag_2S ppt	Elect. cell sludge	Agit. tank sludge
Gold, oz/t	1943	33.3	1169
Silver, oz/t	2055	15,359	21,621
Carbon, pct	14.0	4.9	1.0
Aluminum, pct	0.7	0.1	0.1
Copper, pct	0.03	0.01	<0.002
Iron, pct	0.4	0.06	0.03
Magnesium, pct	0.4	0.1	0.05
Manganese, pct	0.02	0.006	0.006
Nickel, pct	0.02	0.01	-
Lead, pct	0.07	0.01	-
Silicon, pct	5	0.5	0.1
Tin, pct	.008	-	-
Titanium, pct	.05	0.02	0.01
Zinc, pct	.5	0.3	-
Calcium, pct	Major	Major	0.3
Sodium, pct	1	0.5	0.5

Our analyses indicate that the silver sulfide precipitate was contaminated with 14 percent carbon. Microscopic examination revealed the presence of small grains of granular carbon very likely derived from the loaded

activated carbon that degraded during handling and stripping. The high gold contents of the Ag_2S precipitate also indicate that a significant amount of loaded carbon particles and fines are being discharged with the enriched Ag-Au cyanide strip solution and eventually are recovered in the filtration of the Ag_2S flocs. The main source of abraided carbon is probably from the bituminous activated carbon that was blended with the coconut shell carbon at your plant. In our desorption studies conducted on loaded coconut shell type carbon, we have not observed any significant degradation. The major gold producers in the U.S. using the carbon-adsorption process (Homestake Mining Co. and Cortez Gold Mine) employ granular activated coconut carbon because it is harder and more abrasion resistant than other types of carbon.

During our telephone conversation, you mentioned that Summa's heap leaching operation at Manhattan was recovering more silver than they had previously and that the higher silver loadings on the carbon caused additional problems in the desorption plant. We suggested you consider precipitating the silver from the pregnant effluent with Na_2S prior to adsorbing the gold on activated carbon. As supplemental information we are forwarding to you a copy of an oral presentation on our laboratory and pilot plant studies on above subject.

We trust this information will help you solve the metallurgical problems concerning desorption of your loaded carbon.

Sincerely yours,

Harold J. Heinen

Harold J. Heinen
Metallurgist

Enclosure

SILVER EXTRACTION FROM MARGINAL RESOURCES

by

H. J. Heinen, D. G. Peterson, and R. E. Lindstrom

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ABSTRACT

Modification of conventional cyanidation processes are being investigated by the Bureau of Mines to develop more efficient methods of extracting silver values from mine wastes and low-grade and refractory ores. One technique developed involves the selective precipitation of silver as Ag_2S from cyanide solutions followed by recovery of residual gold values on activated carbon. Carbon-in-pulp cyanidation appears promising for processing slimy tailings with the development of an efficient technique for stripping silver from loaded carbon with alkaline ethanol solutions at ambient temperature and pressure. Salt roasting-cyanidation treatment of a refractory ore gave 88-percent silver extraction compared with 60 percent attained by conventional cyanidation. The chloridized calcine is amenable to percolation or heap leaching with dilute cyanide solution.

INTRODUCTION

Bureau of Mines surveys (1-2) indicate that the current domestic consumption of silver continues to exceed silver production from both primary sources as well as that recovered from secondary sources or recycled materials. Currently, only 30 percent of the silver produced comes from ores in which silver was the principal metallic value. Seventy percent of the primary silver is produced as a co-product of copper, lead, and zinc. The projected domestic demand for primary silver by the year 2000 will be two to four times as great as it is now, thus resulting in a cumulative demand of 3.7 to 5.7 billion ounces. This demand would very likely use up our known domestic reserves, which are estimated at 4.9 billion ounces, of which 1.3 billion ounces is attainable from currently operating base-metal mines. Approximately 3.5 billion ounces occur mostly in submarginal resources of precious metal ores, which require either higher silver prices with existing silver-recovery processes, or improved processing techniques to be commercially exploitable. In addition, there are a number of old mine dumps and silver mill tailings in the

Western States that contain 2 to 5 ounces silver per ton and are partially amenable to cyanidation. These resources range from a few thousand to several million tons, frequently in old mining districts that no longer have any milling facilities available. These tailings in aggregate still contain a substantial amount of silver, but on an individual basis, they are too small and too low in grade to justify the capital investment for a new conventional cyanide plant. These projections stimulated the Bureau of Mines to investigate the development of low-capital, low-operating-cost alternatives to conventional methods for silver recovery from low-grade ores, and from resources that are too small to justify the capital expenditure required for conventional processing.

The Bureau of Mines is conducting research to develop more efficient and economical methods for extracting silver and associated gold values from marginal resources with dilute cyanide leach solutions. This report describes the research on (1) heap leaching and downstream processing steps, involving selective precipitation of silver with Na_2S and carbon adsorption of gold; (2) carbon-in-pulp cyanidation and the stripping of adsorbed precious metal values from activated carbon with a newly developed alkaline alcohol solution; and (3) chloridization roasting and cyanidation of an oxidized refractory ore. Most of the previous salt-roasting studies were conducted on complex silver-bearing sulfide ores.

RESULTS AND DISCUSSION

Heap Leaching

Although heap leaching with dilute cyanide solutions has been applied successfully to recovery of gold from low-grade ores, very little work has been done on extending this technique to recovery of silver from low-grade ores and mine dump material. Recent increases in the price of silver have stimulated interest in percolation leaching of silver ores, and our studies indicate that heap leaching may possess potential for treating certain ores, but that the problems encountered and the results achieved are measurably different from those obtained in treating gold ores. This is attributed to the complexity and large variety of silver-bearing minerals compared with that of gold. Usually gold occurs in the native state or as telluride; whereas, silver may occur as several different minerals in the same ore and in the crystal lattices of manganese and iron oxides, and jarosite minerals. In percolation leaching tests conducted on 2- to 5-ounce silver ore containing small amounts of gold, the gold very frequently dissolved at a greater

rate and more completely than the silver. Moreover, the data indicate that the extraction of silver with cyanide is more dependent on particle size of the ore than is gold.

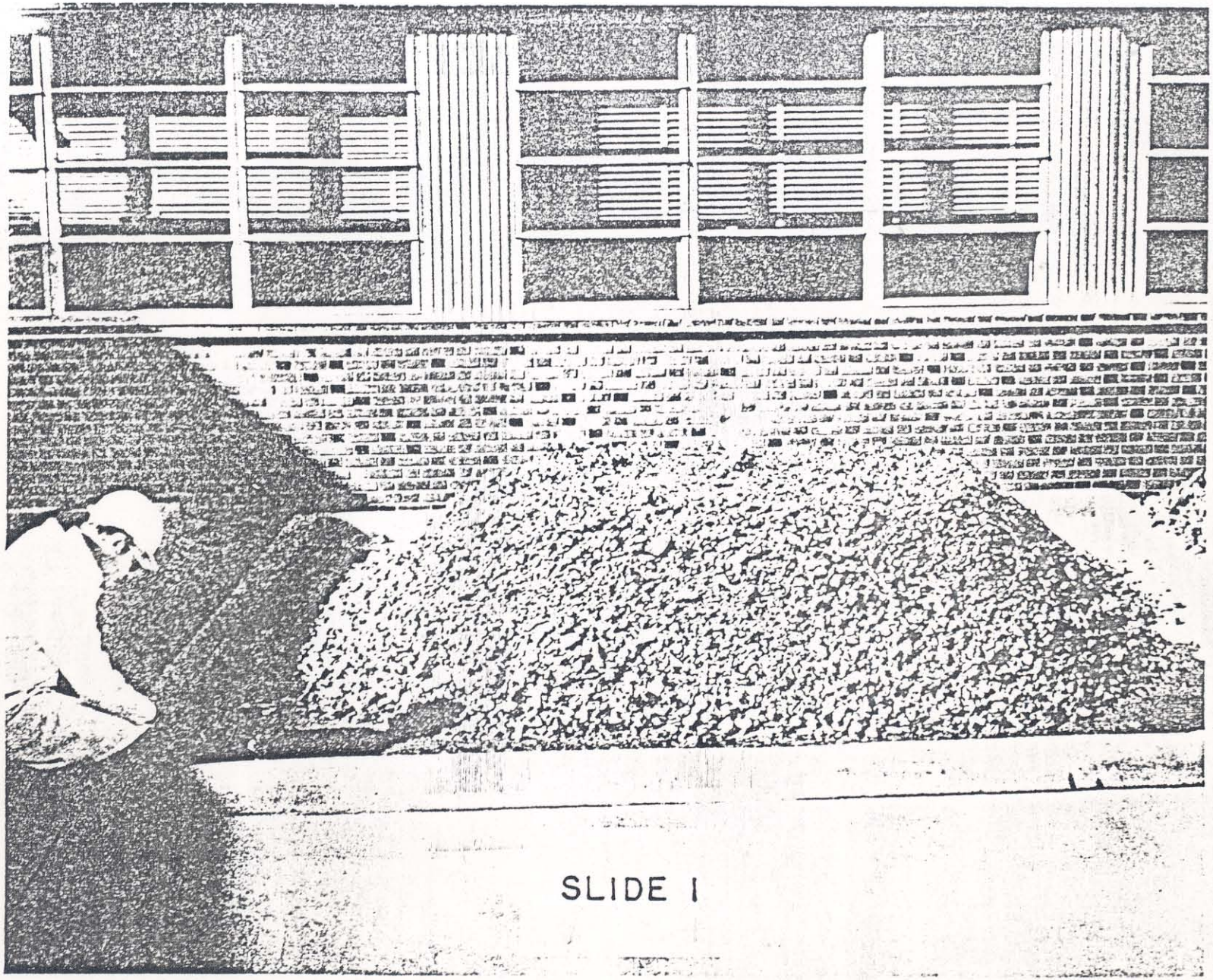
The technology for determining the amenability of a silver ore to percolation leaching with cyanide is well established. Typically, a bottle cyanidation test is conducted on finely ground ore to determine the degree of silver extraction. If the ore is leachable, column percolation tests are made on ore crushed to various sizes. Pilot-scale heap-leaching tests are conducted on a tonnage scale if it is desirable to further quantify reagent requirements. The percolation rate can best be measured in tonnage-scale experiments in 15- or 20-foot-high columns.

Results obtained in applying these test procedures to a refractory silver ore that contained 5.0 ounces silver per ton and 0.02 ounce gold per ton showed that approximately 60 percent of the silver and 75 percent of the gold is extracted in 24-hour bottle tests on ore ground to 65-percent minus 200 mesh, using 2 pounds sodium cyanide and 5 pounds lime per ton of ore. Corresponding percolation leach tests conducted in a 4-inch column on the ore crushed to 3/8 inch resulted in 40-percent silver extraction and 75-percent gold extraction in 7 days. Similar experiments conducted on ore crushed to 1 inch also resulted in 40-percent silver extraction and 75-percent gold extraction.

Subsequently, a pilot-scale heap-leaching experiment was conducted on 8 tons of the ore crushed to 1 inch, as shown in slide 1. The heap was leached for 3 weeks with a cyanide solution containing 1 pound NaCN per ton while maintaining alkalinity at a pH of about 10.5 with CaO. The effluent was pumped upward through columns of 6x16-mesh-size activated carbon, and the barren solution was recycled to the heap. Results showed that 2 ounces of silver and 0.015 ounce gold per ton were extracted. This silver recovery is about two-thirds of that attained by conventional cyanidation of finely ground ore.

Silver Recovery From Pregnant Cyanide Solutions

Activated carbon has been used frequently to recover gold from solutions generated in percolation leaching of low-grade ores. These recovery procedures are generally applicable to silver recovery from solution; however, in the particular case of metal recovery from the relatively clear solutions usually obtained by percolation leaching, there are two factors involved that make the carbon adsorption system less attractive for silver processing than it is for gold: (1) The inventory of carbon required to treat a



SLIDE 1

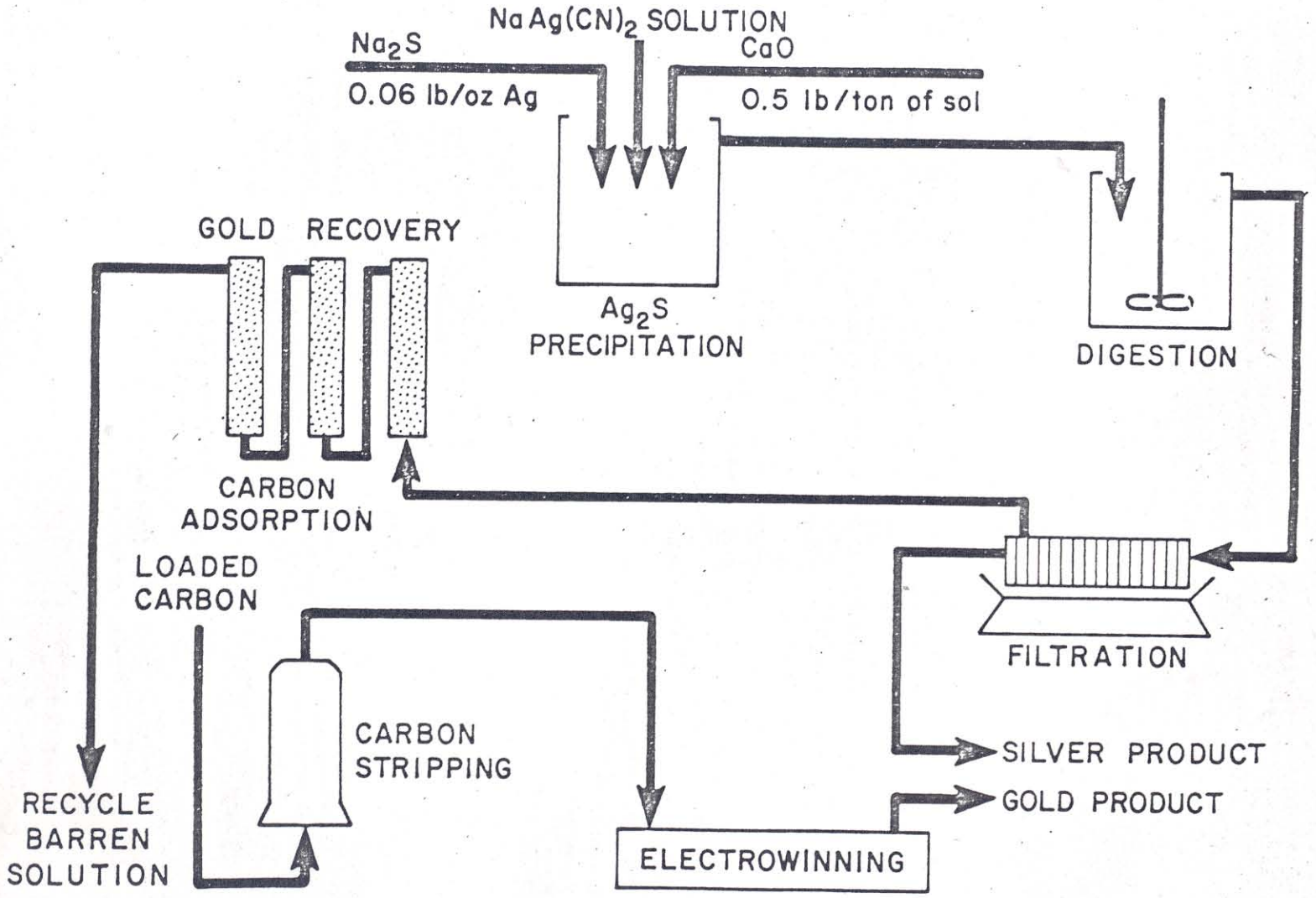
silver ore is nearly 30 times that required to treat a gold ore of equal value; (2) the capital equipment investment is correspondingly higher, and operating costs would be several times as high for stripping and electrowinning from the large amount of carbon, which contains approximately 3 percent of the monetary values typically encountered in gold processing.

In an attempt to avoid problems encountered in handling large quantities of activated carbon, the effectiveness of silver precipitation from solution with sodium sulfide (Na_2S) was investigated. Sodium sulfide has been employed for silver recovery from solution by other workers (6). However, its use was generally limited to atypical silver ores that contain essentially no gold, because gold is not precipitated by the sulfide and may eventually be lost in the system.

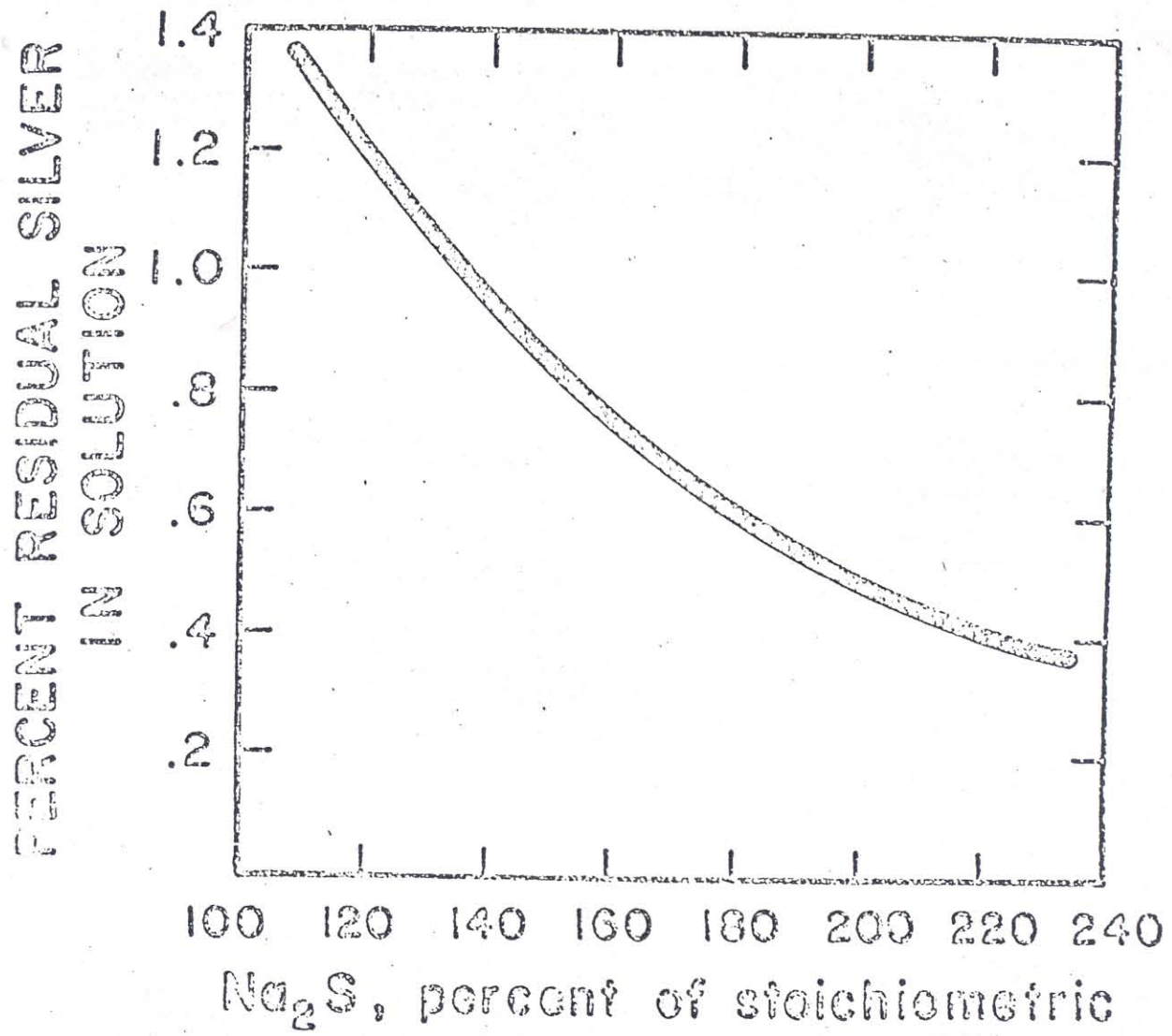
To overcome the possibility of losing gold in the system, a carbon column was inserted into the process sequence so that gold contained in solutions after precipitation of silver would be adsorbed on the column. The process sequence is shown in slide 2. By this procedure, the plant inventory of activated carbon is a mere fraction of that required for a carbon-in-pulp cyanidation treatment of silver-bearing mill tailings or other low-grade sources.

Laboratory-scale experiments were conducted on a synthetic cyanide solution using a silver tracer to delineate process variable relating to Ag_2S precipitation. Results indicate that only a slight excess (10 to 20 percent) of Na_2S reagent is required for precipitating over 98 percent of the total silver from solutions containing up to 2 pounds NaCN per ton. The effect of excess sulfide upon the silver recovery is shown in slide 3. Precipitation was complete with 1 hour of stirring. The key to the formation of readily filterable Ag_2S flocs is the addition of about 0.5 pound of CaO per ton of cyanide solution. Without the CaO additive, the Na_2S produces colloidal suspension of Ag_2S, which is very difficult to filter. In a test using a gold tracer, it was found that no gold was coprecipitated with the Ag_2S, and the residual gold and silver values in the Ag_2S filtrate were quantitatively adsorbed by activated carbon.

Recovery of silver as a sulfide precipitate from a nearly clear decanted pregnant solution obtained by cyaniding a sample of mill tailings was investigated. The pregnant solution, which contained 1 pound NaCN per ton, assayed 46 ppm silver (1.3 ounces per ton), <0.1 ppm gold 30 ppm copper, 29 ppm zinc, <3 ppm lead, and <5 ppm mercury. More than 99 percent of the silver was


Na_2S
0.06 lb/oz Ag
$NaAg(CN)_2$ SOLUTION
CaO
0.5 lb/ton of sol
Ag_2S
PRECIPITATION
DIGESTION
GOLD RECOVERY
CARBON
ADSORPTION
LOADED
CARBON
FILTRATION
CARBON
STRIPPING
RECYCLE
BARREN
SOLUTION
SILVER PRODUCT
GOLD PRODUCT
ELECTROWINNING

SLIDE 2


PERCENT RESIDUAL SILVER IN SOLUTION
1.4
1.2
1.0
.8
.6
.4
.2
100 120 140 160 180 200 220 240
Na_2S, percent of stoichiometric

SLIDE 3

precipitated with only 34 percent stoichiometric excess of sulfide based on the Ag^+ content only. No zinc or copper was coprecipitated with the Ag_2S. Evidently, the presence of comparable amounts of both copper and zinc in relation to silver had no adverse effect on precipitation of the silver.

The feasibility of adsorbing the gold values and residual silver values remaining in the filtrate from the Ag_2S precipitation on activated carbon was investigated in a series of laboratory-scale experiments utilizing radioactive tracers for analysis. A synthetic cyanide solution was prepared that contained 1.0 ounce silver per ton, 0.01 ounce gold per ton, 1.0 pound NaCN per ton, and 0.5 pound CaO per ton, and having a pH of 10.5. The silver was precipitated with two different concentrations of Na_2S. In one case, 90 percent of stoichiometric amount of Na_2S required to precipitate the silver as Ag_2S was employed, and in the other, 150 percent. After 18 hours of agitation, the Ag_2S precipitate was recovered by filtration. Then two series of carbon adsorption tests were conducted on the filtrate. In one series, a portion of the filtrate was tagged with radioactive silver-110m tracer prior to agitation with 6x16-mesh activated carbon, equivalent to 20 pounds per ton of solution, for 24 hours. The solutions and carbon were counted individually to accurately determine the amounts of silver adsorption. A similar series was conducted on aliquots of the Ag_2S filtrate tagged with radioactive gold-195 tracer. Results are summarized in slide 4.

SLIDE 4. - Gold-silver adsorption from Ag_2S filtrate

Test No.	Na_2S addition % of theoretical	Ag_2S ppt % of total Ag	Carbon adsorption			
			Soln heads		Soln tails	
			oz Ag/T	oz Au/T	oz Ag/T	oz Au/T
1	90	88.4	0.116	0.01	0.006	0.00007
2	150	93.2	.068	.01	.004	.00006

These results show that gold was not coprecipitated with the silver sulfide, and that the activated carbon adsorbed essentially all of the residual silver and gold values contained in the filtrate.

The Ag_2S precipitation-carbon adsorption processing was evaluated further in a pilot-scale heap-leaching operation. The procedure was to heap leach 8 tons of ore, crushed to about 1 inch, which assayed 4.4 ounces silver per ton and 0.04 ounce gold per ton, starting with 1.5 tons of cyanide solution containing 4.0 pounds NaCN per ton with CaO for protective alkalinity at a pH of 10.5. The leach solution

was sprinkled over the heap, and the effluent was recycled to produce an enriched pregnant solution for subsequent Ag_2S precipitation. The pregnant solution was agitated for 1 hour with 0.06 pound of 60 percent technical-grade Na_2S per troy ounce of silver (≈145 percent of stoichiometric) and 0.5 pound of CaO per ton of solution. The Ag_2S was separated from solution by filtration. Three Na_2S precipitations were done on a total of 3.4 tons of pregnant solution averaging 145.3 ppm silver and 2.4 ppm gold. The resulting barren-solution assay averaged 3.1 ppm silver and 2.4 ppm gold. This represented a 98-percent recovery of the silver in the cyanide solution. Sodium cyanide equivalent to 0.3 pound per ton of solution was regenerated according to the equation (slide 5),

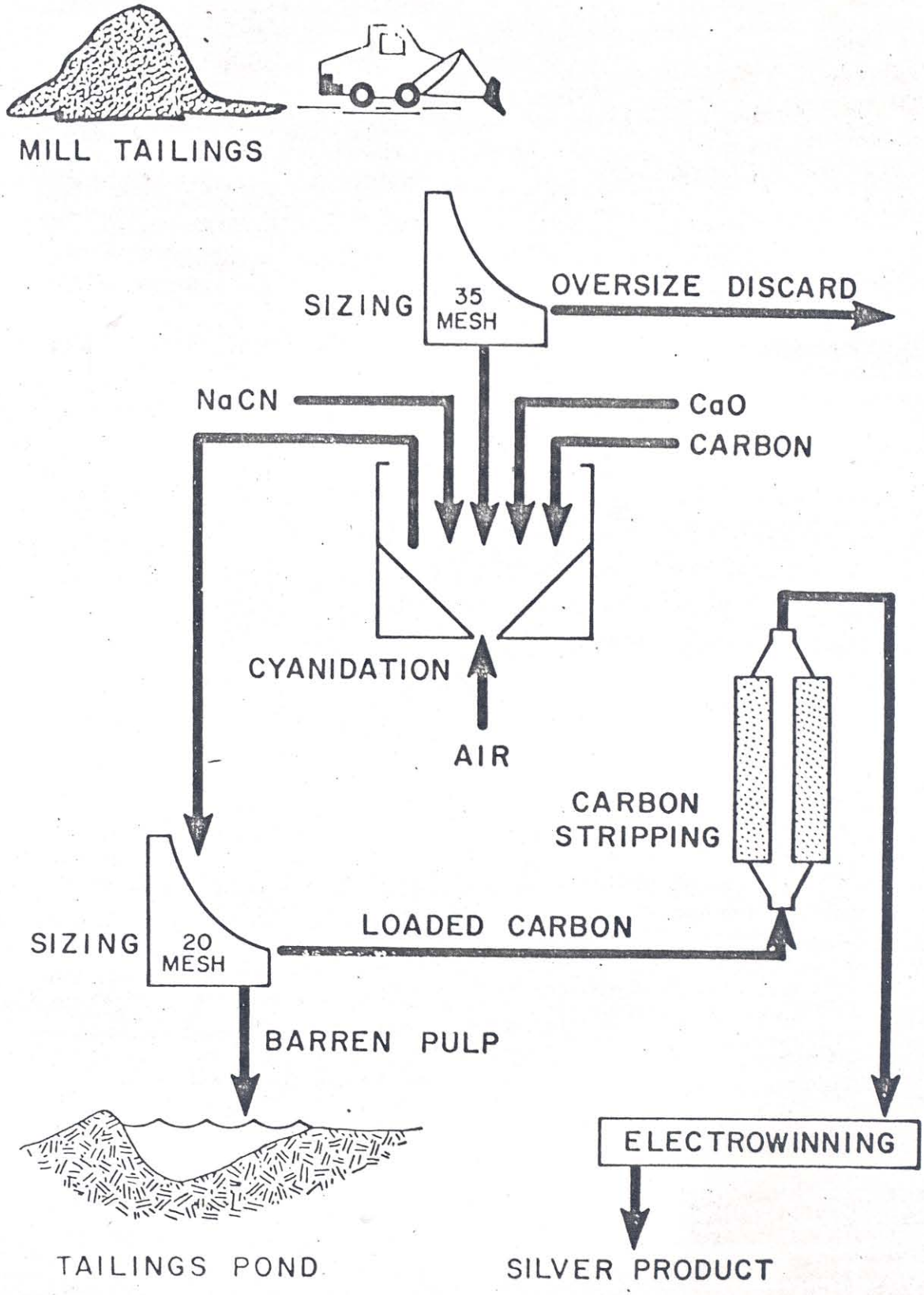
$$Na_2S + 2NaAg(CN)_2 \rightarrow Ag_2S + 4NaCN. \qquad (1)$$

The filtrate from the Ag_2S precipitation was pumped through four columns of carbon, each holding 5 pounds of 6x16-mesh, activated carbon (manufactured from coconut shells), at a rate of 0.5 gallon per minute. The average barren effluent from the carbon columns assayed <0.1 ppm silver and <0.2 ppm gold, which is equivalent to <0.003 ounce silver per ton and <0.006 ounce gold per ton. The Na_2S precipitation recovered 14.1 ounces silver and the carbon columns, 0.3 ounce silver, for a total of 14.4 ounces silver recovered. The carbon columns collected 0.24 ounce gold.

Carbon-In-Pulp Cyanidation

The use of carbon-in-pulp technique for recovery of silver and gold from ore pulps that contain substantial amounts of slimes is dictated by the settling rate of these materials and the resultant difficulties encountered in operating a countercurrent decantation system to separate liquids and solids.

Application of the carbon-in-pulp cyanidation process as used in the gold industry (3-5) was investigated for silver recovery from the slimy pulps encountered in reworking old mill tailings. The process flow sequence is shown in slide 6. Single-stage carbon cyanidation is shown in this case -- the concept employed is that the deposit is too small to justify large capital expenditure. Multi-stage countercurrent carbon cyanidation would be employed on larger deposits to obtain higher carbon loading or, alternatively, passage of slimes through an expanded bed of carbon may be employed to obtain higher carbon loading. As shown in slide 6, the tailings are retreated in leaching vessels such as tanks or gunited pits carved out of a hillside, using compressed air for agitation and


MILL TAILINGS
SIZING
35 MESH
OVERSIZE DISCARD
NaCN
CaO
CARBON
CYANIDATION
AIR
CARBON STRIPPING
SIZING
20 MESH
LOADED CARBON
BARREN PULP
ELECTROWINNING
TAILINGS POND
SILVER PRODUCT

SLIDE 6

aeration. After carbon cyanidation is complete, the loaded carbon is removed from the barren pulp by screening and sent to desorption for silver-gold recovery. The barren carbon is recycled to treat additional cyanide pulp.

Carbon cyanidation experiments were conducted on a sample of amalgamation tailings from the Austin Mining District in Nevada. The sample assayed 4.1 ounces silver per ton, 0.1 ounce gold per ton, and 80 ppm mercury. The silver distribution in the tailings was determined by wet screen analysis. Results are shown in slide 7.

SLIDE 7. - Screen analysis of the mill tailings

Screen size	Weight		Silver			Gold,
	% dist.	% cum.	oz/ton	% dist.	% cum.	oz/ton
+20	23.6	23.6	0.46	2.7	2.7	0.01
-20 +35	11.6	35.2	1.3	3.6	6.3	.01
-35 +48	5.6	40.8	1.8	2.4	8.7	.01
-48 +65	4.4	45.2	2.9	3.1	11.8	.01
-65 +100	12.8	58.0	2.7	8.5	20.3	.01
-100 +200	15.1	73.1	3.4	12.3	32.6	.01
-200 +325	6.7	79.8	3.7	6.1	38.7	.01
-325	20.2	100.0	12.5	61.3	100.0	.02
Composite	100.0		4.1	100.0		.01

Cyanidation bottle tests indicate that approximately 50 percent of the total silver is recoverable by leaching the minus 35-mesh portion of the mill tailings. Results indicate that 65-percent silver extraction was obtained from the minus 325-mesh fraction, compared with about 30-percent extraction from the sand fractions.

Pilot-scale experiments on ton-lot charges were made to determine the rates of leaching the silver and of the adsorption of silver on activated carbon as well as to produce loaded carbon for subsequent study for the purpose of developing a better stripping process, which is the key to economic application of carbon-in-pulp process for silver-bearing materials. The tailings were sized wet on a 35-mesh vibrating screen to remove tramp granitic particles and decomposed cellulose fibers derived from sagebrush and sewage, which might report with the granular carbon upon removal from the barren pulp. Over half of the feed material reported as oversize. The undersize product, containing 9 ounces silver per ton, was leached for 24 hours in a cyanide solution containing 1.0 pound of NaCN per ton at a pH of 10 maintained with CaO. The pulp was agitated and maintained in suspension using a compressed-air sparger at the bottom of a conical shaped tank. The pulp density was 20 percent. After 24 hours of cyanidation,

20 pounds of 6x16-mesh activated carbon was added per ton of feed, and agitation conducted for 24 hours. Samples of pulp were taken periodically to determine the progress of the leaching and carbon adsorption of silver. The loaded carbon, which assayed 325 ounces silver and 0.6 ounce gold per ton, was recovered by passing the pulp over a 20-mesh DSM (Dutch States Mine) screen. Slide 8 shows the extraction of silver from the feed material and the adsorption of silver on the activated carbon. The concentration of silver in the cyanide solution reached 1.5 ounces per ton in 24 hours of leaching, which represents 49.5 percent extraction. The adsorption of silver by the activated carbon is very rapid initially and then levels off after 8-hour retention, leaving about 0.2 ounce silver per ton of solution.

Further laboratory studies were made to establish the loading capacity of the carbon for silver and the soluble-silver losses occurring in a two-stage countercurrent carbon-in-pulp treatment providing 6 hours contact time per stage. The carbon-loading tests were conducted on a pregnant solution containing 1.4 ounces silver per ton and <0.1 ppm gold prepared by cyaniding the sample of old tailings. A 25-gram charge of 6x16-mesh activated carbon was advanced through four successive 5-liter batches of pregnant solution. Contact time was 6 hours per batch of solution. The loading of carbon is shown in slide 9.

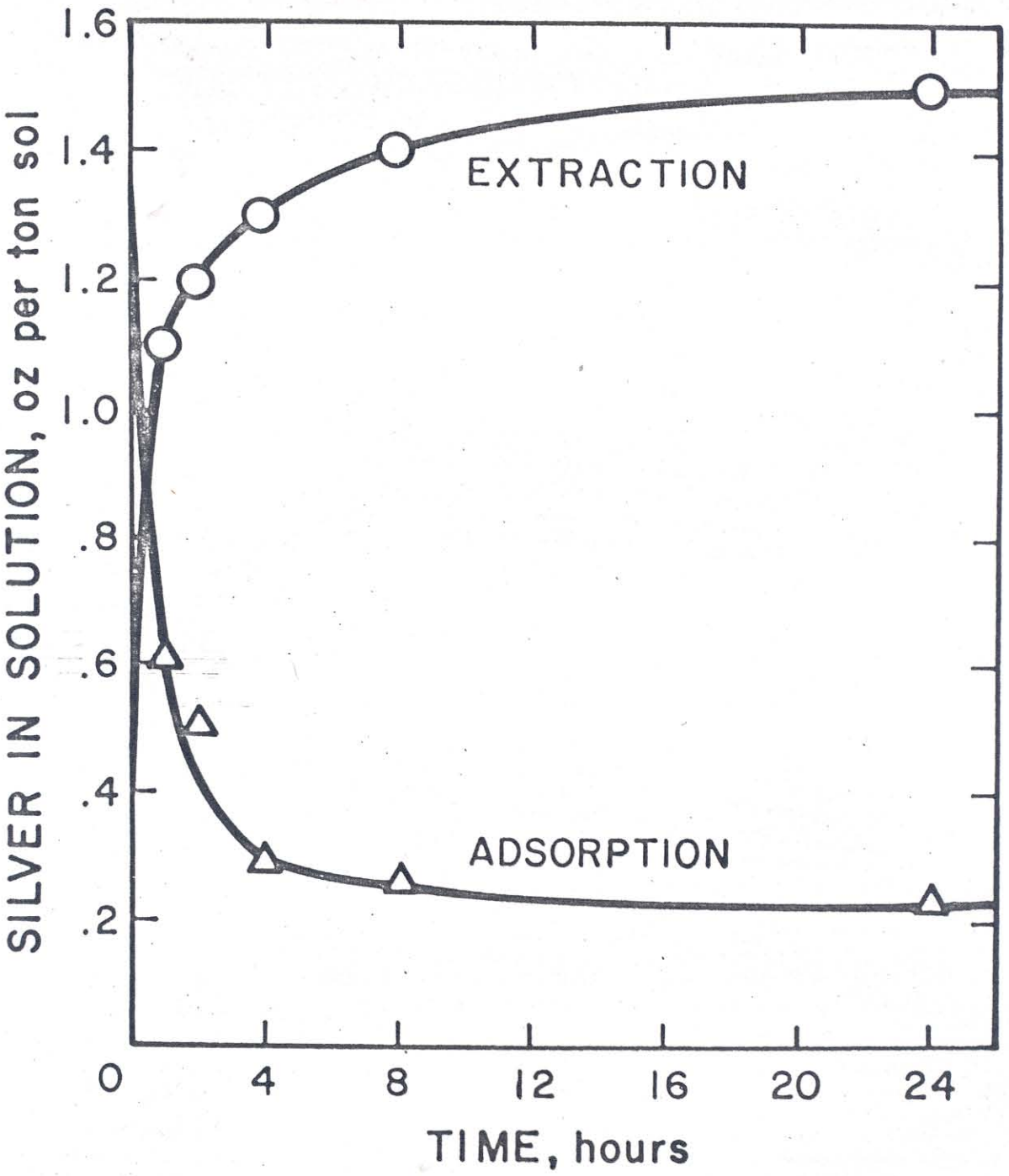
SLIDE 9. - Adsorption of silver on activated carbon

Test No.	Carbon in, oz Ag/ton	Carbon out, oz Ag/ton	Barren solution, oz Ag/ton
1	0	245	0.20
2	245	425	.53
3	425	540	.85
4	540	604	1.11

The barren solution from above test No. 1 was retested with another 25-gram charge of fresh activated carbon for 6 hours. The soluble silver loss was reduced from 0.20 to 0.009 ounce per ton of solution. This carbon was loaded to 38.1 ounces silver per ton.

Desorption of Silver From Activated Carbon

Stripping gold from granulated activated carbon is generally accomplished by elution with hot alkaline cyanide solutions (3-5) but the process can be tedious, often incomplete, and requires frequent thermal regeneration of the carbon. These factors present a potentially serious economic problem in using carbon cyanidation techniques for treating silver ores because of the low monetary value of silver compared with that of gold.


SILVER IN SOLUTION, oz per ton sol
1.6
1.4
1.2
1.0
.8
.6
.4
.2
0
4
8
12
16
20
24
TIME, hours
EXTRACTION
ADSORPTION

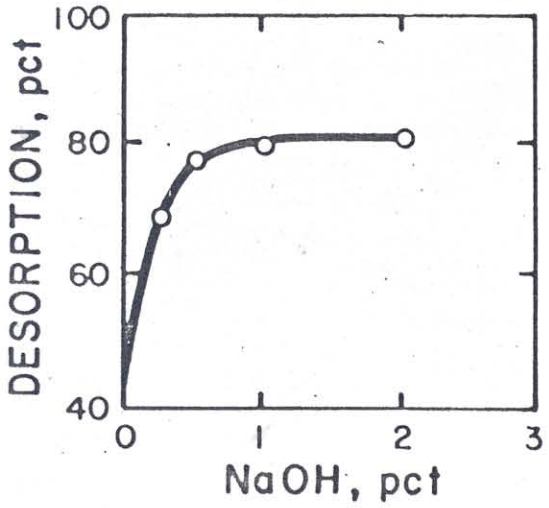
SLIDE 8

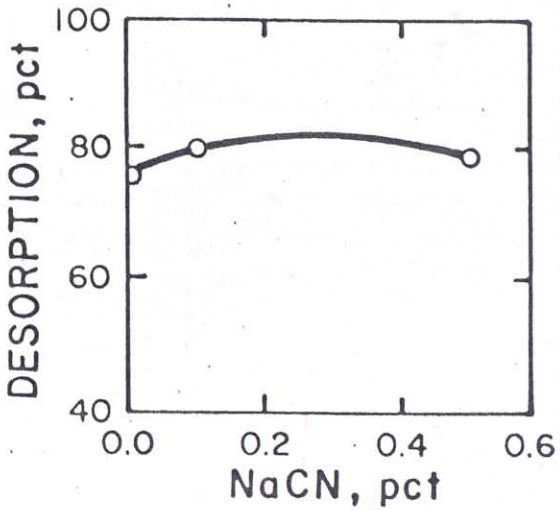
Preliminary experiments using an aqueous solution containing lower alcohols to strip silver from activated carbon indicated that the method (9,11) may be an improvement over the hot caustic-cyanide strip method, and subsequent studies were conducted to determine the efficiency of alcohol stripping agents.

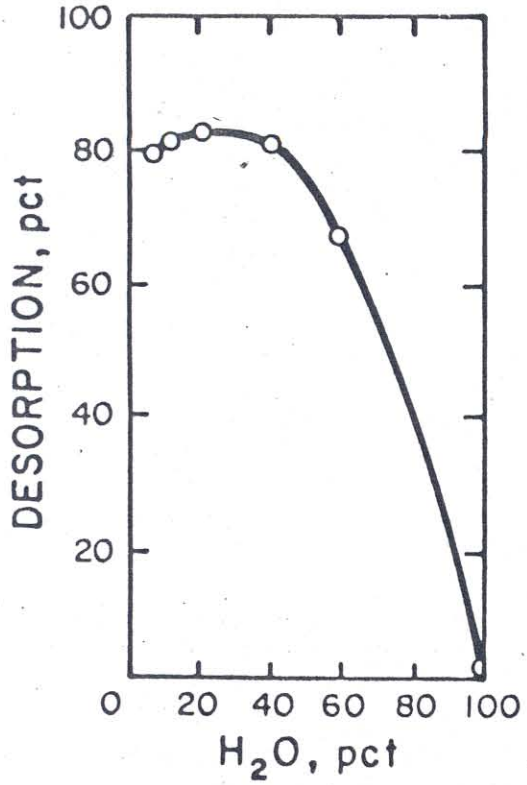
Initial studies were conducted on 6x16-mesh granular activated carbon loaded with 108 ounces silver per ton and tagged with radioactive silver-110m tracer. Silver desorption results were determined rapidly by counting the carbon before and after stripping or by counting the strip solutions. One-bed-volume (50 ml) charges of loaded carbon were agitated briefly with two-bed volumes of strip solutions and then allowed to stand for 24 hours at 30° C. Methyl, ethyl, propyl, and isopropyl alcohols containing 10 percent water, 1 percent NaOH, and 0.1 percent NaCN were evaluated. The NaOH and NaCN were dissolved in water before adding the aqueous solution to the alcohols. The amounts of silver desorbed with ethanol, propanol, isopropanol, and methanol were, in decreasing order, 88, 84, 83, and 67 percent, respectively. Ethanol was selected for further research.

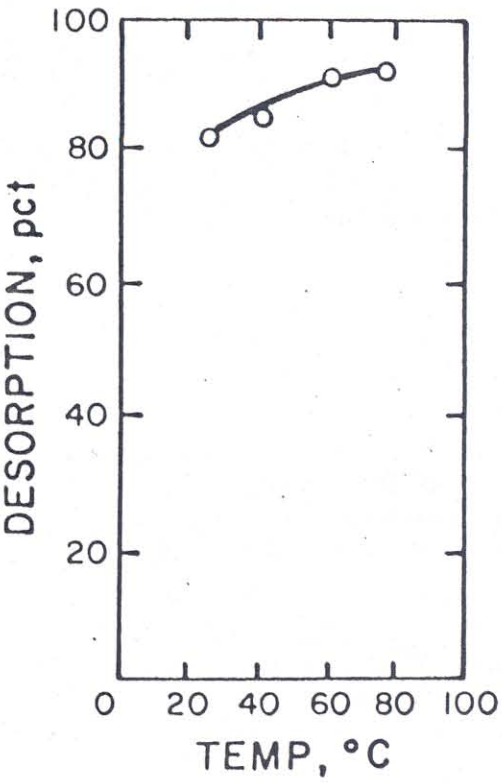
The effect of NaOH concentration between 0 and 2 percent, of NaCN between 0 and 0.5 percent, of H_2O between 5 and 100 percent, and of temperature between 25° and 75° C on the elution of silver loaded carbon with ethanol was investigated. These tests, except the one conducted at boiling temperature, were conducted by continuously recirculating five bed volumes of strip solutions upward through the fluidized bed of loaded carbon for 6 hours. The experiment at boiling temperature was conducted in a simple reflux condenser setup, with the carbon in the boiling strip solution. No attempt was made to remove the silver values from the eluents during these tests. The influence of NaOH concentration on silver desorption is shown in slide 10. The data show that optimum NaOH concentration is about 1 percent. Both cyanide concentration and temperature play only minor roles in desorbing silver from loaded carbon as shown in slide 10. The strip efficiency is very dependent on the water content of the ethanol strip solution, as shown in slide 10. The ethanol system can tolerate up to 40 percent water. It is concluded that near-optimum elution conditions for the ethanol system are 1 percent NaOH, 0.1 percent NaCN, 20 percent H_2O solution at ambient temperature.

Bench-scale experimentation is in progress to develop economic methods for recovering the silver from the pregnant ethanol strip solutions. One promising technique is electrowinning. The process sequence consists of circulating the NaOH-NaCN-ethanol solution upward through


DESORPTION, pct
100
80
60
40
0
1
2
3
NaOH, pct


DESORPTION, pct
100
80
60
40
0.0
0.2
0.4
0.6
NaCN, pct


DESORPTION, pct
100
80
60
40
20
0
20
40
60
80
100
H2O, pct

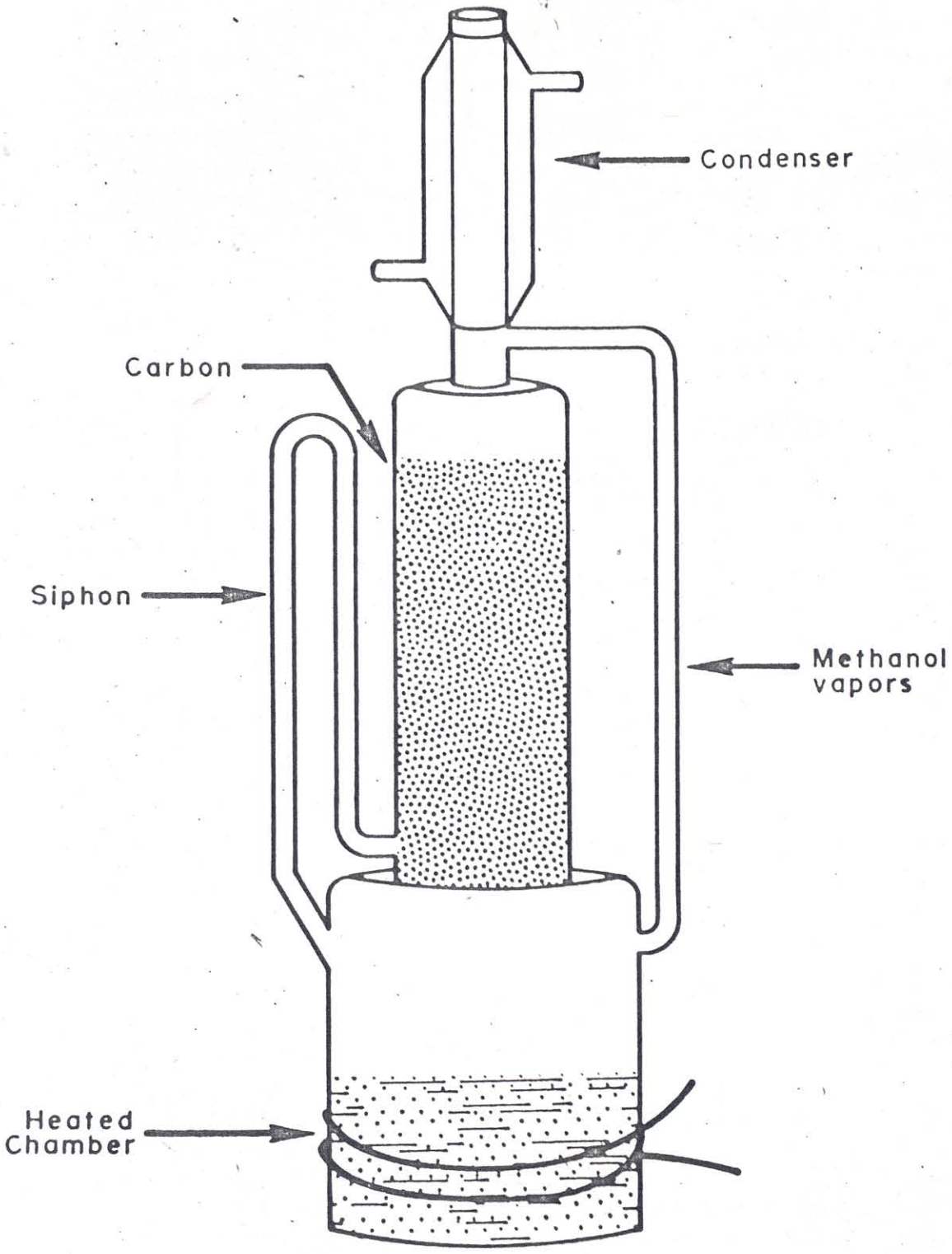

DESORPTION, pct
100
80
60
40
20
0
20
40
60
80
100
TEMP, °C

SLIDE 10

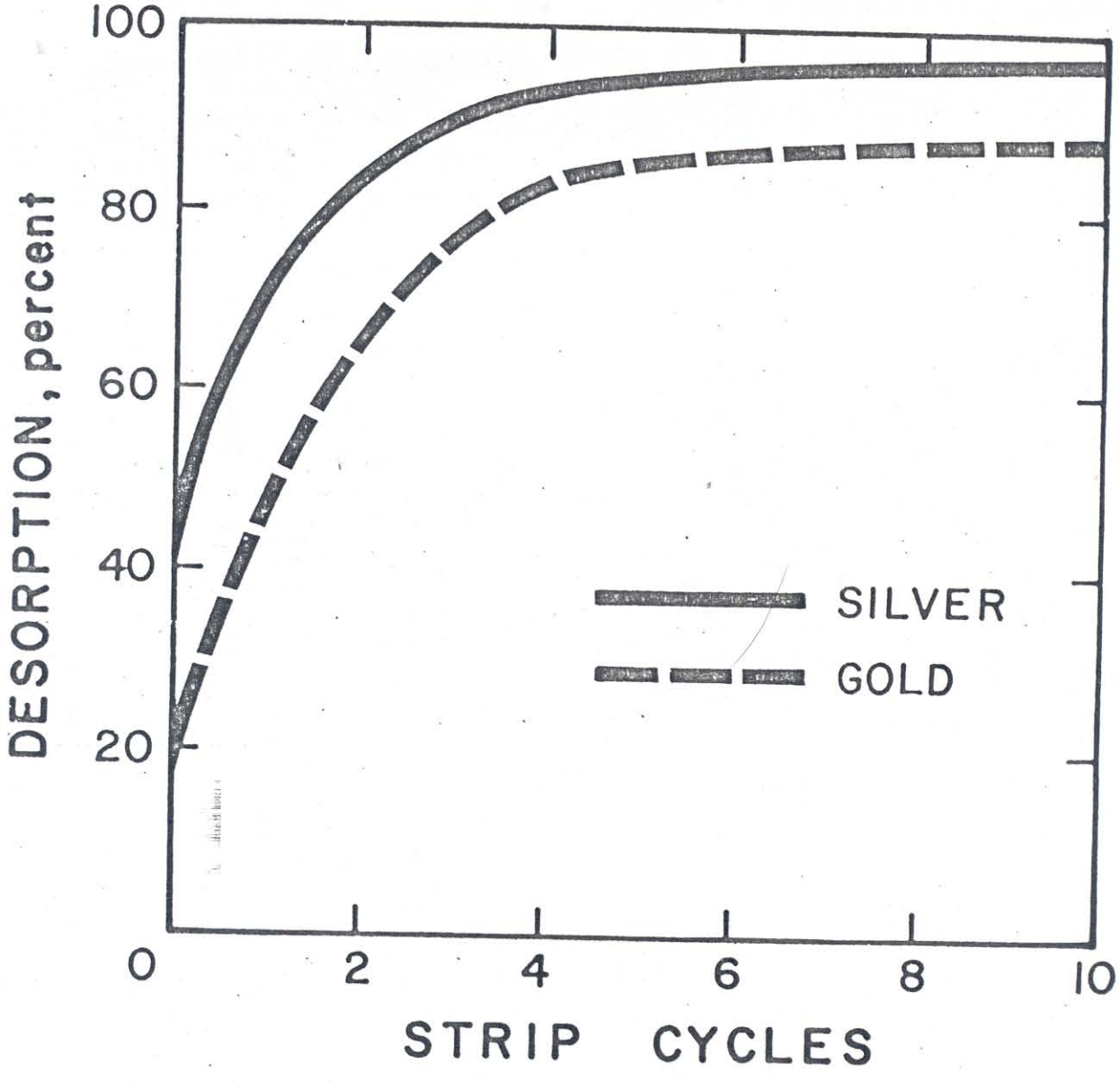
a fluidized bed of loaded carbon, electrowinning the silver from the pregnant effluent, and recycling the spent electrolyte. The simultaneous stripping and electrowinning is conducted at ambient temperature and pressure. The electrolytic cell consisted of two circular stainless steel electrodes. The anode, the smaller diameter electrode, was centered inside the circular cathode. The cell was operated at 3 volts and 0.15 ampere. A 6-hour treatment stripped 98 percent of the silver from activated carbon loaded with 300 ounces silver per ton, and electrowon 99 percent of the silver from the strip solution.

A portion of the ethanol is retained on the carbon during the silver desorption step and for process economy it is essential to recover the alcohol as well as produce a recyclable carbon. Experimental results indicated that the residual ethanol retained on the carbon lowers the loading capacity of the carbon for silver to about 75 percent of that for virgin carbon. Adsorption of contaminating ions, colloids, and organics, which are not completely removed during the elution, also results in a loss of silver adsorption activity. Investigations are underway to develop a technique for reducing these losses in adsorption efficiency. Promising preliminary results indicate that complete removal of ethanol from the desorbed activated carbon followed by a 2-percent HNO_3 wash fully restores the adsorption capability of the carbon for silver from pregnant cyanide mill solutions. Present concept is to remove and reclaim the ethanol from desorbed carbon by either steam distillation or retorting at ~100° C and then reactivating the carbon with a dilute mineral acid wash.

A modification of the alcohol elution process was also investigated utilizing methanol, because it does not form an azeotrope with water, and a reflux condenser-extractor (soxhlet) unit to strip the carbon. The operating procedure in using methanol differed from that employed with ethanol in that the loaded carbon was preconditioned for 16 hours in a 1.0-percent NaOH and 0.1-percent NaCN solution containing 3 parts of methanol and 1 part water. The carbon-preconditioning solution mixture was then transferred to the refluxing unit shown in slide 11. The methanol is distilled from the preconditioning solution, condensed, and the hot condensate passes onto the carbon. When the carbon bed is covered, the methanol automatically syphons off into the heating chamber. Thus, the silver is stripped from the preconditioned carbon with pure methanol at 50° to 60° C. One bed volume of methanol is used repeatedly through the required number of strip cycles of 15 minutes each. Results of desorbing activated carbon carrying 325 ounces silver per ton and 0.5 ounce gold per ton are plotted in slide 12. More



SLIDE II


100
80
60
40
20
0
2
4
6
8
10
DESORPTION, percent
STRIP CYCLES
SILVER
GOLD

SLIDE 12

than 40 percent of the total silver was desorbed in the preconditioning step. About 95 percent of the total silver is desorbed with five rinsing cycles of methanol, and 99 percent with 10 cycles of treatment. The methanol is reclaimed by distillation and the silver is recovered from the enriched alkaline phase by conventional techniques, such as electrowinning, and precipitation with Na_2S or zinc.

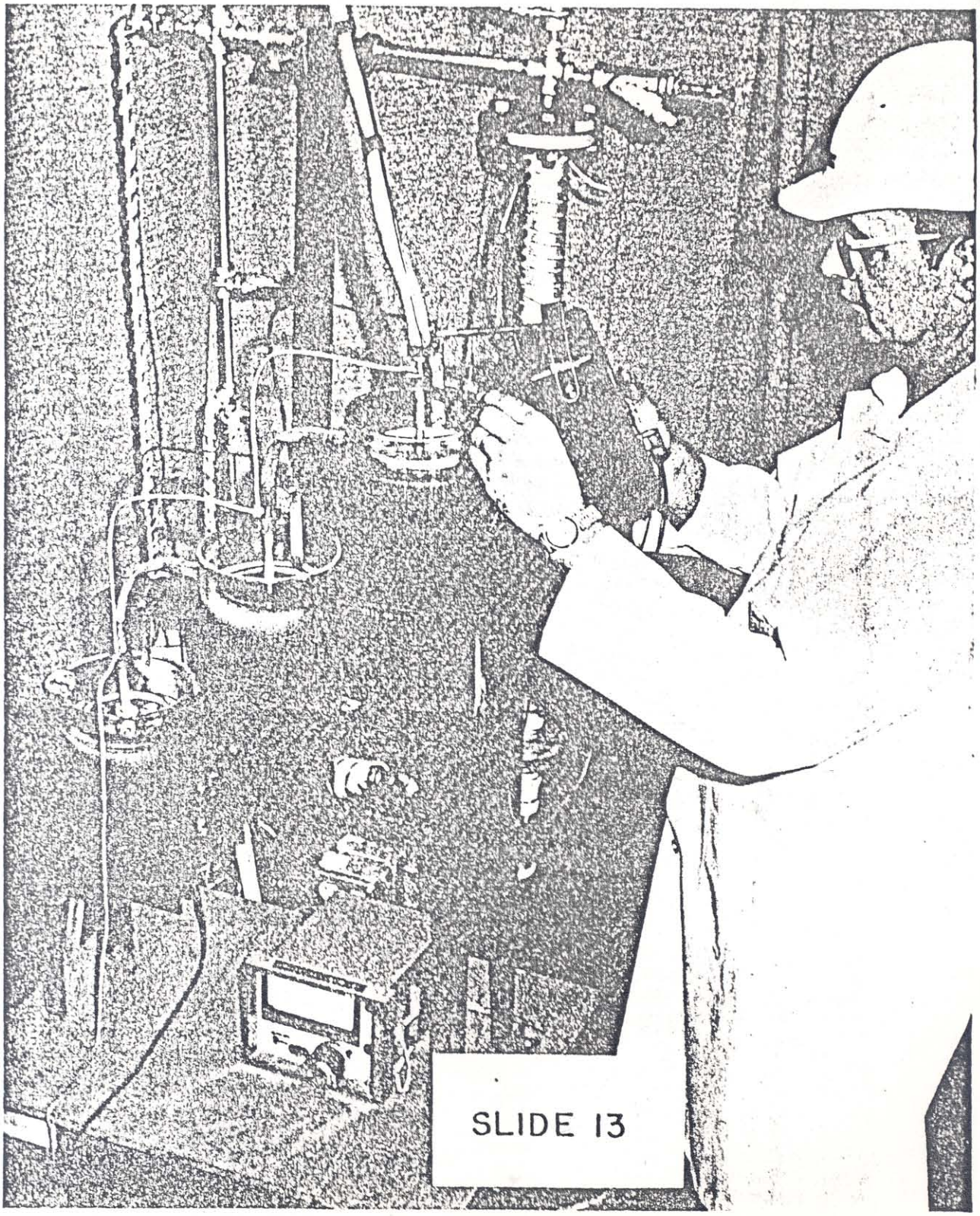
Studies are in progress to further quantify reagent requirements and operating conditions. These are being conducted in the apparatus shown in slide 13. This stripping-electrowinning unit is set up so that it can operate under a variety of temperature and pressure conditions.

Recovery of Silver From a Refractory Ore

Methods for improving the extraction of silver and gold from a variety of refractory ores that are not amenable to conventional cyanidation are also being investigated. A sample of refractory ore from the Candelaria Mining District in West Central Nevada is currently being studied. This district reportedly contains several million tons of ore containing from 1 to 6 ounces silver and 0.01 ounce gold per ton. The refractory nature of the ore is attributed to the wide and varying occurrences of the silver-bearing minerals, of which some are refractory to cyanidation. Detailed microscopy and electron microprobe examinations revealed that silver occurred in several distinct phases. The host minerals included partially oxidized iron-sulfides, antimony-iron oxides, manganese-lead oxides, and antimony-rich sulfides (10). Analyses of a Candelaria ore sample that assayed 5.5 ounces silver per ton and 0.01 ounce gold per ton are shown in slide 14.

Previous studies showed that a nominal 60 percent of the silver was extracted by fine grinding with cyanidation. By altering the iron and manganese oxide minerals with SO_2 in a 20-percent sodium chloride leach solution, the silver extractions were increased to 85 percent (10). Reagent consumption was 150 to 200 pounds SO_2 per ton of ore. Substitution of H_2SO_4 for SO_2 gave similar silver recovery. Even a drastic treatment, such as digestion of minus 325 mesh feed in 20 percent NaCl solution with excess SO_2 at 80° C under 50 psi pressure, did not improve the silver extraction.

Currently, a process sequence involving salt roasting the ore, followed by cyanidation of the chloridized calcine is being investigated. Initial roasting tests were conducted on 3/8-inch, 10-mesh, and 100-mesh feed mixed with 2, 5, and 7.5 weight-percent NaCl at temperatures



SLIDE 13

ranging from 500° to 800° C for 1 hour. Optimum roasting temperature is 600° to 700° C. Best recovery was attained by salt roasting 100-mesh feed with 5 weight-percent NaCl at 600° to 700° C and cyaniding for 24 hours. Recoveries were 88 percent of the total silver and 50 percent of the total gold. The cyanide leach residues assayed 0.8 ounce silver and less than 0.005 ounce gold per ton.

Preliminary results indicate that high silver recovery can be achieved by salt roasting finely ground ore that has been pelletized followed by a percolation cyanide leach of the calcined pellets. Further studies are in progress to determine the feasibility of heap or vat leaching pelletized calcines. Present concept is to recover the silver and gold values from leach effluents by the Ag_2S precipitation-carbon column gold adsorption technique described earlier.

SLIDE 14. - Semiquantitative spectrographic and chemical analyses of the Candelaria ore

Element	Concentration, pct
Aluminum	major
Iron	major
Silicon	major
Chromium	0.03
Copper	.06
Magnesium	1.0
Manganese	.6
Nickel	.01
Lead	1.0
Antimony	.3
Tin	.02
Zinc	.5
Calcium	4.0
Arsenic	.4
Sulfide sulfur	.05
Total sulfur	.3

CONCLUSIONS

Heap leaching techniques are applicable to certain low-grade silver ores. Silver-gold recovery from heap leach pregnant solutions is readily accomplished by silver precipitation as Ag_2S with Na_2S, followed by adsorption of gold values on activated carbon.

Slimy silver-bearing material may be treated for silver recovery by carbon-in-pulp techniques. A carbon stripping sequence employing water-alcohol eluants was shown to be

an effective means of recovering silver from loaded carbon. Carbon is regenerated for reuse by removing residual alcohol by distillation with steam or retorting and washing the carbon with dilute mineral acid.

Refractory silver ores from the Candelaria Mining District were shown to respond to a chloridizing roast, followed by cyanidation. These procedures affect 88-percent silver recovery as opposed to about 60 percent by direct cyanidation.

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Saunders Reading file

summa

Internal Communication

Date: November 11, 1975

To: Bob Baker

From: Fred Saunders

Subject: White Caps Tailings R 26

On October 24, 1975, I went to the White Caps tailings to take 4 samples to be assayed and sent off for spectrographic analysis. These samples were labeled "26 Tailings Test 1-4".

26-Tailing Test #1-75: Taken on upper dump and was a channel sample from 2'-5'.

26-Tailing Test #2-75: Taken on next lower dump and was a channel sample from 2'-5'.

26 Tailing Test #3-75: Taken on the lowest of three main dumps and was a channel sample from 2'-5'.

26 Tailing Test #4-75: Taken in gulch ¼ mile below on small tailing pond 2' channel from surface.

These samples were assayed by Summa's lab, then I sent them to Skyline Labs to be run for spectrographic analysis.

On November 10, 1975, Wally Boundy took a backhoe to the tailings and sampled the tailings to the east and the upper dump again. This was done to get a better sample at depth. His samples were called White Caps #1 and #2.

Sample #1 on the main upper dump was a channel sample from 2'-9'. Sample #2 was taken on tailings pile to the east and was a channel sample from 2'-9'. Both sample trenches were cut in the top center of the dumps.

Fred Saunders

Fred Saunders

cc: Walt
R 26 Geology

Copy
Fred. S.
Ton. Office

Location?
Depth, etc-?

SUMMA CORPORATION
MINING DIVISION – TONOPAH, NEV.

LABORATORY REPORT

DATE 11-13- 1975

DESCRIPTION	GROUP NO.	ozs. Au per ton	ozs. Ag per ton	Cu %	Pb %	Zn %					
Nate Blake Tailings											
Cut #1 Spl #1 1-2'		.100	Nil								
Cut #1 #2 2'-4'		.162	Nil								
Cut #2 #3 1-2'		.336	Nil								
#2 #4 2-4'		.052	Nil								

SHEET NO. _____ OF _____

Bill Robertson
ASSAYER

Mizpah Hotel

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LAB
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SUMMA CORPORATION
MINING DIVISION – TONOPAH, NEV.

LABORATORY REPORT

DATE 11-13- 197 5

DESCRIPTION	GROUP NO.	ozs. Au per ton	ozs. Ag per ton	Cu %	Pb %	Zn %					
TEST #1A Solution (White Caps Upper 11-10-75)		.2336	.0917								
Manhattan											

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Bill Roberson
ASSAYER

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Fordham
Location for qr

SUMMA CORPORATION
MINING DIVISION – TONOPAH, NEV.

LABORATORY REPORT

DATE 11-13- 197 5

DESCRIPTION	GROUP NO.	ozs. Au per ton	ozs. Ag per ton	Cu %	Pb %	Zn %					
Test #2A Soln		.0624	Nil								
White Caps #2 East 11-10-75											

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Bill Robertson
ASSAYER

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For whom?
Spec. location? / Orp?

SUMMA CORPORATION
MINING DIVISION – TONOPAH, NEV.

LABORATORY REPORT

DATE 11-13- 197 5

DESCRIPTION	GROUP NO.	ozs. Au per ton	ozs. Ag per ton	Cu %	Pb %	Zn %					
TEST #2 SOLN		.0530	NIL								
White Caps #2 EAST 11-10-75											

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Test By whom?
Spec. location?

SUMMA CORPORATION
MINING DIVISION – TONOPAH, NEV.

LABORATORY REPORT

DATE 11-13- 1975

DESCRIPTION	GROUP NO.	ozs. Au per ton	ozs. Ag per ton	Cu %	Pb %	Zn %					
TEST #1 SOIN (White Caps Upper 11-10-75)		.280	.0444								

SHEET NO. _____ OF _____

Bill Robertson
ASSAYER

Fred

-4-

SUMMA CORPORATION
MINING DIVISION – TONOPAH, NEV.

LABORATORY REPORT

DATE _____ 197 ____

DESCRIPTION	GROUP NO.	ozs. Au per ton	ozs. Ag per ton	Cu %	Pb %	Zn %					
TEST # 1 Residue		.060									
# 1		.060									
TEST # 1A		.070									
# 1A		.070									
TEST # 2		.100									
# 2		.096									
TEST # 2A		.086									
# 2A											
SCREEN ANALYSIS											
WHITECAPS UPPER +200	½ AT										
WHITECAPS upper +200	½ AT										
WHITECAPS upper -200											
" " " -200											

} which applies to which?

SHEET NO. _____ OF _____

_____
ASSAYER

Copy
B. Robertson
Bob Baker
Office

SUMMA CORPORATION
MINING DIVISION – TONOPAH, NEV.

LABORATORY REPORT

DATE 11-12- 197 5

DESCRIPTION	GROUP NO.	ozs. Au per ton	ozs. Ag per ton	Cu %	Pb %	Zn %					
WHITE CAPS #1 UPPER 11-10-75	26	.080									
" " " "		.078									
WHITE CAPS #2 11-10-75 EAST		.100									
" " " "		.096									

SHEET NO. _____ OF _____

Bill Robertson
ASSAYER

SUMMA CORPORATION

METALLURGICAL TEST DATA SHEET

TEST NO. Test # 4 DATE: 7-7-75

Objective:

NaOH Leach of W.T.B #1 Tailings

Product	Weight or ml.	% Weight	Assay Oz./Ton Au	Assay Oz./Ton Ag	Content Au	Content Ag	Distribution Au	Distribution Ag
Soln	2950		0.004		11.800		17.00	
Residue	600		0.096		57.600		83.00	
Head	600		0.1156		69.400		100.00	
Head assay	-		0.0900					
At 24 hr	Cy = 1.9 #/T	Cyanide Consumption =		0.67 #/ton of ore				
	CaO = 0.3 #/t	NaOH consumption =		8.87 #/ton	Calc as CaO			

Procedure: Start 9:00 A.M. -

600 grams ore
2 Liters H_2O
3 g NaOH
3 g Cy
*2 Hrs pH -

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Conclusion: No evidence of soluble sulfides in preg solution

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11-21-75

11.6
0.96
0.02

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METALLURGICAL TEST DATA SHEET

TEST NO. WC- Screen analysis DATE: 11/14/75

Objective: Screen Analysis of White Caps Upper 11/10/75

Product	Weight or ml.	% Weight	Assay Oz./Ton Au	Assay Oz./Ton Ag	Content Au	Content Ag	Distribution Au	Distribution Ag
+ 200	72.1		0.032		2.301		9.15	
- 200	227.9		0.077		17.528		69.72	
Solution	29.50		0.0018		.5.310		21.13	
Heads (calc)	300		0.084		25.139		100.00	
Head assay			0.079					

Procedure: Wet and dry screen sample on 200 mesh screen.
Save all solutions for gold assay.

Conclusion: This screen analysis seems to indicate that there would be little advantage to be gained by regrinding

The indicated soluble gold in the screening water is questioned and could be a salt. Cyanide extractions do not indicate the such gold solubility is possible

cpy

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METALLURGICAL TEST DATA SHEET

TEST NO. 1 DATE: 11-11-75

Objective: To determine Gold extraction from Tailings Sample

Sample Description — White Cajos Upper 11-10-75 2'-5'

Product	Weight or ml.	% Weight	Assay Oz./Ton Au	Assay Oz./Ton Ag	Content Au	Content Ag	Distribution Au	Distribution Ag
Heads	300 grams		0.0790		23.700		100 %	
Residue	300 grams		0.0600		18.000		75.94 %	
Extraction					5.700		24.06% extraction	
Fire Assay Heads			0.0790					

24 Hour Bottle Rolls

Procedure: 300 grams Ore.
100 mls 1% NACN
100 mls 1% NAOH
500 mls. H_2O

Conclusion: No Settling or filtering Problems

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METALLURGICAL TEST DATA SHEET

TEST NO. 1A

DATE: 11-11-75

Objective: To determine Gold extraction in Sample — White Caps Upper 11-10-75 .2-5'

Product	Weight or ml.	% Weight	Assay Oz./Ton Au	Assay Oz./Ton Ag	Content Au	Content Ag	Distribution Au	Distribution Ag
Heads	300 gr.		0.0790		23.700		100 %	
Residue	300 gr.		0.0700		21,000		88.61 %	
Extraction					2.700		11.39 %	Extraction
Fire Assay Heads			0.0790					

24 Hour Bottle Rolls

Procedure:

300 gr. Ore
100 mls 10% NACN
100 mls 10% NAOH
500 mls H_2O

Conclusion: No Settling or Filtering Problems

Copy - Joseph
Fee
11-21-75

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METALLURGICAL TEST DATA SHEET

TEST NO. 3 DATE: 11-11-75

Objective: To determine Gold extraction from Tailings Sample White Caps #2 East 11-10-75

Product	Weight or ml.	% Weight	Assay Oz./Ton Au	Assay Oz./Ton Ag	Content Au	Content Ag	Distribution Au	Distribution Ag
Heads	300 gr.		0.0980		29.400		100%	
Residue	300 gr.		0.0980		29.400		100%	
Extraction					0.0000		No Extraction	
Fire Assay Heads			0.0980					

24 Hour Bottle Rolls

Procedure: 300 gr. Ore
100 mls 1% NACN
100 mls 1% NAOH
500 ml H_2O

Conclusion: No Settling or Filtering Problems

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Lab
11-21-7
2

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METALLURGICAL TEST DATA SHEET

TEST NO. 2A DATE: 11-11-75

Objective: To determine Gold extraction from Tailings Sample White Caps #2 EAST 11-10-75

Product	Weight or ml.	% Weight	Assay Oz./Ton Au	Assay Oz./Ton Ag	Content Au	Content Ag	Distribution Au	Distribution Ag
Heads	300 gr.		0.0980		29.400		100 %	
Residue	300 gr.		0.0890		26.700		91.83 %	
Extraction					2.700		8.17 %	Extraction
Fire Assay Heads			0.0980					

Procedure: Bottle Rolls 24 Hours
300 gr. Ore
100 mls 10% NaCN
100 mls 10 % NaOH
500 mls H_2O

Conclusion: No Settling or filtering Problems

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3:30 P.M.

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METALLURGICAL TEST DATA SHEET

TEST NO. 1

DATE: 11-11-75

Objective: To determine Gold extraction from Tailings Sample

Sample description: White Caps Upper 11-10-75 2'-5'

Product	Weight or ml.	% Weight	Assay Oz./Ton Au	Assay Oz./Ton Ag	Content Au	Content Ag	Distribution Au	Distribution Ag
Residue	300 GRAMS		0.060		18.000			
Solution	1000		0.280		280.000			
Heads "Calc"			0.993		298.000			
HEADS FIRE ASSAYING			0.0790	N.I	23.7	yes		
							21.2	

Procedure: 24 Hour Bottle Rolls

300 GRAMS Ore
100 ml 1% NaCN
100 ml 1% NaOH
500 ml. H_2O
30% Solids

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Conclusion: No Settling or Filtering Problems

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METALLURGICAL TEST DATA SHEET

TEST NO. 1A

DATE:

Objective: To determine Gold extraction from Tailings Sample Description: White Caps Upper 11-10-75 2'-5'

Product	Weight or ml.	% Weight	Assay Oz./Ton Au	Assay Oz./Ton Ag	Content Au	Content Ag	Distribution Au	Distribution Ag
Heads	300 GRAMS		0.0790		23.700		100 %	
Residue	300 GRAMS		0.0700		21.000		11.39 %	EXTRACTION
					2.700		88.61%	
HEADS FIRE ASSAYING			0.0790					

Procedure:

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Conclusion:

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METALLURGICAL TEST DATA SHEET

TEST NO. 1A DATE: 11-11-75

Objective: To determine Gold extraction from Tailings Sample

Sample description: White Caps Upper 11-10-75 2-5'

Product	Weight or ml.	% Weight	Assay Oz./Ton Au	Assay Oz./Ton Ag	Content Au	Content Ag	Distribution Au	Distribution Ag
Residue	300		0.070		21.000			
Solution	1000		0.2336		233.600			
HEADS "CALC"			0.8486		254.600			
HEADS FIRE ASSAYING			0.0790	N.A.	23.7 gm			
							11.40	

24 Hour Bottle Rolls

Procedure:

300 gr. Ore
100 ml 10% NaCN
100 ml 10% NaOH
500 ml H_2O

Conclusion: No Settleing or filtering Problems

SUMMA CORPORATION

METALLURGICAL TEST DATA SHEET

TEST NO. 2 DATE: 11-11-75

Objective: To determine Gold extraction from Tailings Sample
Sample Description: White Caps #2 EAST 11-10-75

Product	Weight or ml.	% Weight	Assay Oz./Ton Au	Assay Oz./Ton Ag	Content Au	Content Ag	Distribution Au	Distribution Ag
Residue	300		0.098		29.400			
Solution	1080		0.0530		57.240			
Heads "Calc"			0.2888		86.640			
HEADS Fire Assaying			0.0480	17.1	29.4 gm			
							C.O	

24 Hour Bottle Rolls

Procedure:

300 gr. Ore
100 ml. 1% NaCN
100 ml. 1% NaOH
500 ml. H_2O

Conclusion: No Settleing or Filtering Problems

SUMMA CORPORATION

METALLURGICAL TEST DATA SHEET

TEST NO. 2A DATE: 11-11-75

Objective: To determine Gold extraction from Tailings Sample

Sample Description: White Caps #2 EAST 11-10-75

Product	Weight or ml.	% Weight	Assay Oz./Ton Au	Assay Oz./Ton Ag	Content Au	Content Ag	Distribution Au	Distribution Ag
Residue Heads	300 mls		0.0890		26.700			
~~Solution~~ Res.	1110		0.0624		69.264			
Heads "Calc"			0.3198		95.964			
HEADS FIRE ASSAYING			0.0980	N.I	29.4			
							9.9	

24 Hour Bottle Rolls

Procedure:

300 gr. Ore
100 ml 10% CY
100 ml 10% NaOH
500 ml H_2O

Conclusion: No Settleing or Filtering Problems

Susie

215 Nihell St.
Nevada City, Calif. 95959
Nov. 18, 1975

Mr. D. J. Gribbins
Mining Division
Summa Corporation
3421 Las Vegas Blvd.
Las Vegas, Nev. 89109

Dear Mr. Gribbins:

Enclosed is a report of work done over the period of November 9th to 15th, 1975. The samples tested were from the White Caps Mine. Tests were to determine amenability to heap leaching techniques.

Results indicate that the sample material which had apparently been roasted and cyanided previously are not amenable to having their gold extracted by simple heap leaching.

Furthermore, it was extremely discouraging to find that the laboratory equipment contains tramp amounts of gold that renders it next to impossible to come up with a metallurgical balance.

Enclosed, also, are assay sheets, test report sheets, receipts for expenses, as well as a statement of consultation fees.

If you have any comments to offer, upon reading the report, please do not hesitate to contact me.

Sincerely yours,

Robert E. Baker

Robert E. Baker

Copy - Geopat
Lake
Wimmer
11-21-75

BAKER WHITE CAPS REPORT

METALLURGICAL REPORT

By Robert E. Baker — November 10 - 16, 1975

Introduction

On November 7, 1975 a telephone call from Walt Simmons was received. He requested that I come to the Tonopah laboratory with the purpose of conducting some metallurgical extraction tests on new samples of White Caps tailings. On November 10 in company with Clarence Sikkenga, the trip was made to Tonopah.

Purpose

The purpose of the trip and tests was to determine the amenibility of heap leaching techniques to the extraction of gold values from the ~~Silver Peak~~ White Caps tailings.

Conclusions

The results of these tests lead to two conclusions.

1. It is doubtful if any substantial extraction of gold can be achieved from the material as represented by these two samples.

2. It is of paramount importance that the services of a qualified, full time laboratory metallurgist be secured.

Procedure

The most rapid method of determining the amenibility of a sample to cyanidation treatment is to conduct bottle roll tests of the sample.

The two samples that were submitted for these tests were designated:

1. White Caps #1 (See Fred Saunder's memo to Bob Baker 11/11/75
2. White Caps #2 (See Fred Saunder's memo to Bob Baker 11/11/75

The samples were dried, thoroughly mixed, and 300 gram aliquots were bagged for testing and head analysis. The corresponding head assay of these samples on a duplicate assay are:

White caps No. 1 -- 0.080 oz/ton Au and 0.078 oz/ton Au
White Caps No. 2 -- 0.100 oz/ton Au and 0.096 oz/ton Au

Copy - Tonopah W.C. Lab 11-21-75

These duplicate assays are considered reasonably satisfactory checks.

Testing method for cyanide extraction was as follows:

Test No	Sample No	#/ton Cy soln	#/ton NaOH soln	% Solids	Agitation time
1	1	2.85	2.85	30	24 hrs
1A	1	28.50	28.50	30	24 hrs
2	2	2.85	2.85	30	24 hrs
2A	2	28.50	28.50	30	24 hrs

After the 24 hour agitation time the pulps were filtered and given three water washes on the filter. Solutions were measured and both solution and solids were assayed. From these figures it was possible to determine the percent extraction as well as to calculate the heads, based on the metal content of the products.

The basis of calculating the head assay is:

$$\frac{\text{Wt of residue} \times \text{assay of residue} + \text{Wt of solution} \times \text{assay of solution}}{\text{Wt of heads}} = \text{Head assay}$$

The figure is termed calculated heads and should correspond, within reason, to the head assay. The results which were obtained from this series of bottle roll tests are:

Test	Product	Weight	Assay	Assay X Wt.	% Content
1.	Solution	1000	0.280	280.00	93.96
	Residue	300	0.060	18.00	6.04
	Heads	300	0.993	298.00	100.00
1A	Solution	1000	0.234	234.00	91.76
	Residue	300	0.070	21.00	8.24
	Heads	300	0.850	255.00	100.00
2	Solution	1080	0.053	57.24	66.07
	Residue	300	0.098	29.40	33.93
	Heads	300	0.289	86.64	100.00
2A	Solution	1110	0.062	68.82	72.05
	Residue	300	0.089	26.70	27.95
	Heads	300	0.318	95.52	100.00

These calculated heads compare with an assay head of

1 - 0.079 oz/ton Au.

2 - 0.098 oz/ton Au

In order to determine the distribution of the gold in various size fractions a sample of No. 1 ore was screened on a 200 mesh screen, both wet and dry.

The water used for the wet screening was saved, filtered and assayed for gold.

As is the practice, all products were retained and assayed for comparison with the assayed head sample.

The results are as follows on Sample No. 1

Product	Weight	Assay	Assay x Wt	% Content
+ 200 mesh	72.1	0.032	2.307	9.20
- 200 mesh	227.9	0.077	17.548	69.98
Wash Water	2950.0	0.0018	5.220	26.82
Heads	300.0	0.084	25.075	100.00

The fact that in this test, where no cyanide solutions were used, and a reasonable metallurgical balance was achieved, indicates that some of the equipment is salted with gold. The gold dissolves from the equipment when it is in contact with cyanide, whereas a small amount of gold is still being washed from the equipment when no cyanide solutions are used.

Based upon the aforesaid data it is concluded that no solution assays should be used during the present study. It is felt that under the circumstances calculations using only the head and residue assays will, more closely represent the actual facts.

Computations based on head and residue assays:

Test	Product	Weight	Assay	Wt. X Assay	% Content
1	Heads	300	0.079	23.70	100.00
	Residue	300	0.060	18.00	75.95
	Dissolved			5.70	24.05
1A	Heads	300	0.079	23.70	100.00
	Residue	300	0.070	21.00	88.61
	Dissolved			2.70	11.39
2	Heads	300	0.098	29.40	100.00
	Residue	300	0.098	29.40	100.00
	Dissolved			0.00	0.00
2A	Heads	300	0.098	29.40	100.00
	Residue	300	0.089	26.70	90.81
	Dissolved			2.70	9.19

Discussion --

It seems quite evident that this material is not amenable, in its present condition, to gold extraction by cyanidation. If leaching was possible the fineness of the size would preclude the possibility of maintaining a heap without some containment of the periphery.

The fact that the material is granular and has probably had most of the slimes either removed by elution or flocculated during roasting and cyanidation, renders the material fairly porous.

For the satisfaction of all concerned it is recommended that these two head samples be submitted to a qualified custom laboratory, or two, for confirmation of the head analysis.

Respectfully submitted,

Robert E. Baker

Robert E. Baker, Metallurgical Engineer

Susie

Colorado School of Mines Research Institute

P.O. BOX 112 • GOLDEN, COLORADO 80401
PHONE (303) 279-2581 CSMRI

November 14, 1975

CSMRI Project B51018

Mr. David G. Gribbin
General Manager
Land, Exploration and Mining Division
Summa Corporation
5700-B South Haven
Las Vegas, Nevada 89119

Dear Mr. Gribbin:

The intention of this letter is to record some thoughts and observations made during my visit, October 14 and 15, 1975, to your gold heap leaching operation. The initial purpose of the visit was to review the gold stripping operation, with particular emphasis on the disposition of the silver sulfide product which is a result of the silver removal step. The silver is removed in order to produce a high grade gold bullion relatively free from silver. The visit was later expanded to include the observation of the leach dumps.

Neither of the plants was operating when visited, so the thoughts and conclusions presented here are based upon conversations with people associated with the various operations, and some of the suggestions have already been discussed verbally with the persons concerned.

Although the procedures being used in the various steps of the processing are basically satisfactory, a proper assessment of the operations cannot be made because there is very little recorded operating or metallurgical information.

Silver Precipitation

The principal objections to the silver sulfide precipitate are:

cc:- B. Mallison
W. Simmons
Summit File
O. Silkenga

1. That it contains too much gold.

2. That foaming occurs during the melting operation and this not only results in a loss of material but also in furnace wear.

With regard to Item 1, present practice is to discharge pregnant solution from the carbon leach column directly into a stirred vessel which contains sufficient Na_2S to precipitate all of the silver to be stripped from carbon. The overflow from the stirred tank is filtered before being delivered to the electrolytic cell. The two sources of silver sulfide precipitate are the mix tank and the filter. Since it is stated that most of the solids are recovered from the bottom of the stirred tank, with only the very fine material being collected by the bag filter, it may be assumed that much of the fine carbon which overflows the leach column must be being caught in the stirred tank. If this is true, a settling tank, a fine screen, or a filter ahead of the mix tank should satisfactorily remove the carbon carry-over. Based upon the relatively small amount of fine carbon which is discarded by the screening operation (after the heating to expel methanol), it is expected that the particle size of any carbon being carried over is probably coarser than 200 mesh. To determine if the carbon is coarser than the silver sulfide and if it is the gold carrier, simply screen a sample of mix tank product at 200 mesh and analyze both products. The carbon can be the carrier both because gold has not been fully stripped, and because gold has been re-adsorbed from the high grade solution.

Apart from the carbon, the only other possible source of gold in the silver precipitate could be from solution collected with the precipitate. Unless the precipitate is recovered periodically during the operation, gold from this source is highly unlikely because, from analytical results, the gold in solution is very low (less than 10 ppm) when stripping is terminated.

The foaming experienced when precipitate is melted could be caused by the presence of residual sodium hydroxide, or be the result of the rapid reaction between silver sulfide and any oxidizing fluxes which are used in the melt. For the present it is suggested that a sample of the residue be washed with water and then roasted at about 650°C for perhaps one hour, depending upon the size of the sample, before melting. If this does not satisfactorily eliminate foaming, a different method for handling this precipitate can be devised.

Because of the large difference between the value of gold and that of silver, it is of importance to keep the precipitate as free of gold as possible. Consequently, the main effort should be directed toward a separation of gold and silver before the melting process. The amount of gold reporting to the precipitate should be determined regularly, and the eventual processing will be dictated by economic rather than chemical considerations.

In this regard, a metallurgical balance should always be attempted for either single batches of carbon or groups of two or three batches, depending upon production schedules. Samples of loaded carbon are presently being taken for analysis, but the weight of a lot is only an estimation. If there is no way of obtaining the weight of the material when wet, the weight can be determined later from the product through the dryer. To obtain a balance for both metals will require the dry weight of carbon, the dry weight of the silver precipitate, the weight of gold recovered, and the analyses of these materials (including that of the stripped carbon). It is anticipated that suitable balances will not always be obtainable, but the records are necessary nevertheless in order to control operations and to assess problem areas. When metal balances are poor or consistently in one direction, steps should be taken to determine the reason for the discrepancies.

Mr. Mollison has made a good start at recording some of the operating information and plans to expand to more complete records. This should be done as soon as possible.

Heap Leach Operation

The latest heap was just being put into service when the gold extraction plant was visited, and a shortage of water had temporarily suspended the carbon loading section. Here too, however, information necessary for plant assessment was not available, but Mr. Sekenga has initiated a number of additions (pregnant and barren solution storage areas, solution flow measurements, and an operating log for recording solution flow volumes and analyses) which will certainly contribute to the improvement of the overall operation. These changes will allow better control of operations, and a means for daily computing the extractions of gold and silver.

A section of the leach plant which could need attention might be that for solution stripping (or carbon loading). Here again the maintenance of suitable records would be necessary in order to relate solution flows with gold removal. For a time at least, the overflow from each tank should be sampled so that gold removal rates can be determined. Since the design of the bottom of each tank is different, it is quite possible that at certain flow rates the carbon bed of some tanks is not being utilized fully.

Leaching, General

An apparent concern is that the gold extraction is much lower than had been anticipated. Of the many possible reasons for this, perhaps the principal one could be that insufficient preliminary work had been done to determine the leaching characteristics of the ore. Another could be that the relative distribution of values in the coarse and fine material is different from what was expected.

As now constituted the heaps contain material sizes ranging from 6 inches or more to fines. To obtain, from that particle size distribution, a meaningful head analysis of the approximately 110,000 tons which are contained in the latest heap requires a comprehensive sampling program, one which perhaps would not be warranted. Sometimes it is sufficient to know that the grade of the material in general is high enough to be worthy of treatment.

Probably of more significance is a knowledge of the distribution of the gold and silver in the coarse and fine material. In any type of leaching operation, the dissolution rate is a function of mineral or metal particle size and degree of liberation. Consequently, the first values to be extracted will be those which are most accessible and which have the smallest particle size. Without a reasonable concept of the proportion of values which are readily extractable, the recovery rate and extraction efficiency cannot be determined. All that will be known is that a given amount of metal (gold) was extracted from a weight of material in a certain time.

From simply a visible assessment of the heaps, and the assumption that the values are slightly higher in the fines than in the coarse, a rough estimate of the extraction to be expected over a reasonable leach time would be about 50% of that which had been obtained by tests on ground material.

Ordinarily, tests are performed at a variety of particle sizes and conditions to determine the leach method which would provide the most economical commercial venture. It has been postulated that apart from the coarseness of the rock material, a part of the gold is relatively coarse and the slower leach rate for this material is partially responsible for the lower than expected extractions. Although it is too late to do anything about the heap currently in operation, it is suggested that laboratory work be done on to-be-mined material to determine if better results can be obtained by a change in treatment procedure, i.e., crush the ore to a smaller particle size.

By this time some of the contemplated changes have been made and more information than was available during the visit has now been generated. If you feel that sufficient new data have been developed to be worth reviewing, simply send us copies and we shall provide you with our impressions.

Of the samples which Bill Mollison shipped to us, part of the sample of silver sulfide which was contained in a large sample envelope was lost because the envelope opened during shipment. Nevertheless, we are making tests to determine if carbon is the principal gold source in this product, and if so means for removing it. This might not have too much meaning if you have made any changes in the carbon stripping section, but certainly will be helpful in determining the effect of these changes.

We shall keep you informed of any information that is developed here, and shall be glad to discuss with you any questions or thoughts that might have occurred to you since our last conversation.

Sincerely,

M. E. Defoe
Project Manager
Metallurgical Division

ebm

DKH Susie
WCM

Colorado School of Mines Research Institute

P.O. BOX 112 • GOLDEN, COLORADO 80401
PHONE (303) 279-2581 CSMRI

December 12, 1975

CSMRI Project B51018

RECEIVED
SUMMA CORPORATION
MINING DIVISION

DEC 18 1975

Mr. David G. Gribbin
General Manager
Land, Exploration and Mining Division
Summa Corporation
5700-B South Haven
Las Vegas, Nevada 89119

Dear Mr. Gribbin:

The purpose of this letter is to report the results of tests made with the two samples of silver sulfide precipitate which were shipped to us. One sample was of precipitate collected by the filter ahead of the electrolytic cell, and the other a grab sample from the stock drum.

The stock drum material was screened wet at 400 mesh and the two fractions were analyzed for gold and silver and were given an x-ray diffractometer scan. The weight distribution and analysis of the fractions were:

Material	Weight %	Analysis Au oz/t	Analysis Ag oz/t	% Distribution Au	% Distribution Ag
Stock Drum Sample	100.0	214(1)	2555(1)	100.0	100.0
+400M	20.1	738	387	69.1	3.1
-400M	79.9	83	3100	30.9	96.9

1/ Computed.

Apparently most of the gold and very little of the silver is in the coarser material. The removal of this fraction should considerably reduce the amount of gold presently reporting to the metal bars produced from precipitate.

The x-ray diffractometer scan showed that the +400-mesh portion contained a considerable amount of calcite and small amounts of quartz,

Copy - W. Simmons
Original - Susie

mica, and Ag_2S. Some of the material was amorphous (carbon ?), and several scan peaks could not be identified. The same mineral assortment was found in the 400-mesh fraction, but the Ag_2S content was noticeably higher.

The sample of filter product (310g dry weight) was still contained in the filter bag, so it represented the actual product from some period of operation. A separation at 270 mesh yielded the following information:

Material	Weight %	Analysis Au oz/t	Analysis Ag oz/t
Filter Bag Product	100.0		
+270M	12.9	1.64	10,100
-270M	87.1	1.06	13,100

There is very little gold in the filter product; consequently, this material should not be mixed with material recovered from the precipitation tank.

Although the product from the filter was not examined by x-ray, considerable calcite was readily evident by microscope.

We assume that some changes have now been made to the precipitation section of the stripping plant. If not, the removal of coarser material from the tank product, and combining only the fines with the filter product, should do much to reduce the amount of gold in the silver bars.

The Institute has appreciated the opportunity to have been of service to Summa Corporation, and should you have any questions about the gold operation or any other problems that might arise, please do not hesitate to call.

Sincerely,

Michael E. Defoe
Project Manager
Metallurgical Division

/cjm

SKYLINE LABS, INC.
SPECIALISTS IN EXPLORATION GEOCHEMISTRY
12090 WEST 50TH PLACE • WHEAT RIDGE, COLORADO 80033 • TEL.: (303) 424-7718

REPORT OF SPECTROGRAPHIC ANALYSIS

Job No. M-3652
November 14, 1975

Summa Corporation
Land Exploration & Mining Division
P.O. Box 1126
Tonopah, Nevada 89049

Attention: Bob Baker

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			
	26-Tailings			
Element	Test #1-76	Test #2-76	Test #3-76	Test #4-76
Fe	2%	1.5%	1.5%	2%
Ca	10%	5%	7%	10%
Mg	.2%	.15%	.2%	.2%
Ag	<1	<1	<1	<1
As	3,000	2,000	2,000	3,000
B	20	20	20	20
Ba	150	300	100	50
Be	<2	<2	<2	<2
Bi	<10	<10	<10	<10
Cd	<50	<50	<50	<50
Co	<5	<5	<5	<5
Cr	30	30	20	30
Cu	20	15	15	20
Ga	<10	<10	<10	<10
Ge	<20	<20	<20	<20
La	20	20	20	20
Mn	1,000	500	700	1,000
Mo	10	10	7	10
Nb	<20	<20	<20	<20
Ni	7	5	5	10
Pb	500	200	200	200
Sb	2,000	3,000	2,000	2,000
Sc	<10	<10	<10	<10
Sn	<10	<10	<10	<10
Sr	200	200	200	300
Ti	300	300	300	300
V	20	20	15	20
W	<50	<50	50	50
Y	20	15	15	15
Zn	200	200	200	<200
Zr	50	50	50	30

Gordon H. VanSickle
for Charles E. Thompson
Chief Chemist

Copy Baker file 3-16-76 JHM. copy Metallurgical Reports.

Nevada City, Ca
November 24, 1975

Mr. D.G.Cribbins
Mining Division
Summa Corp
3421 Las Vegas Blvd
Las Vegas, Nv. 89109

Dear Dave:
I have at hand a copy of a spectrographic analysis which was sent to me from Tonopah. It is, I assume, the material from the White Caps tailings.

Judging from the analysis there is still a substantial amount of both arsenic and antimony left in this material, even though it has been both roasted and subjected to cyanidation.

The roasting of stibnite ans well asrealgar and orpiment is a very difficult matter. The low melting point sulfides tend to dissolve some of the oxide that has already been formed to create a high melting point oxy sulfide that is not vilitile at ordinary roasting temperatures. This is known as Antimony Class or Glance. It is practically inert to chemical attack except at high concentrations.

It is highly possible that this type of compound was formed during the initial roasting of the material. If so, I know of no inexpensive way of recovering the gold from the material. It is probably locked, in solid solution, with this antimony and asenic compound.

The same thing happened with an antiony sulfide concentrate produced in Idaho. After considerable testing the material was discarded as being impossible to treat econamically.

I hope that this will help to shed some light on the problem as well as to indicate my thoughts on the matter of trying recover any more gold and silver economically from this very refractory material.

Sincerely yours,
Robert E. Baker

Robert E. Baker

c.c. Walt Simmons
Tonopah file

Manhatten - Big Four Mine Dump
Microscopic Examination of Panned Concentrate

Major Constituents

Quartz
Pyrite
Hematite
Pyrohotite
Magnetite
Arsenopyrite

Minor Constituents

Chalcedony (Chert)
Apatite
Gold (native) Untarnished and bright
Chrysocolla (very small amount)
Electrum (very minor)

Float Test #1

500 g ore
-48 mesh grind
pH 7.3 (natural)
.05 #/ton Aerofloat 208
.05 #/ton Xanthate 350
Condition 5 min.
Temp. - room
10 min float time
frother Crysilic acid .04#/ton

Head 500 g.
Conc. 29.5 g. @ 5.52 = 162.84 gm-oz.
Tail 456 g. @ .10 = 45.6 gm-oz.

208.44 = .457 oz.

HxP .457 456

$\frac{162.84}{208.44} \times 100 = 78.12\%$ recovery

Float Test #2

500 g. ore
-48 mesh grind
pH 7.4 (natural)
.05 #/ton 208
.05 #/ton 350
Cond 5 min.
Float time 10 min.
2.0#/ton Sodium Sulfide
5 min. Cond.
10 min float
Crysilic Acid .04#/ton

Float Test #2 continued

Head 500 g.
Conc. 54 g. @ 2.6 oz. = 140.4 gm-oz.
Tail 435 g. @ .08 oz. = 34.8 gm-oz.

175.2 = .358 oz/ton
489

HxP = .358 oz/ton

$\frac{140.4}{175.2}$ x 100 = 80.14 % recovery

Float Test #3

500 g ore
-100 mesh grind
pH 7.4 (neutral)
.05 #/ton 301
.05 #/ton 208
Pine oil .04 #/ton
Cond 5 min
Float Time 10 min

Head 500 g @ .32 = 160.0
Conc 46 g @ 2.88 = 132.48
Tail 450 g @ .05 = 22.50

154.98 = .312
496

HxP = .312

$\frac{132.48}{154.98}$ x 100 = 85.48 % recovery

Cyanide Test #1

Lime consumption
1000 ml water
100 g ore
2.0 g CaO @ 50% avail. = 20#/ton of ore

Titration: Final 4.00
Initial 3.00
Net 1.00 = 1#/ton solution CaO free

= 10#/ton of ore consumed

200 g ore -48mesh
600 ml of 1.5#/ton ore soln.
12.0 #/ton ore CaO
24 hr. Agitation

Preg. soln.	=	.11	oz.	@ 600	=	66.0
Tail	=	.04	oz.	@ 200g.	=	8.0
Head	=	.29	oz.	@ 200g.	=	58.0
						66.0

$\frac{58}{66}$ = 87.9 % recovery

$\frac{66}{74}$ = 89 % "

Cyanide consumption

Final 1.55
Initial 1.00

Net .55 #/ton soln.

Lime consumption

Final 12.0
Initial 9.0

Net 3.0 #/ton soln.

Partial Grinding Test #1

500 g of ore crushed to - 1/4" assay .04
25 min. grind in ball mill (dry) with very light ball load
screen at 48 mesh

							Wt.
-48 mesh	30 g	@	.51 oz.	=	15.30	= 76.3%	5.9
+48 mesh	475 g	@	.01 oz.	=	4.75	= 23.7%	94.1
					20.05	100	100

HxP = .039

Conclusions:

As can be seen from the test work the ore from the Big Four area is ~~answerable~~ amenable to both flotation and cyanidation. Recovery of close to 90% can be made by both methods. The most interesting test I ran was the crushing and partial grinding test. As can be seen from the test a dump sample assaying .04 was crushed to -1/4" and then rattled around in a ball mill with 6 or 8 balls for 25 minutes. This same type of partial grinding could be accomplished with some type of a hammer or impact mill. The sample was then screened at 48 mesh and the results were really quite astounding. I got a concentration of about 13 to 1, 76% of the gold in 6% of the material.

As you can see from the above test work the best recovery on Big Four ore would be to fine crush, screen and agitate leach the minus 48 or what ever mesh further test work shows to be the best, with carbon in pulp. The coarse could then be heap leached or jigged whatever would test out the best. Both myself and Dave Pruett highly recommend that more test work be done along these lines.

March 12, 1975 Paul M. Skinner

David L. Pruett
612 East 2nd Street
Winnemucca, Nevada
623-5679

May 9, 1975

Summa Corp. Mining Division
P. O. Box 1126
Tonopah, Nevada 89049

Attention: Mr. Dave Gribbin
Mr. Walt Simmons

Sirs:

Since alteration work never seems to be complete for any processing plant, I have a few comments on the work initiated at Tonopah and Manhattan.

Potential Problems -- Manhattan

1. The carbon loading tanks are designed for a flow rate of 20 gpm/sq.ft. or 1000 gpm. The carbon should teeter at about 500 gpm. Lower flow rates will cause poor agitation. Samples should be taken at tank center and side to assemble data for the minimum flow rate acceptable to provide good uniform loading. The dispersal plate in the bottom of the tank has a 6% void ratio (1½" holes on 6" centers) as recommended by the Bureau of Mines. Low flow rates will actuate radially, leaving the sides with less agitation.

2. The carbon on hand, mainly PBC-10/30, may be too fine for use with a slimy solution from a newly loaded pad. The flotable carbon and slime will plug the 30 mesh screens. Constant brushing to keep the screens free would be necessary. 6/16 carbon and 20 mesh screen would help the slime problem. Union Carbide pellet is not available enough to count upon. Pond settling

of slime before introduction to carbon tanks would help.

Tonopah Stripping Plant

1. The pump on the filter may be moved and used as a spare on the preheater. The internal screen may be replaced with a valve and closed when loading.

2. Complete data must be assembled to properly operate this unit.

3. The unit should not be altered until comparison data can be assembled. The time to strip, the costs, and recovery ratio, and bullion grade are probably better than the current practice at Homestake or Cortez. This is a preliminary estimate and must be proved or disproved before radical modification.

4. Stripping time may be reduced by conditioning the carbon in the storage tank with NaOH.

5. Temperature control is critical to stripping in reasonable time. Standard solutions should be made up with NaOH, NaS_2, methanol and water. The boiling and flash points observed.

6. If a decision is made to sell bullion to a broker, such as Simmons Refining, a melting-brokerage fee of 2% is taken regardless of fineness. The silver precipitation can be discontinued and dore bullion shipped direct to refiner.

7. Ideal location of a stripping plant is at the millsite, eliminating transport and handling. Extra solution not completely stripped can be returned to pads. However, the control necessary to run the unit would require an A/A unit at the same location.

8. You should tie to Sierra Pacific Power and use the diesel as standby.

   A spare pump should be installed.

   These should be Clarence's problems to figure out.

David L. Pruett

Randy Burke

Salt Lake City Metallurgy Research Center

March 7, 1974

Mr. David J. Gribben
Summa Corp.
5700-B So. Haven
Las Vegas, Nev. 89119

Dear Dave:

We completed a percolation cyanidation leaching test on a sample of gold ore submitted by Randy Burke and identified as Manhattan, Nev., pit run ore. Our calculated head assay of the material was 0.09 oz gold and 0.08 oz silver per ton, and the sample was about 35 percent plus 4 inches in size. The test procedure and detailed results are given in the attached memorandum from Harris Salisbury.

The test results indicated that about 80 percent of the gold in the calculated head was recovered in 600 hours of leaching. Reagent consumption was 1.5 lb NaCN and 3.3 lb CaO per ton of ore.

We hope this information will be helpful.

Sincerely yours,
Signed George M. Potter

George M. Potter
Research Supervisor

Enclosure

cc: Randy Burke

Salt Lake City Metallurgy Research Center

March 7, 1974

Memorandum

To: George M. Potter, Research Supervisor, Salt Lake City Metallurgy Research Center

From: Harris B. Salisbury, Project Leader

Subject: Percolation cyanidation test on ore submitted by Randy Burke, Summa Corp.

Approximately 267 lb of pit-run ore was loaded into a 14-inch diameter transite column. Lime solution was percolated downward through the bed at 16 ml/min until a pH value of 10.4 was reached. The solution was then made up to 0.1 pct NaCN and percolation continued for 600 hours, at which point the pregnant solution assayed 0.09 ppm Au and 0.05 ppm Ag, values considered too low to justify further leaching. The pregnant solution was collected in 24-hour increments then weighed and assayed before passing through an activated carbon column prior to reuse. The first carbon removed after 384 hours assayed 252 mg Au and 115 mg Ag compared to cumulative solution assays that indicated a recovery of 268 mg Au and 126 mg Ag. The final char was lost making it necessary to use the solution assays and atomic absorption residue assays for final calculations. Consumption of NaCN was 1.5 lb per ton of ore, and CaO usage was 3.3 lb.

Harris B. Salisbury

Attachment

508 — Run 73.2 — 2-25-74

Leaching time, hours		Solution volume	Assay, ppm		Metal Content, Mg		Cumulative Metal Content Mg		CUM % RECOVERED		CUM LBS/TON RECOVERED	
INCREM.	CUM	LITERS	Au	Ag	Au	Ag	Au	Ag	Au	Ag	Au	Ag
24	24	18.596	1.2	.94	22.3	17.5	22.3	17.5	.006	.004	6.6	5.0
	48	17.722	2.2	1.03	39.0	18.3	61.3	35.8	.015	.009	16.5	16.3
	72	20.768	2.0	.86	41.5	17.9	102.8	53.7	.026	.014	28.6	17.5
	96	21.569	1.45	.66	31.3	14.2	134.1	67.9	.034	.017	37.4	21.3
	120	20.992	1.1	.52	23.1	10.9	157.2	78.8	.040	.020	44.0	25.0
	144	21.330	.9	.42	19.2	9.0	176.4	87.8	.044	.022	48.4	27.5
	168	21.809	.65	.33	14.2	7.2	190.6	95.0	.048	.024	52.7	30.0
	192	22.010	.60	.26	13.2	5.7	203.8	100.7	.051	.025	56.0	31.3
	216	21.703	.55	.23	11.9	5.0	215.7	105.7	.054	.027	59.3	33.8
	240	20.939	.60	.19	12.6	4.0	228.3	109.7	.058	.028	63.7	35.0
	264	20.881	.40	.17	8.3	3.5	236.6	113.2	.060	.029	65.9	36.3
	288	21.474	.25	.14	5.4	3.0	242.0	116.2	.061	.029	67.0	36.3
	312	21.618	.45	.13	9.7	2.8	251.7	119.0	.063	.030	69.2	37.5
	336	21.187	.30	.12	6.4	2.5	258.1	121.5	.065	.031	71.4	38.8
	360	20.316	.25	.12	5.1	2.4	263.2	123.9	.066	.031	72.5	38.8
	384	22.699	.21	.11	4.8	2.5	268.0	126.4	.068	.032	74.1	40.0
	408	22.679	.12	.13	2.7	2.9	270.7	129.3	.068	.033	74.1	41.3
	432	25.747	.10	.14	2.6	3.6	273.3	132.9	.069	.034	75.8	42.5
	456	22.436	.15	.15	3.4	3.4	276.7	136.3	.070	.034	76.9	42.5
	480	22.529	.12	.10	2.7	2.3	279.4	138.6	.070	.035	76.9	43.8
	504	22.398	.12	.07	2.7	1.6	282.1	140.2	.071	.035	78.0	43.8
	528	22.603	.09	.06	2.0	1.4	284.1	141.6	.072	.036	79.1	45.0
	552	11.947	.14	.09	1.7	1.1	285.8	142.7	.072	.036	79.1	45.0
	576	21.185	.14	.06	3.0	1.3	288.8	1440	.073	.036	80.2	45.0
	600	17.906	.09	.05	1.6	.9	290.4	144.9	.073	.037	80.2	46.3
	RESIDUE								.018	.043	19.8	53.7
CALC	HEAD.		↓ tailing loss						.091	.080	100.0	100.0

Started June 25, 1974
Ended Aug. 2, 1974

TUB TEST 1

Tub Test 1 was started as an experiment to discover the ability and practicality of leaching Manhattan ore and to try and improve recovery rates and reagent effiency. And then with this knowledge to project it, for the coming forty thousand ton pile.

I started out with 1000 pounds of Reilley Pit ore averaging .088 oz/ton in gold. I started out with 30 liters of solution. Puring the experiment I continually had to add cyanide, water and caustic to keep concentrations up. The total use of cyanide for the 38 days was 405 grams, or10.6 grams per day. Expanding this to a forty thousand ton pile 1840 pounds of cyanide will be used each day. The water consumed through evaporation was 280 liters total, or 7.3 liters used per day. Expanding this to a square area of 41,600 sq. ft. including ponds, 8016 gals. will be lost through evaporation. So after looking at the relationship that 1 gram Cn per liter H2O is equal to 2 pounds CN per ton of solution it can be seen that 7.3 grams of cyanide per day was needed for the 7.3 liters of solution lost, so this would leave 4.7 grams of cyanide per day consumed by the ore. Expanding this to 40,000 tons of rock 824 pounds of cyanide will be used each day for the ore.

The ~~pm~~ P.H. of the water at the lab is seven, after running the plain water through the pile the pm was six. The amount of caustic needed for the 38 days to maintain a pm of eleven was 76

grams or 2 grams per day consumed. Expanding this to a forty thousand ton pile, 352 pounds will be used each day. Again looking at the water added, it takes .166 grams of caustic to turn one liter of water from a pm of 7 to a pm of 11; therefore since 7.3 liters were added each day 1.21 grams were used to raise the pm to 11. Therefore .79 grams were consumed to raise the pm from 6 to 7. Expanding this to a 40,000 ton pile, 136 pounds of caustic will be consumed by the ore itself. Also the consideration that lime is going into the pile must be taken into account, thus, these above figures would probably be lower because in the tub test no lime was used. Also the fact that at Manhattan the water is alittle more basic then the water at the lab, again meaning less caustic.

Looking at the graphs it can be seen that between June 30 through July 21 was the time of peak operating period. This was basically because the cyanide concentration was very close to 2 pds cn per ton solution. The graphs show that the steadier the concentration is around 2 pds per ton the better effiency there is.

As for the flow rate, looking at thegraph it can be seen that around 150 mills per minute is on the rise of the gold plot. This works out to be 3168 gals. per minute for a 40,000

ton pile. This flow might seem very rapid but because the ore is of the nature of free gold, I think that this flow rate is not altogether unrealistic.

The silver concentration is following a definite pattern as can be seen from the graph. Everytime the cyanide concentration goes up beyond 2 pds per ton the silver value also jumps. This is logical because after running concentration tests on this ore I found that around 3 pds/ton was optimum operating conditions for silver. Also the higher peaks are because the silver is there in much greater quantity then the gold.

Two dispercing agents were also tried on the test to try and shake the rock up abit. The first agent GPG I added on July 11, looking at the graph it can be seen the jump that it gave the gold concentration. But since GPG is no longer made I tried a new product, AD101 which is supposed to be similiar to GPG. On July 15 I added AD101, again the jump can be seen on the graph, especially the silver. The conclusion is that this agent works extremely well, with very little agent needed.

As for the leach ability and recovery rate of this rock, it is excellent. The ore leachs extremely well with average use of cyanide. Also the recovery rates are excellent, the average was 4.51 ppm in the head solution for the 38 days. This is 72% re-

covery in 38 days. The highest assay on the rock turned out to be .220 oz. per ton, the highest assay on the head solution was .255 oz. per ton. The little amount of difference is negligible when you think that the rock assays are random. During the 38 days time, .188 oz. of gold per ton of ore was taken out of the rock. Projecting this to a 40,000 ton pile, recovery would probably be very close because once the pile is saturated and concentrations brought up to level, there is very little difference between the tub and the pile itself. The only difference might be the temperatures, the tubs environmental temperature stayed the same day and night, whereas with a 40,000 ton pile it would be affected by constantly changing temperatures. But even the changing temperatures should only slightly affect the recovery rate, unless there was a very drastic change.

SUMMARY

1. That 405 grams total, or 10.6 grams per day of cyanide was consumed.
2. That 280 liters or 7.3 liters per day of water was lost.
3. That 76 grams or 2 grams per day of caustic was consumed.
4. That the optimum flow rate was found to be 150 ml per minute.
5. That 728 of the gold was recovered within 40 days.

6. That the silver concentration is directly proportional to cyanide concentration above or below 2 pds per ton.

7. That gold recovery is at its optimum when the cyanide is kept at a steady concentration of 2 pds per ton of solution.

Projections from these results to a 40,000 ton pile.

1. That 1840 pds of cyanide will be used each day.

2. That 8016 gallons of solution will be lost each day.

3. That 352 pds of caustic will be used each day.

4. That optimum flow rate was found to be 3168 gals. per min.

5. That 72% of the gold will be recovered within 80 days.

Cost analysis of reagent consumption per day using projected amounts.

1. 1840 pds of cyanide at 35 cents per pd.
$644.00 per day

2. 352 pds of NaOM at 25 cents per pd
$88.00 per day

3. 1000 pds of carbon at 60 cents per pd
$600.00 per day (running at 1ppm head and 1000gal/min)
Total $1,332.00 per day

This amount of reagent will collect 150 oz. of gold
150 oz. at $150.00 Cost of reagents $1,332.00
On this value of gold $22,500.00

That cost of reagent will be 11% of profit

CONCLUSIONS:

That Manhattan ore is excellent leach rock. Also because the ore has no sulfides, cyanide and caustic ore not

consumed to any ridiculous amounts. From my experience doing tub tests and column leaches the results obtained from these tests are invaluable for projecting to larger piles with excellent results and this includes flow rates. Out of this tub test 1, flow rate is the only thing that has me baffled. I am not sure whether it is because I have never dealt with flow rates of 3168 gal. per min. or that I cannot accept this because of logical reason. This is what the experiment showed so until it is disproved in the field this is the result I will go by. As for the cost of reagents they could be cut down, but then the recovery rate would also be cut considerably. The whole cost is actually based on how much water that is lost each day. There will be very little difference, or no difference in water loss between pumping 3,000 gals per min to 1000 gals per min, this is because you will still have the same surface area. It could also be looked at that more water will be lost at slower flow rates because the water has a longer time period in the atmosphere. So it can be seen in finality that these costs are not going to change in any degree unless water loss is brought down.

Paul Ricks

Met Correspondence

GEOLOGY
EXTRACTIVE SYSTEMS
MINE PLANT DESIGN

DAVID L. PRUETT
MINING GEOLOGIST
GOLDFIELD, NV 89013

May 25, 1976

Mr. William J. Robinson
Summa Mining Division
P. O. Box 1126
Tonopah, Nevada 89049

Re: Manhattan Ore Treatment

Dear Mr. Robinson:

The test work just completed confirms the original estimates and recoveries stated in the May 7 letter to you. The data shows that by:

1. Crushing to minus one inch from Mine Run.
2. Scrubbing in a ball mill.
3. Screening to minus ten mesh will upgrade initial head assay by 2.5 to 1.

85% of the total gold will be contained in 32% of material on the average ore.

The minus 10 mesh fraction may be agitated in tanks with cyanide. 88% of the gold may be recovered from solution by activated cargon. These figures are averages.

The cost quoted ($145,000) will cover a plant with 500 ton per day minimum capacity. With some machinery exchange and additions, capacity can be raised to 1,000 ton per day - cost total $210,000. This decision should be made before construction start.

The basic data for the plant concept is uniform and with no surprises.

A summary of test work and the recent word data sheets is attached.

Sincerely,

David L. Pruett

David L. Pruett

DLP:ps

cc: David K. Hamilton

GEOLOGY
EXTRACTIVE SYSTEMS
MINE PLANT DESIGN

DAVID L. PRUETT
MINING GEOLOGIST
GOLDFIELD, NV 89013

SUMMARY: Screen Analysis

Agitated Cyanide Leaching

Big Four Mine

Big Pine Pit

Group 26

USGS Bulletin #723 - Ferguson

Coarse crushed ore (3 inch) screened to (- 5/8 inch)

(- 5/8 inch) 45% of total material contains 80% of gold. Deeper ore does not upgrade as well by screening.

March 12, 1975 - Skinner

Mine Run crushed to minus 1/4 inch; tumbled in lab ball mill 25 minutes.

		Assay	Percent Material
Material screened to	-48 mesh	.51 oz/ton	5.9
	+48 mesh	.01 oz/ton	94.1

Upgrading 13/1

76% of gold in 6% material.

Scrubbing and screening upgrades this ore.

March, 1976 - Boundy

Summary - Screen analysis Big Pine Pit

Group 26

+3 inch contains good grade - gold in fractures unreached by cyanide heap leach.

Screening alone does not upgrade Manhattan ore. Blasting recommended to open fractures.

GEOLOGY
EXTRACTIVE SYSTEMS
MINE PLANT DESIGN

DAVID L. PRUETT
MINING GEOLOGIST
GOLDFIELD, NV 89013

May 3, 1976 - Hamilton

Big Pine Pit Screen #3 and #4 - Boundy

-1/4 inch material in Bottle Roll Test.

72 hours. Recovery in solution 90% and 84.75% respectively.

Gold in -1/4 inch is soluble.

May 7, 1976 - Boundy & Pruett

Big Pine Pit Screen #4 (-1 inch composite)

Placed in lab ball mill with twenty 1 inch balls

3 liters water for ten minutes

83.3% of gold in 51% of material (-1/4 inch)

16.7% of gold in 49% of material (-1 inch + 1/4 inch)

May 12 through 24, 1976 - Pruett & Boundy

6 Scrub and cyanide tests were run on Big Pine Pit Mine Run crushed to (-1 inch).

The material was placed in a lab ball mill with twenty 1 inch balls for ten minutes.

Larger charges produced better results.

86% of gold - remains in (-10 mesh) 32% of total material. Agitated cyanide tests on this -10 mesh material shows:

88.6% Recovery	6 hours
92.5% Recovery	12 hours
95 %+ Recovery	24 hours

Base at 86% of gold in 32% of material gives an overall recovery of 76%, with 6 hour leaching.

CONCLUSION:

Big Four Big Pine Pit Group 26 ores are adaptable to upgrading by crushing, scrubbing and screening. Upgraded fines can be processed by cyanide in agitation tanks with much higher recoveries than heap leach.

SHEET NO. _____ OF _____ ASSAYER

GEOLOGY
EXTRACTIVE SYSTEMS
MINE PLANT DESIGN

DAVID L. PRUETT
MINING GEOLOGIST
GOLDFIELD, NV 89013

May 3, 1976 - Hamilton

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SHEET NO. _____ OF _____

ASSAYER

Scrub Test #1

SUMMA CORPORATION
MINING DIVISION – TONOPAH, NEV.

LABORATORY REPORT

DATE 5/6/ 1976

B.P. Pit Screen Analysis #4 - Split B

DESCRIPTION	GROUP NO.	ozs. Au per ton	ozs. Ag per ton	Cu %	Pb %	Zn %					
"to -1" Heads	26	.026									
" to -1" Heads		.027		calc Head .073					11/1 upgrade		
from +½" to -1" GRAVITY TAILS		.316		* Agitator Head					4/1.	"	
" to -1" Scrubbed		.040									
" to -1" GRAVITY Concentrates		.897									
¼" to -½" Heads		.008		calc head .1034							
¼" to -½" Heads		.006					Refer: DAVE PRUETT for				
¼" to -½" GRAVITY TAILS		.048		Agitator Head TEST RESULTS							
¼" to -½" - Scrubbed		.022				2/1. upgrade calc.					
¼" to -½" GRAVITY Concentrates		1.135									
¼" Heads		.072		calc Ag head .172							
¼" HEADS		.062				upgrade					
from -¼" Head Scrub TEST		.157									
" GRAVITY Conc from Scrub Test		2.036									
-1 + ½"	Scrubbed 10 min lab ball mill wet										
	Gravity concentrate panned from -¼										

SHEET NO. 1 OF 1

B. ROBERTSON
ASSAYER

Sample Big Bne Pit Mine Run-Boundary Split 1A - (Screen Only) SCREEN ANALYSIS Date

Summa Corporati
Mining Division
Tonopah, Nevada

SIZE	WEIGHT (Grams)	ASSAY Tons	ASSAY Gold oz/T	Contained Gold Milligrams	% Total Gold	% Total Material	Cumulative % Gold	Cumulative % Material	Calculated Head oz/T	REMARKS
Mine Run			.188							Mine Run Crushed To (- 1")
+3"										Dry Screened. and weighed
-3"+1"										
-1"+½"	1381.9	47.38	.134	634.89	32.93	46.06	99.96 ~~100~~	99.52 ~~100~~		
-½+¼"	828.5	28.41	.107	303.99	15.76	27.62	67.03	53.46		
-¼+6m										
-6m+10m	235	8.06	.201	161.95	8.40	7.83	51.27	25.84		
-10m+35m	232.8		.424	338.43	17.~~35~~	7.76	42.87	18.01		
-35m+48m	43.2	1.48	.331 ~~.815~~	4902.20 ~~137~~	2.54 ~~7.11~~	1.44	25.32	10.25		
-48m+80m	49.1	1.68	.815 ~~.410~~	137.20 ~~302~~	7.11	1.64	22.78	8.81		
-80m	215	7.38	.410	302.25	15.67	7.17	15.67	7.17		
									.188 - Good Sample	

* 1 assay ton = 29.166 grams
.01 ounce/ton = 1 milligram/assay ton

Vault Pruett
ENGINEER

Sample Scrub Test 2 — Big Pine Mine Run - Boundy — SCREEN ANALYSIS — Date

Summa Corporat
Mining Divisio
Tonopah, Nevad

SIZE	WEIGHT (Grams)	ASSAY Tons	ASSAY Gold oz/T	Contained Gold Milligrams	% Total Gold	% Total Material	Cumulative % Gold	Cumulative % Material	Calculated Head oz/T	REMARKS
Mine Run		100	.188							Mine Run crushed To -1" scrubbed
+3"										in Lab Ball mill 10 min 20 balls- 2L water
-3"+1"										Weight on screen fractions are dry weights
-1"+½"	1184	42.35	.189	767.34	42.6		100	100	←	Poor Screening
-½+¼"										see- Screen test 1A
-¼+6m	721.5	25.81	.046	113.80	6.84		53.78	57.71		Bad Data - here -1+½
-6m+10m	84	3	.202	58.18	3.5		46.94	31.90		
-10m+35m	81	2.90	.217	58.94	3.54		43.44	28.90		
-35m+48m	69.2	2.48	.379	89.82	5.40		39.9	25.94		Test
-48m+80m	62.8	2.25	.454	97.61	5.87		34.5	23.46	.349	CN #1 Manhattan
-80m	593	21.21	.234	475.77	28.63		28.63	21.21		
	2795.5	100		1661.46					.173	

* 1 assay ton = 29.166 grams
.01 ounce/ton = 1 milligram/assay ton

ENGINEER

METALLURGICAL TESTING DATA

Summa Corporation
Mining Division
Tonopah, Nevada 89049

Type Test: Agitated Cyanide #1

Date: 14 May 76

SAMPLE DESCRIPTION	TIME	WEIGHT ORE / SOLUTION	ASSAY TONS	ASSAY GOLD oz./ton	CONTAINED GOLD mg.	ASSAY SILVER oz./ton	NaCN % / % NaOH	CONSUMPTION NaCN / NaOH	% GOLD RECOVERY	% SILVER RECOVERY
-35 Mesh Scrub #2 ~~Scrub #1~~ BPP Manhatten	0	300g. / 700ml	10.28	.378 calc	378					
Tailing-Residue	12 Hrs	300 gr.		.028	28.79					
Solution	12 Hrs ~~24~~	1200	41.14	.085	349.69				92.5%	
		Some solution lost in 24 Hr.								
		Pulp wash. - do not use results								
		of 24 Hr Test								

Paul E Pruett

Sample Scrub Test #4 (2c) Big Pine Pit Run Split 1c (1A-1B) SCREEN ANALYSIS Date

Summa Corporati
Mining Division
Tonopah, Nevada

SIZE	WEIGHT (Grams)	ASSAY Tons	ASSAY Gold oz/T	Contained Gold Milligrams	% Total Gold	% Total Material	Cumulative % Gold	Cumulative % Material	Calculated Head oz/T	REMARKS
Mine Run			.188							Mine Run Crushed
+3"										to -1". Scrubbed
-3"+1"										in Lab Ball Mill
-1"+½"	340 5	11 67	.028	38 44	1 75					10 minutes with 3c-water - 20 balls
-½+¼"										
-¼+6m	706 29	24 21	.101	244 52	11 17					Poor Split - Not enough coarse for
-6m+10m	165 80	5 68	.230	130 64	5 97					normal ratios however assays
-10m+35m	224 00	7 68	.322	247 29	11 30					per size range look good.
-35m+48m	62 -	2 12	.524	111 08	5 07					
-48m+80m	107 50	3 68	.406	149 40	6 83				.334	81% of Gold in 55% of Material
-80m	1239 -	42 48	.298	1265 90	57 87					

* 1 assay ton = 29.166 grams
.01 ounce/ton = 1 milligram/assay ton

David J. Rhoades
ENGINEER

Sample Heap #1 (+3" Leached) Scrubs SCREEN ANALYSIS Date

Summa Corporati
Mining Division
Tonopah, Nevada

SIZE	WEIGHT (Grams)	ASSAY Tons	ASSAY Gold oz/T	Contained Gold Milligrams	% Total Gold	% Total Material	Cumulative % Gold	Cumulative % Material	Calculated Head oz/T	REMARKS
Mine Run										+3" Material
+3"										from Heap #1.
-3"+1"										Crushed to (-1")
-1"+½"	2106	72 21	.024	173 30	25 91	36 26	100	100		Scrubbed in Lab Ball Mill - 3< H2O 20 Balls
-½+¼"										
-¼+6m	1674	57 39	.026	149 21	22 31	28 82	73 77	63 74		
-6m+10m	460	15 77	.008	12 91	1 88	7 92	51 46	34 92		
-10m+35m	521	17 86	.008	14 28	2 14	8 97	49 58	27 -		
-35m+48m	97 5	3 34	.016	5 34	0. 79	1 68	47 44	18 03		
-48m+80m	120 -	4 "	.060	24 66	3 68	2 06	46 65	16 35		
-80m	830	28 46	.101	287 44	42. 97	14 29	42 97	14 29		
	5808 5	199 15	278 81	668 84					.0335	

* 1 assay ton = 29.166 grams
.01 ounce/ton = 1 milligram/assay ton

ENGINEER

Sample Scrub Test #6 Big Pine Pit Mine Run Split ID SCREEN ANALYSIS Date

Summa Corporation
Mining Division
Tonopah, Nevada

SIZE	WEIGHT (Grams)	ASSAY Tons	ASSAY Gold oz/T	Contained Gold Milligrams	% Total Gold	% Total Material	Cumulative % Gold	Cumulative % Material	Calculated Head oz/T	REMARKS
Mine Run	3920	134.4	.188				100	100		
+3"	Mine Run crushed to -1", placed in Lab Ball Mill									
-3"+1"	Dry and Tumbled for 10 minutes then screened and weighed									
-1"+½"	1597	54 75	.008	43 8	1.6	41				comparison to
-½+¼"										Scrub Test #4 shows
-¼+6m	900	30 86	.084	259 22	9 45	23 1	98 40	59 65		1 - Dry more effective or 2 - Higher % Coarse
-6m+10m	140	4 8	.134	64 32	2 35	3 35	88 95	36 55		Scrubs better ? or 3 larger volume in ball mill scrubs better
-10m+35m										
-35m+48m										
-48m+80m										
-80m	1225	43 02	.552	2374 70	86 60	32 22	86 60	32 22		1140 grams to Test. CN #2 Manhattan
				26851785						

* 1 assay ton = 29.166 grams
.01 ounce/ton = 1 milligram/assay ton

Daniel J Pruett
ENGINEER

METALLURGICAL TESTING DATA

Summa Corporation
Mining Division
Tonopah, Nevada 89049

Type Test: Agitated Cyanide #2 Manhattan

Date: _____

SAMPLE DESCRIPTION	TIME	WEIGHT ORE / SOLUTION	ASSAY TONS	ASSAY GOLD oz./ton	CONTAINED GOLD mg.	ASSAY SILVER oz./ton	NaCN % / % NaOH	CONSUMPTION NaCN / NaOH	% GOLD RECOVERY	% SILVER RECOVERY
-10 m Scrub #6 Big Pine Pit Split 1D		1140 / 4600	3908	.552	2157 ±1		.1 / .1			
Residue	6 HRS.			.063 ~~.037~~	246.2				-11.4	
Solution	6 Hrs			.036					88.6%	
			Some solution spilled washing							
			filter cake - no solution cross check.							
			% looks average with 12 + 24 ~~HR~~							
			Tests.							

Sample +3" BPP - Screen #1 Boundy — SCREEN ANALYSIS — Date

Mining Divisi
Tonopah, Neva

Screen #9

SIZE	WEIGHT (Grams)	ASSAY Tons	ASSAY Gold oz/T	Contained Gold Milligrams	% Total Gold	% Total Material	Cumulative % Gold	Cumulative % Material	Calculated Head oz/T	REMARKS
Mine Run										
+3"	Sample from +3" pile in lab yard - Screen Analysis # 1 Boundy Big Pine Pit									
-3"+1"	crushed to minus 1" scrubbed 10 min									
-1"+½"	2461	84 38	.012	101.255	39 07	62 38	99 97	99 96		-Very low head assay-calc .019 Ratios holding
-½+¼"										
-¼+6m	606	20 78	.008	16 62	6 41	15 36	60 90	37 58		well despite lack of fines in head sample. !!
-6m+10m	243	8 32	.042	34 99	13 50	6 15	54 48	22 22		
-10m+35m	213	7 33	.035	25 56	9 86	5 39	40 98	16 07		
-35m+48m	38	1 3	.060	7 81	3 01	0 96	31 12	10 68	.0314	Calc. 54.48% Gold
-48m+80m	46	15 78	.080	12 62	4 86	1 16	28 11	9 72		22.2% material
-80m	338	11 589	.052	60 26	23 25	8 56	23 25	8 56		
	3945	135 241		259.13	99 96	99 96			.019	Calc Head.

* 1 assay ton = 29.166 grams
.01 ounce/ton = 1 milligram/assay ton

Dwight Burnett
ENGINEER

GEOLOGY
EXTRACTIVE SYSTEMS
MINE PLANT DESIGN

DAVID L. PRUETT
MINING GEOLOGIST
GOLDFIELD, NV 89013

May 7, 1976

Bill Robinson
Summa Corporation
3421 Las Vegas Blvd, South
Las Vegas, Nevada89019

Dear Mr. Robinson:

Here is the basic proposal and the main data sheets for equipment cost, installation cost, engineering fee, and operational costs. Our test work shows a 73% recovery to be possible with this arrangement.

I would require 50% of the estimate in advance, 25% on completion of the plant and turn key to you, and the balance on completion of the buildings. You may wish to write up some specifications for the work. I would be responsible for all costs, labor, and engineering supervision. If this goes thru we can hash out the details.

Sincerely,

David L. Pruett

David L. Pruett

cc: David K. Hamilton

Also filed in Mill construction folder

GEOLOGY
EXTRACTIVE SYSTEMS
MINE PLANT DESIGN

DAVID L. PRUETT
MINING GEOLOGIST
GOLDFIELD, NV 89013

May 7, 1976

MANHATTAN CARBON-IN-PULP
Proposed Flow Sheet
50 Ton Per Hour Plant Feed

HP		COST
15	Wobble Feed Scalper	$ 9,500
	-8"	*2,500
	+8"	Summa
5	30" Conveyor	*2,500
		Summa
30	Hammer Mill	*2,500
		Summa
5	24" Conveyor	*2,500
	Stockpile	*1,500
5	Tunnel Load Conveyor	7,500
		*4,000
	Merric Weightometer	2,700
		*1,500
150	Ball Mill Scrubber	10,000
		*4,500
10	Screen-48 Mesh	4,500
		*1,500
	+48M Tail	
15	4" Pump Cyclone	3,200
		*1,000
25	Agitator #1	10,000
		*2,500

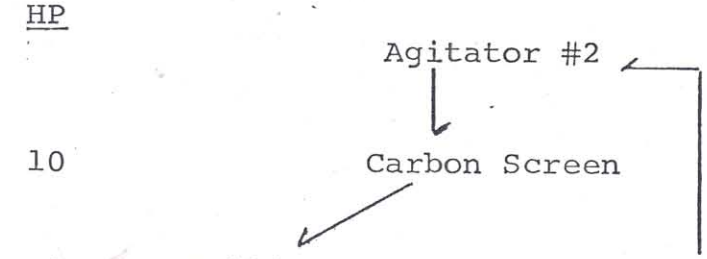
* Installation Cost

RLP

GEOLOGY
EXTRACTIVE SYSTEMS
MINE PLANT DESIGN

DAVID L. PRUETT
MINING GEOLOGIST
GOLDFIELD, NV 89013

Flow Sheet - Page Two



HP			COST
		Agitator #2	$ 10,000
			*2,500
10		Carbon Screen	*4,500
			* 500
1	Tailing	Carbon Conveyor	1,500
			* 500
290 Hp Total		Total Equipment	58,900
		Installation	*30,000
	40 x 60 Building - New - Metal Insulated		25,000
	20 x 30 Building		6,000
		Total Buildings	31,000
	Tailing Pond by Summa		
		Subtotal	$119,900
		Engineering	25,000
		Total Project	$144,900

DLP

GEOLOGY
EXTRACTIVE SYSTEMS
MINE PLANT DESIGN

DAVID L. PRUETT
MINING GEOLOGIST
GOLDFIELD, NV 89013

May 7, 1976

Manhattan Carbon-in-Pulp
Direct Operating Cost
50 Ton Per Hour

	Cents/Ton
Electrical 225 KW $7.00/hour	$.15
Chemical	.50
Labor $280.00/shift 3 Men plus Supervisor 2 Crusher, 1 Plant Operator	.70
Maintenance and Wear	.10
General	.20
TOTAL	$1.65/Ton

DLP

GEOLOGY
EXTRACTIVE SYSTEMS
MINE PLANT DESIGN

DAVID L. PRUETT
MINING GEOLOGIST
GOLDFIELD, NV 89013

MANHATTAN CARBON-IN-PULP
Scrubbing Tests May 5 & 6 - Boundy and Pruett

Material: Big Pine Pit Screen Analysis #4 - Group 26

3 Tests on (-1"+ ½")(-½" + ¼") and (-¼")

100 Assay tons of each was placed in a 12" x 12" lab ball mill with 20 - 1" steel balls, 2 L. of water and tumbled for 10 minutes each.

Summary: 16.7% of total Au was rejected in the -1" + ¼" scrubbed material. 83.3% of total Au is contained in the -¼". This is an suitable product for carbon-in-pulp. If the Skinner work on scrubbing last year bears out 76% of this Au will be -48 mesh.

This test work should be continued.

DLP

Internal Communication

summa

Date: April 26, 1976

To: David K. Hamilton

From: William J. Robertson

Subject: Melting Results from Englehard Refinery

I am pleased to report on our melting results of 41 bars of dore bullion at Englehard Refinery, in Newark, New Jersey on April 22, 1976.

The bars were divided into five separate melts and melted under Control #CN31579-1CA thru #CN31579-5E. Each lot was weighed in by Englehard and witnessed by me. The results were as follows:

	Weights Troy Ounces	
Melt #	Before Melt	After Melt
CN31579-1(a)	867.150	865.680
CN31579-2(b)	898.280	897.820
CN31579-3(c)	891.340	890.310
CN31579-4(d)	805.030	805.190
CN31579-5(e)	378.150	377.870
	3839.950	3836.870
Less Summa's samples (5 x 0.250)		1.250
	Settlement Weight	3835.620

Upon completion of assays, we will effect our assay exchange.

I went back with the inventory list showing 3855.91 Troy ounces of dore metal had been shipped. An error in bar #19 had been discovered at Englehard. Summa showed bar #19 to weigh 184.60 oz. where Englehard's weight showed bar #19 to weigh 168.54 Troy oz. We then searched to determine how an error of such weight could have happened in our weighing. We discovered that one of two things could have happened.

Our bar weights are determined by first weighting each bar on a kilogram balance. The weight received in grams is then calculated into Troy oz. by dividing the amount of grams by 31.1 (31.1 grams equal 1 Troy oz.) After calculating back the possibility of the wrong weight in grams being either read wrong at the balance or entered wrong in the calculator had happened. Example: 184.60 Troy oz. = 5741.06 grams; 168.54 Troy oz. = 5241.59 grams.

What should have been 5241 grams was calculated 5741 grams. That is the only possibility that I can see where an error that large can be made. The difference in the tenths of the oz. could be weighing between the different balances.

Sampling at Englehard

After the five melts were made the sampling of our bullion then took place which I witnessed.

During lots 1 - 5 a sample bar was poured for determining fineness of each lot. Englehard's procedure of sampling is as follows:

Lots #1 - 4 ranged between 800 - 900 oz dore
Lot #5 weighed befor melt 378.150 oz dore

Bars were poured weighing approximately 400 oz. Each lot produced the 400 oz. bar - a sample bar and then what they call a scrap bar weighing under 400 oz. The sample bar is then locked in a steel container and sent out to be sampled. Englehard's sampling procedures are very effective. The bar is sawed in three different places, all residue is collected, mixed and a sample is held for Englehard, one given to the customer, and one sealed and vaulted for an umpire if necessary.

After all sampling was completed, I received 5 samples covering the 5 lots each weighing 0.250 oz. Troy for determining fineness in our facility locally. The samples have been turned over to Don Tomany for safe keeping until I am ready to test them.

The following day I toured Englehard's facilities, in which a report will be written stating my findings.

Internal Communication

summa

Date: April 26, 1976

To: David K. Hamilton

From: William J. Robertson

Subject: Direct Shipment of Bullion to Englehard as we Process It

Upon arrival at Englehard's plant in New Jersey, I discussed with Mr. R. L. Searle, commercial manager of the refinery, the possibility of Summa making direct shipments of bullion by Postal Service as we produce it. I found out that bullion can be shipped directly by registered mail in accordance to U. S. Postal Regulations. Englehard does prefer to work bullion in amounts of 400 oz. lots. They will, however, inventory our bullion until they receive approximately 400 oz. They have a minimum charge of $250.00 for refining.

summa

Internal Communication

Date: April 14, 1976

To: David K. Hamilton

From: William O. Mollison

Subject: Stripping Plant Progress

Realizing that you all are under internal pressures, I am at the present time trying (with extremely limited help) to construct and finalize the gold stripping unit designed by David Pruett.

With the help of only one person it is impossible to meet your projected completion date of 4/19/76, especially with modifications required.

With stated deliveries from our purchasing agent and present availability of a competent welder, helper, etc., I would estimate that as of 4/14 completion would require 30 working days.

If adequate and competent help is available immediately, I'm sure this time could be halved. This half-time is also based on assured competency of outside contractors -- Logan, Skanovsky, Perchetti -- and their stated dates of completion.

Internal Communication

summa

Date: April 14, 1976

To: Paul G. Reeve

From: William J. Robinson WJR

Subject: Sample Priorities and Assay Distribution

Attached are the memoranda regarding sample priorities and assay distribution. The priorities were established to give the quickest results for production work. All samples are assayed rapidly and no samples have been delayed because of the priority system.

As a matter of fact, the exploration samples are presently being picked up twice a week with an average assay turn-around of 3 days. Previously, the exploration samples were picked up once a week with a 7-day turn-around. This means a faster assay result process for exploration samples because of a scheduled flow of work.

Assay distribution is directed to the concerned parties and the individual who requested the assay.

Attachments

(Virgil Barker Tongah)

PURCHASE ORDER

SITKIN SMELTING & REFINING, INC.

No. MP 2100

BRASS, BRONZE & ALUMINUM INGOTS
GOLD, SILVER & NOBEL METALS
P.O. BOX 708 - R.D. # 3
LEWISTOWN, PA. 17044
PHONE: (717) 543-5631

SAMPLE 5

MR. JACK BARRETT
P. O. BOX 267
CAMPTONVILLE, CA 95922

Date _____

TERMS FOR REFINING & PURCHASE OF GOLD AND/OR SILVER.

SITKIN SMELTING & REFINING, INC. AGREES TO PURCHASE FROM THE ABOVE

QUANTITY	MATERIAL	PRICE

MATERIAL: To be determined.

PAYMENT: Settlement in full ten days after receipt, based upon outturn.

Gold: 96.4% of London Metal Exchange second fixing price, date of settlement.

Silver: 96.4% Handy & Harman quote, less $0.20 per troy ounce, date of settlement.

CONDITIONS: Shipment must contain a minimum lot of 5 troy ounces of contained metal.

FREIGHT: For customer account.

Refining chg: quoted upon request.

F.O.B.

Ship On or Before

Terms

Remarks

Thanks U

① HYDRAULICALLY BRICKED OR BRIQUETTED MATERIAL NOT ACCEPTABLE.

② ANY MATERIAL OVER 15 INCHES IN MORE THAN TWO DIMENSIONS WILL BE SUBJECT TO A PREPARATION CHARGE.

③ VARIATION OF PLUS OR MINUS 5% FROM THE AGREED-UPON AMOUNT IS ALLOWABLE.

KINDLY SIGN THE ORIGINAL COPY AND RETURN IT TO US FOR OUR FILES, RETAINING THE OTHER COPY FOR YOUR FILES.

Accepted _____

By _____

Date _____

SITKIN SMELTING & REFINING, INC

By LEWIS SITKIN, C.E.O.

WAREHOUSE COPY

ABOVE SUBJECT TO ATTACHED CONDITION SHEETS

Virgil Barker
Tonopah.

PURCHASE ORDER

SITKIN SMELTING & REFINING, INC.

No. MP 2098

BRASS, BRONZE & ALUMINUM INGOTS
GOLD, SILVER & NOBEL METALS
P.O. BOX 708 - R.D. # 3
LEWISTOWN, PA. 17044
PHONE: (717) 543-5631

SAMPLE

THE BLUE DICK MINE & MILLING
P. O. BOX 248
GOLDFIELD, NV 89013
ATTN: MRS. JEAN FOSTER

Date

COMMITMENT REQUIREMENTS FOR PROCESSING ORE CONCENTRATES.

SITKIN SMELTING & REFINING, INC. AGREES TO PURCHASE FROM THE ABOVE

QUANTITY	MATERIAL	PRICE

VALUE BASIS & SITKIN/CUSTOMER SPLIT

Value in Recoverable Gold and/ Or Silver per Net Ton	Sitkin/Customer Net Split
$1,000.00 - $2,500.00	50% Sitkin / 50% Customer
$2,501.00 - $3,500.00	30% " 70% "
$3,501.00 - $4,500.00	25% " 75% "
$4,501.00 - $6,000.00	20% " 80% "

PAYMENT

30% of customer share ten days after receipt of concentrates at Sitkin plant, based upon Sitkin assay. Final payment and settlement 60 days after receipt of concentrates at Sitkin plant based upon smelters outturn.

PRICE

Gold: 96.4% of London Metal Exchange second fixing price, date of settlement.

Silver: 96.4% of Handy & Harman quote, date of settlement.

CONDITIONS: 1. A preliminary assay of all concentrates is to be made by General Engineering Company, Tonopah, NV, or by Sitkin Smelting & Refining, Inc., Lewistown, PA
2. Shipments will not be accepted by the smelter without an authorized purchase order issued by Sitkin Smelting & Refining, Inc.

F.O.B. All freights are to customers account, shipped collect to smelter.

Ship On or Before

Terms--

Remarks

Thanks u

① HYDRAULICALLY BRICKED OR BRIQUETTED MATERIAL NOT ACCEPTABLE.

② ANY MATERIAL OVER 15 INCHES IN MORE THAN TWO DIMENSIONS WILL BE SUBJECT TO A PREPARATION CHARGE.

③ VARIATION OF PLUS OR MINUS 5% FROM THE AGREED-UPON AMOUNT IS ALLOWABLE.

KINDLY SIGN THE ORIGINAL COPY AND RETURN IT TO US FOR OUR FILES, RETAINING THE OTHER COPY FOR YOUR FILES.

SITKIN SMELTING & REFINING, INC

Accepted

By

Date

By LEWIS SITKIN, C.E.O.

WAREHOUSE COPY

ABOVE SUBJECT TO ATTACHED CONDITION SHEETS

MANHATTAN PLANT - CARBON CLOGGING

The problem is that Carbon at the Manhattan Plant is not accepting Gold and that something is seriously clogging it. So after sending a Carbon sample from column two to Skyline Labs, they reported that the Carbon contained .7% Mg, 1% Ca, 500 PPM Ag, 1000 PPM Mn and trace amounts of other metals. (Page 3). The specific question I had asked the Head of the Lab, Mr. Thompson (a Metallurgist) was what was clogging the Carbon. His conclusions were that the only thing that could hamper our loading capabilities would be the large amount of Magnesium present in the Carbon. This seems like a logical conclusion because of the high amount of Mg present and because Mg is highly susceptible to Cyanide. Also the spec on the rock from the Manhattan area shows that Magnesium varies from 500-1000 ppm. This explains why the Carbon will not load more Gold, because it is already loaded with Magnesium.

The electrochemical order of metals in Cyanide explains this very clearly.

+Mg
Al
Zn
Cu
Au
Ag
Mg
Pb
Fe
-Pt

This shows that Mg will dissolve faster then Gold and that Mg will kick the Au off of the Carbon. Ag will load first, but then the Au will start kicking it off, then the Mg will start loading kicking Au off the Carbon.

From this it can be seen why the tails are sometimes higher then theheads, because Gold is being kicked out of Column 1 by the Magnesium. For a final test I set up the AA for Magnesium and tested heads, tails and tails #2. The results are on Page 4, 5, and 6. These results show that some Mg is loading on the Carbon but not nearly as much as I thought. The Mg is acting very similiar to Au in its complete randomness of loading. So the conclusion from this is that Mg is not what is blocking the Carbon, because the Mg is being blocked also. So, although, Mg is loading, its effect is not that adverse to our system.

Other experiments tried, have been to bring up the Lime Concentration, and then bringing the Cyanide Concentration up. Both these experiments had no positive results. So knowing that the concentration of Cyanide and Lime is not the problem and that there is not much else shown in the spec that would give us our problem. The only thing left to try is the Carbon itself. Maybe there is something wrong with the last couple batches of Carbon used. The last two batches have not been screened or washed and they are a different brand Carbon then used on our very successful run. So maybe this is the answer. This should be the next thing tried; new Carbon screened and washed and the old brand.

Then if this does not work, I recommend changing all solution in the circuit and cleaning the mixing tank, pond and pipelines. Also, to bring the AA out to the area for instant results on solutions running through the system. Hopefully the problem can be placed into one of these areas, the Carbon or Stale Solution.

Paul Ricks

Paul Ricks

Dated July 30, 1974

SKYLINE LABS, INC.
SPECIALISTS IN EXPLORATION GEOCHEMISTRY
12090 WEST 50TH PLACE • WHEAT RIDGE, COLORADO 80033 • TEL.: (303) 424-7718

REPORT OF SPECTROGRAPHIC ANALYSIS

Job No. M-2666
July 24, 1974

Summa Corporation
P.O. Box 1126
Tonopah, Nevada 89049

Attention: Paul Ricks

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.

Element	Sample No. K2	Carbon LDJ
Fe	10%	<.02%
Ca	.2%	1%
Mg	.2%	.7%
Ag	5	500
As	<500	<500
B	20	50
Ba	1,000	10
Be	2	<2
Bi	<10	<10
Cd	<50	<50
Co	30	<5
Cr	300	<10
Cu	150	20
Ga	20	<10
Ge	<20	<20
La	<20	30
Mn	500	1,000
Mo	10	<2
Nb	20	<20
Ni	70	50
Pb	50	<10
Sb	<100	<100
Sc	20	<10
Sn	10	<10
Sr	200	<50
Ti	2,000	20
V	50	<10
W	<50	<50
Y	70	<10
Zn	<200	<200
Zr	150	<20

SUMMA CORPORATION
MINING DIVISION
Tonopah

JUL 26 1974

RECEIVED

Charles E. Thompson
Chief Chemist

Paul Ricks

LABORATORY ASSAYS OF MANHATTAN PLANT SAMPLES
FOR GOLD AND SILVER

DATE 7\26\74

TIME	SAMPLE NO.*	PPM Au	PPM Ag
6:00 AM	HEADS	.56	
	COL #1	.38	
	TAILS #1	.16	
	TAILS #2	.16	.099
2:00 PM	HEADS	.52	.061
	COL #1	.16	.060
	TAILS #1	.16	.056
	TAILS #2	.04	.054

* Assay samples taken from the following points:

SAMPLE NO.	SAMPLE LOCATION
1	Weir overflow from dump
2	Pregnant to column 1
3	Barren from column 1
4	Barren from column 2
5	Barren from column 3
6	Barren from column 4
7	Leach solution to dump

FOR GOLD AND SILVER

DATE 7\22\74

TIME	SAMPLE NO.*	PPM Au	PPM Ag
2:00 PM	HEADS	.43	.039
	COL #1	.24	.021
	TAILS #1	.24	.013
	TAILS #2	.03	.070
10:00 PM	HEADS	.64	.031
	COL #1	.32	.030
	TAILS #1	.16	.030
	TAILS #2	.23	.066
6:00 AM	HEADS	.60	.049
	COL #1	.60	.046
	TAILS #1	.24	.051
	TAILS #2	.16	.354

* Assay samples taken from the following points:

SAMPLE NO.	SAMPLE LOCATION
1	Weir overflow from dump
2	Pregnant to column 1
3	Barren from column 1
4	Barren from column 2
5	Barren from column 3
6	Barren from column 4
7	Leach solution to dump

LABORATORY ASSAYS OF MANHATTAN PLANT SAMPLES
FOR GOLD AND SILVER

DATE 7/18/74

TIME	SAMPLE NO.*	PPM Au	PPM Ag (Hg)
6:00 AM	HEADS	.56	.049
	COL #1	.40	.046
	TAILS #1	.16	.051
	TAILS #2	.16	.354
2:00 PM	HEADS	.60	.089
	COL #1	.16	.021
	TAILS #1	.05	.013
	TAILS #2	.05	.070
10:00 PM	HEADS	.56	.031
	COL #1	.40	.030
	TAILS #1	.16	.030
	TAILS #2	.16	.046

* Assay samples taken from the following points:

SAMPLE NO.	SAMPLE LOCATION
1	Weir overflow from dump
2	Pregnant to column 1
3	Barren from column 1
4	Barren from column 2
5	Barren from column 3
6	Barren from column 4
7	Leach solution to dump

Internal Communication

summa

PR-8-76-280

Date: August 3, 1976

To: W. J. Robinson

From: P. G. Reeve

Subject: Letter from Freeport Exploration Re: Mary Mine

Attached for your information is a copy of the subject letter. I believe you or Fred Saunders had earlier indicated to me that Freeport had experienced some problems with the assays of samples taken during their recent work at the Mary Mine. This confirms that information.

Please add this letter to your file of Mary Mine data.

PGR:LTP

Attachment

BCC: Bob Lutz w/attachment
Fred Saunders

FREEPORT

FREEPORT EXPLORATION COMPANY • SECURITY NATIONAL BANK BUILDING, RENO

ADDRESS CORRESPONDENCE P. O. BOX 1911
RENO, NEVADA 89505
702-323-2251
TELEX: 910-395-7008

July 27, 1976

Mr. Paul G. Reeves
Summa Corporation
P. O. Box 14000
Las Vegas, Nevada 89156

Re: Mary Mine - Esmeralda County, Nevada

Dear Mr. Reeves:

Due to series of errors in assaying our samples taken in the Mary Mine we have been delayed in pursuing further discussions regarding an association.

We took 98 samples in the mine which were assayed by Atomic Absorption by Rocky Mountain Geochemical Corporation in Reno with 20 check samples fire assayed by Union Assay in Salt Lake. The fire assaying generally showed about double the values obtained by AA so we had Rocky Mountain fire assay everything in their Salt Lake Laboratory. The comparison of these results showed a marked variation so both companies re-assayed the pulps with similar irregular results.

The conclusion was that the gold occurs as coarse grains and that accurate assaying would require minimum 2 to 5 assay ton samples. Freeport have a Research and Development Laboratory in Louisiana which has sophisticated X Ray (non dispersive) computer print out equipment and they are now looking at a composite sample to try and determine the characteristics of the gold. As soon as we have these results we will have a better idea of the problems involved in exploring the mine.

We are unfortunately stymied in selling an exploration program to our management until we resolve the reasons for the variations in the sample results. As soon as we find an answer to these problems we will resume our discussions; however, please feel free to talk to other prospective partners during this period.

Sincerely yours,

FREEPORT EXPLORATION COMPANY

E. B. Bell
District Exploration Manager

EBB:nm

Copy J. Fillerup 7-29-76

President

S. Parker Gay, Jr. has been engaged in mineral exploration, airphoto interpretation, and groundwater exploration for twenty-three years. He holds a B.S. degree in geology from MIT (1952), an M.S. degree in geophysics from Stanford University (1961) and is a registered geologist in California. He has several major mineral and groundwater discoveries to his credit in Peru, Alaska and several western states and has made a number of important contributions to the technology of three fields of endeavor, as attested to by his many publications and a number of awards. One of his major interests is the control of Precambrian basement fracturing on later tectonism, the effect of this fracturing on petroleum and mineral occurrences, and ways and means of detecting the fracture systems by geophysical methods.

Vice President

David A. Smith has been engaged in geophysical and mineral exploration and in commercial aviation pursuits for twenty-one years. He holds a B.A. degree in Earth Physics from UCLA (1960) and is a registered geophysicist in California. He has carried out and directed ground geophysical surveying in Ontario, Alaska, and all eleven western states, and has been involved in flying operations in Africa, Central America and South America. His present major interest is the modeling of three-dimensional IP anomalies by various electrode arrays for improving AGI's survey effectiveness.

Field Operations Manager

Frederick R. Hilton has carried out and directed geophysical operations for Applied Geophysics since its founding in 1971 and before that with another major contractor for two years. He is a graduate geologist (B.S. University of Utah, 1969) and thus carries out geophysical surveying consistent with the geologic conditions of each property surveyed. Fred specializes in the difficult or impossible surveys: helicopter-supplied tent camps during the rainy season in Alaska, burro or mule camps (including transportation of our heavy transmitters) high in the Sierra Madre of Mexico, or long camp-ins at the end of a tough six-hour four-wheel drive in the Idaho or Montana wilderness.

CONSULTANTS AND CONTRACTORS TO THE MINING INDUSTRY

applied geophysics inc

675 SOUTH 4TH EAST · SALT LAKE CITY, UTAH 84111 · (801) 328-8541

applied geophysics

EXPLORATION EXPERTISE

Applied Geophysics, Inc. has, since its founding in 1971, carried out dozens of ground geophysical projects in all eleven western states plus Alaska and several provinces of Mexico. Fixed wing airborne geophysical surveys have likewise been carried out since 1972 in eight western states, and helicopter-borne surveys since 1974 at several localities in both the eastern and western United States. Prior to incorporation of AGI, the two principal partners carried out geophysical surveying for many years and at various times in South America and Canada. Clientele over the years have included most of the major exploration companies in the mining business, plus a number of government agencies. Our references would include exploration or divisional managers for all these many organizations.

We would welcome the reader as a new or repeat client.

applied geophysics inc

675 South 4th East
Salt Lake City, Utah 84111
Phone (801) 328-8541

applied geophysics

EXPLORATION PHILOSOPHY

Applied Geophysics' exploration philosophy is to employ for our clients those geophysical tools which are most helpful to them on each exploration project *based on the geological conditions of that particular project.* We have no axes to grind as to types of geophysical methods or kinds of equipment employed; we use them all, but *each in its proper place* and each based on field experimentation, not on wishful, purely-mathematical, unwarranted-computer-approach, or unproven-physical-principal thinking. Consequently, we have enjoyed a very good reputation among exploration managers and have an excellent track record of successful exploration projects including correct evaluation of many unpromising prospects, as well as making a number of important discoveries on good properties.

Where only the *data* from airborne or ground geophysical surveys is desired and no interpretation, we gather it as carefully and as accurately as if we ourselves were charged with the responsibility of evaluating it.

applied geophysics inc
675 SOUTH 400 EAST • (801) 328-8541 • SALT LAKE CITY, UTAH 84111

July 29, 1976

Gentlemen:

Since its founding in 1971, Applied Geophysics, Inc., has grown to be the largest combined air and ground mining geophysical contractor in the United States. We average around 25 employees on a year-round basis and have from two to five survey crews out at any given time. Our clients seem to be satisfied with our work, and we feel this is due to our common-sense approach to geophysics and the high quality of our data. High quality data result from: 1) good people, 2) good equipment, and 3) experience. We have all three.

Lately, we have been carrying out a number of helicopter-borne magnetic and gamma-ray spectrometer surveys, singly or in combination. All our airborne data, both fixed-wing and helicopter, are gathered digitally on one-half inch magnetic tape; magnetic data are recorded on analog charts also and compiled manually on request. We have developed our own digital data system, which is the best in industry, and we are presently marketing it.

On the ground, our volume of EM surveying is, for the first time, approaching that of IP surveying. We also carry out on a routine basis, ground magnetics, gravity, and resistivity (for geothermal surveys and ground water); and on one recent survey we were able to sort out and evaluate vein-type IP-EM anomalies with the "old fashioned" (but underrated) self-potential method. So we are flexible, and as stated in the attached brochure, we employ any and all geophysical methods as required by the geological conditions present in each project area.

Call us any time. We would be pleased to discuss the proper geophysical tools to use on your exploration project.

Yours very truly,
APPLIED GEOPHYSICS, INC.

S. Parker Gay Jr.

S. Parker Gay, Jr.
Chief Geophysicist

SPG/mb

P.S. This summer and fall we will have ground crews at locations in several western states and can quote mob-free rates for many areas.

Randy Burke

Gr. 26

Salt Lake City Metallurgy Research Center

March 7, 1974

Mr. David J. Gribben
Summa Corp.
5700-B So. Haven
Las Vegas, Nev. 89119

Dear Dave:

We completed a percolation cyanidation leaching test on a sample of gold ore submitted by Randy Burke and identified as Manhattan, Nev., pit run ore. Our calculated head assay of the material was 0.09 oz gold and 0.08 oz silver per ton, and the sample was about 35 percent plus 4 inches in size. The test procedure and detailed results are given in the attached memorandum from Harris Salisbury.

The test results indicated that about 80 percent of the gold in the calculated head was recovered in 600 hours of leaching. Reagent consumption was 1.5 lb NaCN and 3.3 lb CaO per ton of ore.

We hope this information will be helpful.

Sincerely yours,
Signed George M. Potter

George M. Potter
Research Supervisor

Enclosure

cc: ✓ Randy Burke

SUMMA CORPORATION
MINING DIVISION
Tonopah

MAR 11 1974

RECEIVED

Salt Lake City Metallurgy Research Center

March 7, 1974

Memorandum

To: George M. Potter, Research Supervisor, Salt Lake City Metallurgy Research Center

From: Harris B. Salisbury, Project Leader

Subject: Percolation cyanidation test on ore submitted by Randy Burke, Summa Corp.

Approximately 267 lb of pit-run ore was loaded into a 14-inch diameter transite column. Lime solution was percolated downward through the bed at 16 ml/min until a pH value of 10.4 was reached. The solution was then made up to 0.1 pct NaCN and percolation continued for 600 hours, at which point the pregnant solution assayed 0.09 ppm Au and 0.05 ppm Ag, values considered too low to justify further leaching. The pregnant solution was collected in 24-hour increments then weighed and assayed before passing through an activated carbon column prior to reuse. The first carbon removed after 384 hours assayed 252 mg Au and 115 mg Ag compared to cumulative solution assays that indicated a recovery of 268 mg Au and 126 mg Ag. The final char was lost making it necessary to use the solution assays and atomic absorption residue assays for final calculations. Consumption of NaCN was 1.5 lb per ton of ore, and CaO usage was 3.3 lb.

Harris B. Salisbury

Attachment

E 28 Idm 73.2 2-25-74

Leaching time, hours		Solution volume	Assay, ppm		Metal Content Mg		Cumulative Metal Content Mg		CUM oz/TON RECOVERED		CUM % RECOVERED	
INCREM.	CUM	LITERS	Au	Ag	Au	Ag	Au	Ag	Au	Ag	Au	Ag
24	24	18.596	1.2	.94	22.3	17.5	22.3	17.5	.006	.004	6.6	5.0
✓	48	17.722	2.2	1.03	39.0	18.3	61.3	35.8	.015	.009	16.5	16.3
✓	72	20.768	2.0	.86	41.5	17.9	102.8	53.7	.026	.014	28.6	17.5
✓	96	21.569	1.45	.66	31.3	14.2	134.1	67.9	.034	.017	37.4	21.3
✓	120	20.992	1.1	.52	23.1	10.9	157.2	78.8	.040	.020	44.0	25.0
✓	144	21.330	.9	.42	19.2	9.0	176.4	87.8	.044	.022	48.4	27.5
✓	168	21.809	.65	.33	14.2	7.2	190.6	95.0	.048	.024	52.7	30.0
✓	192	22.010	.60	.26	13.2	5.7	203.8	100.7	.051	.025	56.0	31.3
✓	216	21.703	.55	.23	11.9	5.0	215.7	105.7	.054	.027	59.3	33.8
✓	240	20.939	.60	.19	12.6	4.0	228.3	109.7	.058	.028	63.7	35.0
✓	264	20.881	.40	.17	8.3	3.5	236.6	113.2	.060	.029	65.9	36.3
✓	288	21.474	.25	.14	5.4	3.0	242.0	116.2	.061	.029	67.0	36.3
✓	312	21.618	.45	.13	9.7	2.8	251.7	119.0	.063	.030	69.2	37.5
✓	336	21.187	.30	.12	6.4	2.5	258.1	121.5	.065	.031	71.4	38.8
✓	360	20.316	.25	.12	5.1	2.4	263.2	123.9	.066	.031	72.5	38.8
✓	384	22.699	.21	.11	4.8	2.5	268.0	126.4	.068	.032	74.1	40.0
✓	408	22.679	.12	.13	2.7	2.9	270.7	129.3	.068	.033	74.1	41.3
✓	432	25.747	.10	.14	2.6	3.6	273.3	132.9	.069	.034	75.8	42.5
✓	456	22.436	.15	.15	3.4	3.4	276.7	136.3	.070	.034	76.9	42.5
✓	480	22.529	.12	.10	2.7	2.3	279.4	138.6	.070	.035	76.9	43.8
✓	504	22.398	.12	.07	2.7	1.6	282.1	140.2	.071	.035	78.0	43.8
✓	528	22.603	.09	.06	2.0	1.4	284.1	141.6	.072	.036	79.1	45.0
✓	552	11.947	.14	.09	1.7	1.1	285.8	142.7	.072	.036	79.1	45.0
✓	576	21.185	.14	.06	3.0	1.3	288.8	1440	.073	.036	80.2	45.0
✓	600	17.906	.09	.05	1.6	.9	290.4	144.9	.073	.037	80.2	46.3
RESIDUE									.018	.043	19.8	53.7
CALC		HEAD.			Tailing loss				.091	.080	100.0	100.0

Land Exploration
and Mining Division

Post Office Box 1126
Tonopah Nevada 89049
702 482 3584

A Division of
Summa Corporation

Bureau Labs.
Metallurgy

June 26, 1975

Mr. R. S. Shoemaker
Consulting Metallurgist
Mining & Metals Division
Bechtel Corporation
Fifty Beale Street
San Francisco, California 94119

Dear Bob:

Many thanks for your time and effort in our behalf. We also appreciate your summation of our telephone call for our files.

We trust that you and Clarence will meet each other tomorrow as we have left a message for Clarence at Metallurgical Laboratories to that effect.

With best wishes,

Sincerely,

WS:sfm

Walt Simmons
General Superintendent

P.S. Just heard from Clarence. He has Bob Baker with him and both are returning to Tonopah today. Mr. Baker feels that he can do what we want right here.

August 7, 1975

TO: Distribution

FROM: R. E. Baker

SUBJECT: General Information

1. The last two bars have shown an increasing percentage of silver.

2. The ratio of silver to gold in the pregnant solution has been increasing over the last while.

3. The steel wool cathodes have been dissolved in hydrochloric acid, which will precipitate rather than dissolve silver.

4. The sludge from the sodium sulfide precipitation of silver contains more gold than silver.

5. Gold can be removed from pregnant carbon using sodium sulfide solution as a solvent. The U. S. Bureau of Mines in Reno used sodium sulfide to strip carbon prior to their use of hot cyanide solution. This was discontinued because it did not remove silver from the carbon.

6. If silver is to be separated from gold by means of acid it is accomplished by the use of nitric acid. This parting is only possible if the ratio of silver to gold is greater than three parts of silver to one part of gold.

   The exception to this is the method of electrolysis. By this method the dore' is rolled into thin sheets and used as an anode; the cathode is a sheet of pure gold; the electrolyte is nitric acid. When a current is passed through, the pure gold deposits on the cathode, the silver goes into solution as silver nitrate along with copper, iron, etc. The platinum group metals remain as sludge. This method is used by the mint.

7. Another method is to pass chlorine gas through the molten metal. Silver chloride floats on the surface of the melt and is skimmed off, along with some gold, as silver chloride. This is an imcomplete and cumbersome process. Graphite crucibles cannot be used because the oxidizing action of the chlorine attacks the graphite.

   Furthermore, the thermal shock of the cold, expanding gas makes it difficult to prevent ceramic crucibles from cracking or breaking.

8. The last bar had a higher percentage of silver than the one previous even though twice as much sodium sulfide had been used in order to remove silver from solution.

9. Manganese dioxide has been reported to have the effect of carrying silver into the slag. However, the use of this compound has a tendency to make the slag viscous with the attendant potential loss of metal in the slag.

10. When Zadra of the Bureau of Mines at Reno first started his strip circuit at the Getchell Mine he used sodium sulfide solution as the means of removing gold from pregnant carbon. The gold was removed satisfactorily but not the silver. The result was that the carbon, on reuse, was partially loaded with silver and was, thus, less efficient for the adsorption of gold from the cyanide solution.

    It is possible that this is the cause of our present high barren solutions. Reactivated carbon is being used which is still partially loaded with silver.

R E Baker

R. E. Baker

Distribution:

W. Simmons
D. J. Gribbin
W. O. Mollison
W. Robertson
C. Sikkenga
R. E. Baker
Tonopah file

HUSKY INDUSTRIES
Inc.
INDUSTRIAL DIVISION
ROUTE 1, BOX 275 / DUNNELLON, FLORIDA 32630 / TELEPHONE (904) 489-3336

July 14, 1975

Mr. Walt Simmons
General Superintendent
Summa Corporation
Box 1126
Tonopah NV 89049

Dear Walt:

Thank you for your letter concerning activated carbon. We hope to be able to provice the service you need.

We have developed a source of carbon for the approximate sizes you indicated interest in. To facilitate rapid handling of the request, I need to know the quantities Summa is interested in purchasing, the point of delivery, and the price that will make our carbon interesting to you. Also, could a 4 x 8 or 4 x 10 carbon be utilized?

I appreicate your interest and look forward to your reply.

Sincerely,

D C Norcross

D. C. Norcross
Vice President & General Manager
Industrial Division

DCN/saw

Land Exploration
and Mining Division

Post Office Box 1126
Tonopah Nevada 89049

A Division of
Summa Corporation

File: Carbon

July 21, 1975

(Dict. July 18, 1975)

Mr. D. C. Norcross
Vice President & General Manager
Industrial Division
Husky Industries
Route 1 Box 275
Dunnellon, Florida 32630

Dear Cope:

Thank you for your letter of July 14.

Regarding your questions, we will use something in the neighborhood of 6000 to 25,000 pounds per year. I realize that this is a wide spread but that is as precise as I can be right now. We have just purchased 6000 pounds at $1 per pound.

What is the base material of the 4 x 8 or 4 x 10 carbon? Has it been activated? In any case, we would appreciate a couple of pounds of each to test in our laboratory to see if it will serve our purposes.

Sincerely,

Walt Simmons (8/4)

WS:sfm

Walt Simmons
General Superintendent

From the Desk of -
Lloyd W. Taggart

J. Fillerup
A quote for act. carbon from D.C. Norcross - my son in law -
LWT
Teresa - Please send to Walt.
For Summons
tfhf.

1606 INDUSTRIAL RD. LAS VEGAS, NEVADA 89114
TELEPHONE (702) 384-1210

HUSKY INDUSTRIES INC.

ROUTE 1 BOX 275 • DUNNELLON, FLORIDA 32630 • TELEPHONE 904/489-4700

June 17, 1975

Mr. L. W. Taggart
WMK Transit Mix
Box 14697
Las Vegas NV

Lloyd 384-1210 Coconut

RE: SUMMA Corporation

Gentlemen:

Thank you for the opportunity to bid on your carbon requirements.

Uncertainty in the availability and cost of raw materials precludes us from entering into firm contracts. Prices in effect at time of delivery are those quoted on the date the order is received.

With the aforementioned in mind, I am quoting the following prices for immediate deliveries:

Delivered TO	QUANTITY	VIA	FORM	BASE PRICE PER TON	PALLETIZING AND/OR SLIP-OVER PRICE PER TON	FREIGHT PRICE PER TON	DELIVERED PRICE PER TON
Las Vegas	40,000#	Truck	Bags	$310.00	$25.00	$143.00	$478.00
Los Angeles	40,000#	"	"	310.00	25.00	154.00	489.00

PALLETIZING - too fine

All freight increases are to be assumed by buyer. Terms are net 30 days. Shipment is dependent upon product availability at time of order and no guarantees will be made with regard to length of delivery time. There will be no penalties paid by Husky Industries, Inc. for specifications other than those shown on our specification sheet ~~XXXX~~ #713.

I would appreciate hearing from you upon receipt of my quotation.

Respectfully,

D C Norcross

D. C. Norcross
Vice President and
General Manager
Industrial Division

prefer Coconut shells
if no coconut then whatever
+10 – 30 U S S
+12 – 30

3 or 4# sample
or whatever convenient
for testing

DCN,pcf

cc: Billing Dept.

Land Exploration
and Mining Division

Post Office Box 1126
Tonopah Nevada 89049
702 482 3584

A Division of
Summa Corporation

July 2, 1975

Mr. D. C. Norcross
Vice President and
General Manager
Industrial Division
Husky Industries Inc.
Route 1 Box 275
Dunnellon, Florida 32630

Dear Cope:

I have on hand your letter of June 17 to Lloyd Taggard of WMK Transit Mix regarding a quote for carbon for our company.

The type of carbon quoted is too fine for our needs, unfortunately. Could you please quote us a price on -10 +30 or -12 +30 (U. S. Standard screen), preferably carbon from coconut shells. Should coconut shell carbon not be available, then please send us a sample of the nearest comparable carbon in those sizes so we may test it in our laboratory.

For that matter, we would greatly appreciate a sample of any carbon in those size ranges that you make.

We look forward to hearing from you in the near future.

Sincerely,

Walt Simmons

WS:sfm

Walt Simmons
General Superintendent

PRUETT, David L. - Metallurgy Reports 130

HUGHES TOOL CO. 130
METALLURGY
METALLURGY - REPORTS (DAVID PRUETT)

February 1, 1975

Summa Corporation
Land Exploration & Mining Division

Attention: Mr. D. J. Gribbin and Mr. Wlater Simmons

Dear Sirs:

Confirming our telephone conversation, my charges for contract mill construction and metallurgical design are $200 per day plus expenses unless by special assignment.

Truly yours,

David L. Pruett

David L. Pruett
612 East 2nd Street
Winnemucca, Nevada

DATE: February 5, 1975

TO: D. J. Gribbin and Walt Simmons

FROM: Dave Pruett

SUBJECT: Pilot Plant Functions

1. Use as a Sampling Plant: Sampling can be done much more accurately when done in bulk. The larger the sample, the better statistical base. The finer the sampled material is crushed, the more accurate the results of sampling. No sampling is completely unbiased but the mechanical type, if set upright can correct and average most error. The fine bin discharge from this pilot plant may easily be routed outside to a sampler if desired. Capacity: 3 tons per hour minus ¼".

The auto samplers provided will give good head and tail information of the ground and processed pulps.

2. Crushing and Grinding Evaluations: Observations of time, power requirements, size distribution, particle shapes may be correlated directly to requirements of full size machinery.

3. Carbon in Pulp-Cyanide Plant -- Gold and Silver Ores: This pilot plant is a direct copy of the most current gold-milling trend in gold extraction. The newest American plant using this technique is the Homestake Mill, Lead, South Dakota. Three new major plants in South Africa built within the last four years have been carbon in pulp. Ores not generally adaptable to heap leaching or of value too high to risk the losses inherent in heap leaching are often treated economically with carbon in pulp.

Advantages compared to conventional cyanide:

(1) Minimum amount of equipment.
(2) Ores with difficult settling or filtration character may be treated.
(3) Free cyanide need not be maintained in final

stages, a major problem in environmental waste.
(4) Fouled solutions do not retard precipitation.
(5) Low demands on operational personnel, technically and in time required.

This process is not experimental but a well proven method.

4. Flotation: Ores other than gold and silver may be run. Silver ores often treat better in flotation than cyanide. Methods are well established and operational costs low. Also, fine carbon can be treated by flotation and recovered. Cheaper carbons may be tested in conjunction with the cyanide plant. Flotation of carbon plant tails check carbon loss and abrasion property.

5. In-House Test Work: Although outside consultants may be necessary often, the advantages of test work by existing personnel are readily apparent. If a full-scale operation is envisioned many of the bugs can be eliminated by pilot operation. Experiments altering the flow of existing plant machinery can be run first in pilot.

6. Adaptable Ores: The carbon in pulp plant represents the probable flow sheet and treatment of Tonopah ore and the Tonopah Belmont Tailings. Checks can be run on Manhattan ore and others. The flotation plant could process the type of ore from the Lida area and the properties in Mineral County or any flotable ore. By acid proofing the agitators and pumps copper ore could be leach tested. Any bulk samples could be crushed and split.

7. Rates of Treatment:

Sampling	3 tons per hour	(-¼ inch)
Carbon in pulp	15 tons per day	(-35 mesh)
Flotation	10 tons per day	(-65 mesh)

(Higher for softer ores and coarser grinds.)

8. Labor Requirements:

1 crusher man	1 shift/24 hours
1 mill operator	1 per shift

Dave Pruett

SUMMA CORPORATION: TONOPAH, NEVADA

PILOT PLANT COSTS: FLOTATION, CARBON IN PULP

	Material	Labor
Feed Hopper 5' x 9'	$ 600.	$ 200.
Elevator Repair	100.	100.
2' x 4' Screen	600.	250.
6" x 16" rolls	1,500.	250.
Fine Ore Bin	750.	300.
Feeder Installation		150.
Screw Classifier	1,500.	200.
Pump Installation with catch	125.	100.
Reagent Feeders:		
2 Clarkson Wet	500.	30.
1 Dry	150.	80.
Auto Sampler	350.	150.
Conditioners Tanks (3)	1,800.	300.
Conditioners Motors	300.	100.
Carbon Screens (3)	1,500.	250.
Carbon and Pulp Air Lifts	250.	200.
Compressed Air -- 30 cfm.	1,750.	200.
Carbon Feeder	200.	50.
Auto Sampler	350.	150.
Tail Pump Installation	150.	100.
Pan Filter	300.	250.
Pan Filter Vacuum Blower	400.	150.
Flotation Machines	1,750.	200.
Supervision, Engineering, 40 days to complete		6,000.
	$14,925.	$9,860.
Set Amalgamation Unit	250.	250.
Set furnace	400.	250.
Carbon Bin	400.	250.
Drum Deck Wood	600.	300.
	$16,575.	$11,010.
Estimate plus Contingency	$27,575. + 2,425.	
TOTAL ESTIMATE		$30,000.

RL
see also Gr. 26
P3-S

David L. Pruett
612 East 2nd Street
Winnemucca, Nevada
623-5679

May 9, 1975

Summa Corp. Mining Division
P. O. Box 1126
Tonopah, Nevada 89049

Attention: Mr. Dave Gribbin
Mr. Walt Simmons

Sirs:

Since alteration work never seems to be complete for any processing plant, I have a few comments on the work initiated at Tonopah and Manhattan.

Potential Problems -- Manhattan

1. The carbon loading tanks are designed for a flow rate of 20 gpm/sq.ft. or 1000 gpm. The carbon should teeter at about 500 gpm. Lower flow rates will cause poor agitation. Samples should be taken at tank center and side to assemble data for the minimum flow rate acceptable to provide good uniform loading. The dispersal plate in the bottom of the tank has a 6% void ratio (1½" holes on 6" centers) as recommended by the Bureau of Mines. Low flow rates will actuate radially, leaving the sides with less agitation.

2. The carbon on hand, mainly PBC-10/30, may be too fine for use with a slimy solution from a newly loaded pad. The flotable carbon and slime will plug the 30 mesh screens. Constant brushing to keep the screens free would be necessary. 6/16 carbon and 20 mesh screen would help the slime problem. Union Carbide pellet is not available enough to count upon. Pond settling

of slime before introduction to carbon tanks would help.

Tonopah Stripping Plant

1. The pump on the filter may be moved and used as a spare on the preheater. The internal screen may be replaced with a valve and closed when loading.

2. Complete data must be assembled to properly operate this unit.

3. The unit should not be altered until comparison data can be assembled. The time to strip, the costs, and recovery ratio, and bullion grade are probably better than the current practice at Homestake or Cortez. This is a preliminary estimate and must be proved or disproved before radical modification.

4. Stripping time may be reduced by conditioning the carbon in the storage tank with NaOH.

5. Temperature control is critical to stripping in reasonable time. Standard solutions should be made up with NaOH, NaS_2, methanol and water. The boiling and flash points observed.

6. If a decision is made to sell bullion to a broker, such as Simmons Refining, a melting-brokerage fee of 2% is taken regardless of fineness. The silver precipitation can be discontinued and dore bullion shipped direct to refiner.

7. Ideal location of a stripping plant is at the millsite, eliminating transport and handling. Extra solution not completely stripped can be returned to pads. However, the control necessary to run the unit would require an A/A unit at the same location.

8. You should tie to Sierra Pacific Power and use the diesel as standby.

   A spare pump should be installed.

   These should be Clarence's problems to figure out.

David L. Pruett

612 East 2nd Street
Winnemucca, Nevada 89445

February 12, 1976

Mr/ D. J. Gribbin
Summa Corporation
Mining Division
3421 Las Vegas Blvd. South
Las Vegas, Nevada 89109

Dear Dave:

The attached report summarizes my recommendations for the modification of the carbon desorbtion circuit, Tonopah, Nevada.

Sincerely,

David L. Pruett

DLP:sfm

Attachment
cc. attachment: DKH

Xc WJR
WDM

CARBON ADSORPTION-DESORBTION

REACTIVATION - SIZE REQUIREMENTS

Cycle

Adsorption -- Manhattan ± 1600 oz. Au/month

± 50 oz. Au/day

Loaded carbon 1 ton/4 days/200 oz. Au load

Transport Manhattan to Tonopah

2000 lb. Carbon wet (45 lb./cu.ft.)
(Dry: 30 lb./cu.ft.)

100 cu. ft. tank approx. needed
+ weight of tank @ 1000 lb.

2 ton capacity trailer -- cone bottom
with brakes.

Desorbtion Tonopah -- 2000 lb. batch
cycle from transport through stripper
to reactivation with Ag-Au to bullion
form -- complete less than 8 hours.

Reactivation Tonopah: 3 hour preheat

200 lb./hr. furnace limit

2000 lb.: 13 hours, sized and weighed carbon
and assayed; returned in barrels
to Manhattan.

Transport:

The carbon transport tank if placed on a trailer will save a truck from permanent assignment:

One ton of (dry) carbon plus adsorbed water and the tank itself will weigh about two tons. The transport trailer must have brakes coupled to the tow vehicles. The pickups may be used.

Transport tank should be rigged for easy cleandown and fast unloading. Capacity should be about 100 cubic feet.

Desorbtion:

Operational experience of the past year has shown the basic fundamentals of methanol-caustic stripping to be valid but in poor application.

Recent testwork by myself in your Tonopah lab shows that much of the published data on time to desorb gold and silver from carbon is false. With proper agitation, temperature and solution makeup, gold-silver desorbtion is practically instantaneous.

Agitation is the key to fast desorbtion. Fluid agitation is recommended to hold down carbon disintegration; however, a mechanical agitator option is provided in the desorbtion unit plans.

Solution Makeup depends on pressure and temperature limits of the desorbtion vessel, low pressure (-15 lb. psi) and low temperature (-210°F.) require approximately 10% methanol to react well. Operating temperature of 250°F. and pressure of +60 psi would probably not require any alcohol. The requirement of 2% NaOH (caustic soda) is to provide a solvent for the gold and silver cyanide complex and an electrolite for the electroplating unit. If electrowinning is not used, the NaOH level may be lowered.

Precipitation:

In place of the current practice of electrowinning the gold-silver from solution, I recommend a direct precipitation of the gold-silver from solution by the addition of aluminum granular metal to the pregnant solution taken from the reaction vessel. Plus 99% of the Ag-Au is precipitated from an agitated 150° F., pregnant solution as metallic gold and silver. Retention time in precipitation of twenty minutes.

Option/Electrolytic Cell:

In the event the aluminum precipitation reaction causes unforeseen problems, the electrolytic unit cell of the current type may be used.

Data/Present Plant Cell:

15 amps/sq.ft/one basket of steel wool

3 volts DC
3½ gpm flow
200 amp rectifier
cells 13" x 13"/.3 cu.ft./cell/2.2 gal.
Retention time one cell = 30 seconds
10 cells = 3 minutes

Problems: Amperage draw per cell package is not even. Cells with lower grade solution draw more amps (less resistance).

Amperage used is excessive to that needed.

Acid treatment of steel wool is sloppy and time consuming.

The following cell can be used for 50 gpm. flow. See plate #3.

Amperage of 7-8 amps/sq.ft. on this type of vertical steel wool package is ample to electroplate Au-Ag.

Agitated Carbon Desorbtion:

Test #1: 100 grams carbon

loading 141 oz. Au
47 oz. Ag

Solution 10% methanol
2% NaOH
water

Temperature raised to 180°F.
Open beaker agitated by prop.
Tests could not be run to completion.
Methanol evaporation drops, Ag-Au back into carbon.

See plate #1.

Test #2-5: Same pattern as #1.

Ag-Au desorbtion reaction is extremely rapid.

Conclusion: In a pressure vessel, a properly agitated carbon in a caustic methanol solution at +180°F. can be stripped to -5 oz./ton loadings of Ag-Au in less than 1 hour.

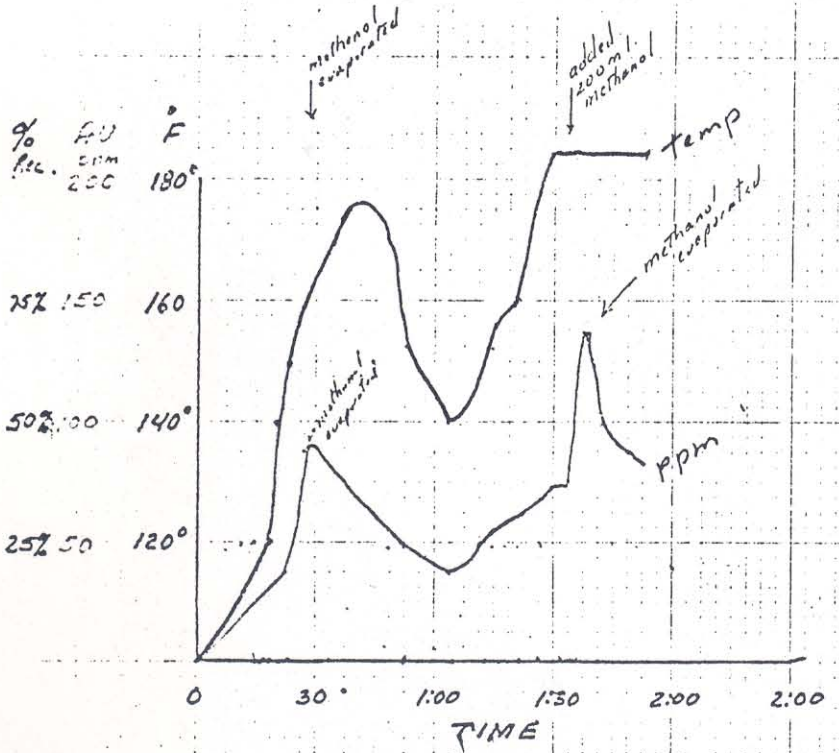
Experiment 100 grams of Carbon Assay 131 oz Au 47 oz Ag.

2000 ml water
200 ml Methenol
40 grams NaOH

Agitated by prop. in beaker on hot plate

100 grams = 3.4T Assay tons
x 131 mg/AT
448.02 mg of Au contained

100% = 443.02 mg / 2200 ml = 203 ppm



Data Result: Carbon strip is not a time critical and dependant system as assumed by USBM. and our prior work.
Gold dissolution is almost instantaneous with the right temp and methenol

Agitated Stripping may be 100% (or close) complete in less than 2 hours.

Dave Pruett
25 Jan 76.

Plate # 1.

Aluminum Precipitation of Ag-Au

Test #6:

Hot 150°F. solutions of methanol NaOH containing gold and silver stripped from carbon were taken from the operational desorbtion circuit.

Au-Ag loadings were determined by both fire assay and atomic absorption. 1000 ml. batches were placed on a hot plate, maintained at 150° F. and agitated by prop.

Aluminum foil, ratio 3/1 of the contained Ag-Au weight, was placed in with solution.

Test #6 Results:

Time	°F Temp.	NaOH%	NaCN lb./t	Ag. oz./t	Au. oz./t	Al.
0	150	1.43	.76	5.71	3.062	3 g.
5	150	--	--		16 ppm*	
20	125	1.57	.38	0.1	.07	--

Precipite contained some free aluminum metal, metallic gold and silver.

Recovery: 97.7% of gold
98.3% of silver

Button fineness: 353 Au

Temperature decrease caused Ag-Au to go back into solution.

* ppm = atomic adsorption assay

Test #7-12: Same basis as Test #6.

4 tests were run with temperature maintained 150° F.

Results and Conclusions: Heads averaged +500 ppm. Au
Tails " - 4 ppm. Au

Aluminum metallic foil precipitated +99% of Ag-Au from 150% caustic methanol cyanide solution in less than 20 minutes.

Excess Al is dissolved and forms a filterable gel. Ag-Au precipitate is metallic and fluxes easily.

Aluminum must be agitated to scrub surface for Ag Au collection. Aluminum must not be too fine or NaOH will react rapidly and consumption is excessive.

Temperature should be maintained.

The reaction consumes cyanide, frees OH radical.

Test #13 and #14:

Aluminum Contamination of carbon:

1000 ml. of a solution of 2% NaOH, 10% methanol, .1% NaCN was heated to 160°F. Aluminum was added to approximate 5% weight of solution.

Aluminum dissolved and formed a gel. Agitation dispersed the gel through the solution. The gel passes fine filter paper rapidly.

100 grams new carbon was added and agitated for 2 hours with the aluminum gel solution. The carbon removed from solution and hung with a separate package of new carbon in an Ag-Au-CN solution.

Result: Loading rates on both carbons were identical.

Conclusion: The aluminum taken into solution in this manner does not affect the loading rate of activated carbon.

Reactivation of Carbon:

The present trommel dewatering of educted carbon from the desorbtion tower to the reactivator furnace storage bin is inadequate. Discussions with Sweco Inc., indicate the 48" unit presently on the furnace discharge would be ideal for the dewatering. A small 18" Sweco would handle the 200 lb./hour/day sizing of furnace discharge.

The reactivator storage tank increase to 2500 lb. capacity may be done with a cone above the leg stands.

Option: If the auger feed unit were replaced with a longer screw section. The storage tank shape and size could be made to approximate the Cortez reactivation unit feeder and storage. See plate 2.

Layout:

The following work pages include recommended tank, filtering, precipitation units for a plant based on this new data and the operating experience of the present desorbtion system.

HEAT CALCULATIONS

Present Desorbtion Unit:

30" diameter x 10' long x 5/16" walls

6" heat exchanger x 10' long

Burner 1.65 gal./hr. @ 135,000 btu./gal. = 222,000 btu./hr.
Electrical strip heat 30 kw. = 102,000 btu./hr.
Total = 324,000 btu./hr.

Temperature raise 60°F to 200°F (140°F/3 hrs.)

Radiant surface area 13 sq.ft./pipe
5 sq.ft./elec. A = 18 sq.ft.

Input heat 324,000 btu./hr.
Radiation loss 140,000 btu./hr. (calculated)

Btu -- applied 18,000 btu./sq.ft. exchange area
-- exchanged 10,000 btu./sq.ft.

Overall coefficient heat transfer = .57

550 gal. of solution + carbon/975,000 btu/1 hr./140°F (rise)
or 1772 btu./gal.

Planned Unit:

Using the heat transfer figures from the present stripper, 1,950,000 btu./140°F (rise)/hr. would be required for a unit with 1150 gal. of solution + 2000 lb. carbon.

The efficiency of this unit will be much greater.
1,950,000 btu. = 14.4 gal. of fuel oil.
3 2.5 gal./hr. burners = 7.5 gal./hr.
2 hours would be required for preheat.
Improved heat exchange and insulation should cut this to estimated 1 hour 20 minutes.

HEAT CALCULATION CONVERSIONS

1 hp. = 0.745 kw. = 42.4 btu./min. = 2544 btu./hr.

1 boiler hp. (bhp.) = 33.475 btu./hr.

1 kw. = 1.34 hp. = 56.88 btu./min. = 3413 btu./hr.

1 btu. = .029 kw./hr.

1 cu.ft. water = 62.4 lb. @ 60°F.
1 gal. = 8.34 lb. water @ 60°F.
1 cu. ft. = 7.48 U. S. gal.

fuel oil = 135,000 btu./gal.

propane = 87,000 btu./gal. liquid

Engineering Representatives and Equpment Suppliers:

The following have been consulted about this project:

Reactor:	Butane Tank Corp.
Agitators, Steam, Meter Pumps:	Duncan Engineering & Equipment Co.
Screens:	Sweco Inc.
Screen, Filtration, Pumping, Screw Feeders:	R. A. Trabert Co. Inc.

DALE C. MATHEWS

3185 EAST WASHINGTON BLVD..
LOS ANGELES, CALIF. 90023
(213) 261-5118

R.A. TRABERT CO., INC.
MANUFACTURERS REPRESENTATIVES
Process Equipment • Chemical Engineers

R. M. Gray

Post Office Drawer "B"
534 W. Manchester Blvd.

Phone (213) 678-4303
Inglewood, Calif. 90301

John R. McLaughlin
17910 Skypark Circle
Box 19099
Irvine, California 92713
Telephone: 714-549-2914
213-944-6256

DUNCAN
ENGINEERING & EQUIPMENT CO.

FLOYD A. LUNDY
Technical Advisor, Vibro-Equipment

SWECO, INC. 6033 East Bandini Boulevard
Los Angeles, California 90051 Phone (213) 726-1177

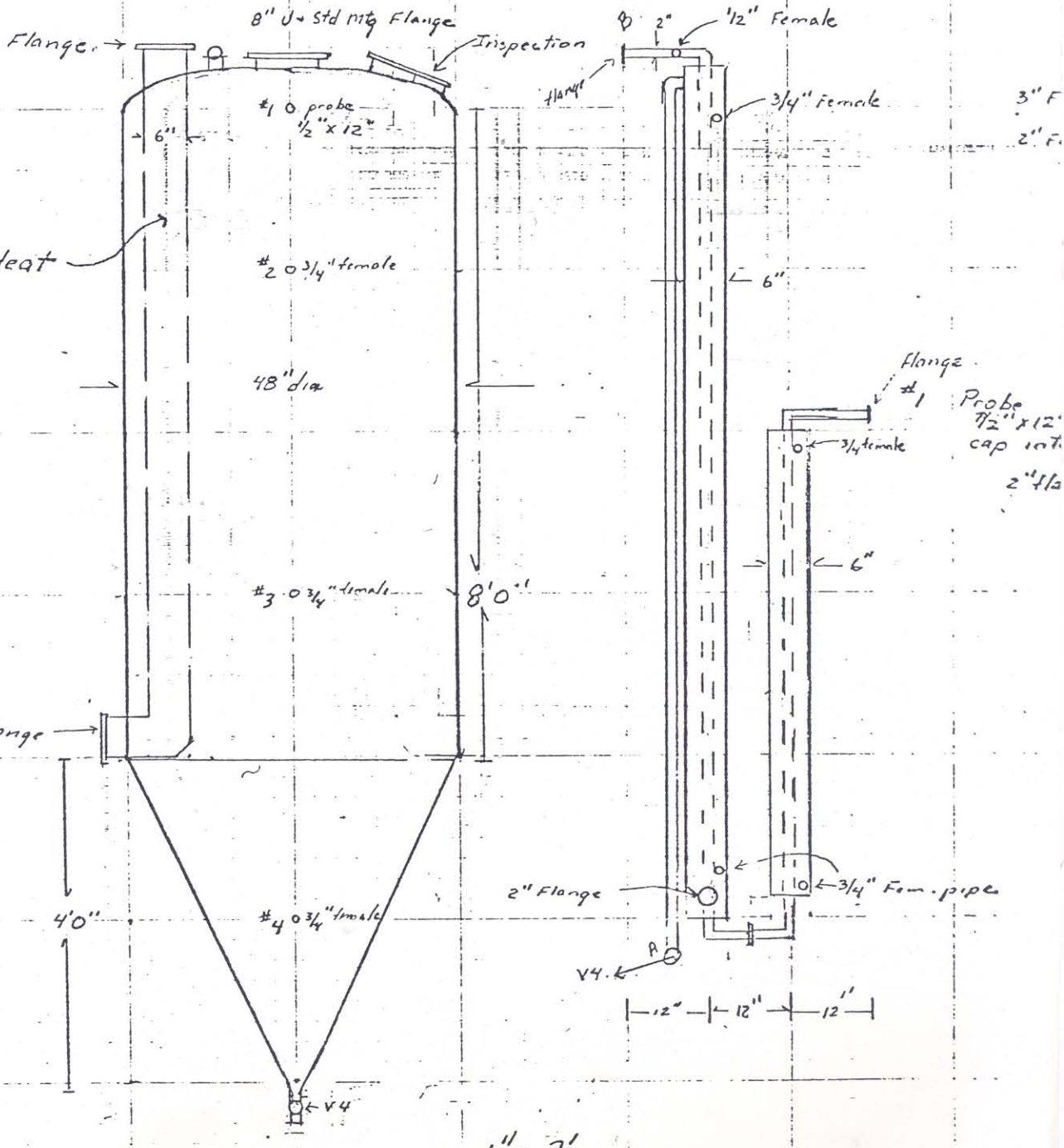
Requirement: 2000 lb. Carbon Strip Unit

48" Dia = 12.56 Cu Ft/F * 8' + Bell = 100 cu ft

48" Dia Cone to 2" - 4' deep (60°) = 24.36 Cu Ft

This unit 125 sq FT Approx - 14.4 cu FT in Heat Pipes = 115 cu Ft

2000 lb Carbon @ 30 lb/sq Ft = 66 Cu Ft.



Summa Corp.

Feb - 1976

Dave Pruett

= 100 cu ft

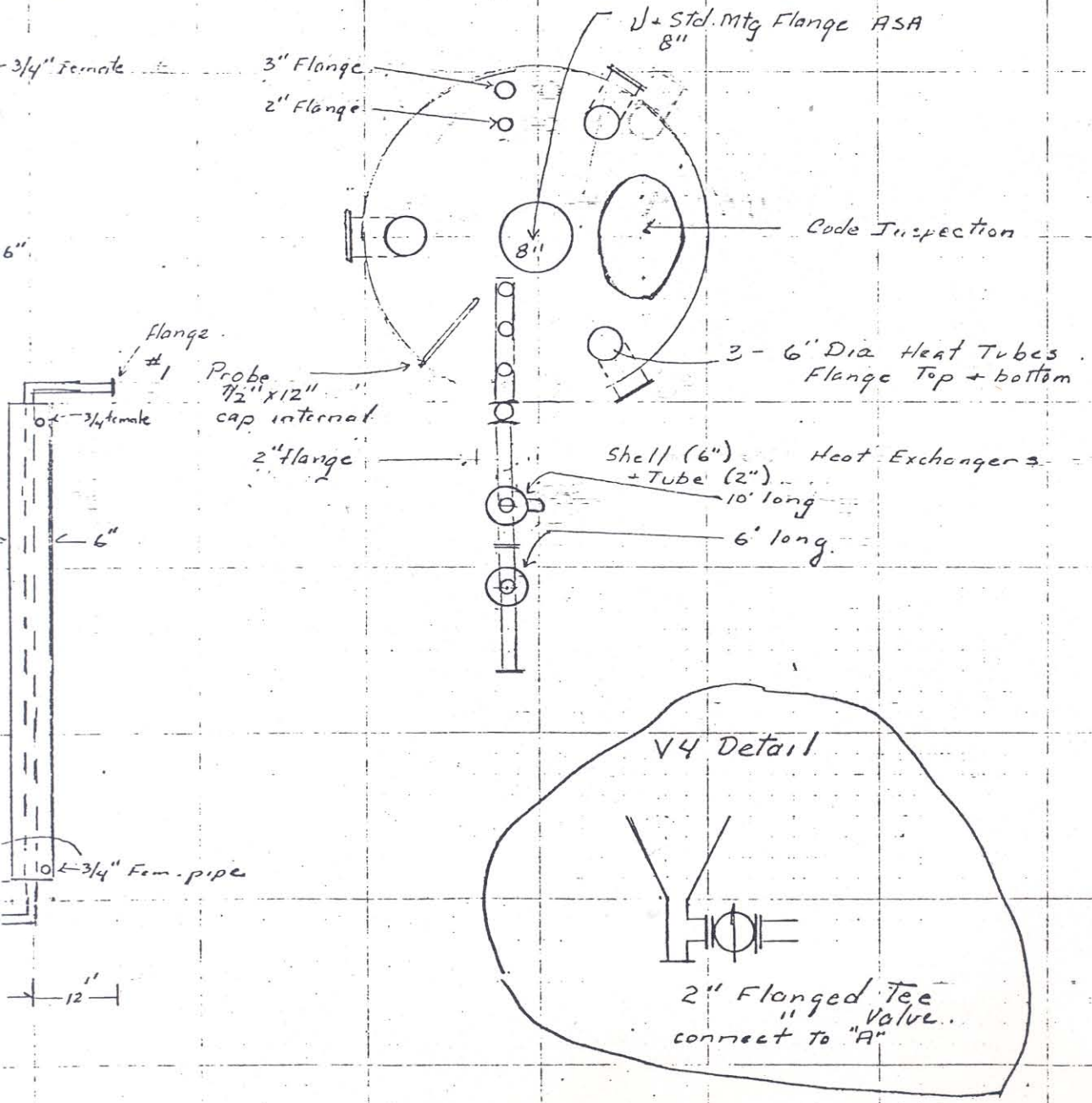
.36 Cu Ft

Pipes = 115 cu Ft

Female

-3/4" Female

6"



BUTANE TANK CORPORATION

3185 EAST WASHINGTON BLVD., LOS ANGELES, CALIFORNIA 90023

AREA CODE 213 261-5118

76-DM-2-10

QUOTATION

• Summa Corporation
• Box 1126
• Tonopah, Nevada 89049

Attention: Dave Pruett

Date: February 10, 1976

Inquiry:

Terms: Net 10th Prox.

F.O.B. Point:

Delivery:

UNITS	DESCRIPTION	UNIT PRICE	TOTAL
	Gentlemen:		
	Regarding your verbal request for quotation, we offer the following for your consideration:		
One (1)	48" O.D. x 8'0" Sm/Sm vertical Carbon Strip Tank for 150 PSI. Shell and semi-elliptical heads 3/8" SA-285-C P.V.Q. material. Vessel to be hydrotested.		
	Estimated shipping weight = 4,500# each Cost F.O.B. our Los Angeles plant =	$7,974.00 each	
	Drawing: 10 days after receipt of order.		
	Delivery: Approximately 5 to 6 weeks.		
	We thank you for the opportunity of quoting on your requirements and if any questions please do not hesitate to contact us.		
	THIS OFFER DOES NOT INCLUDE STATE, FEDERAL, SALES OR ANY OTHER TAX.		

ACCEPTED:

By_____

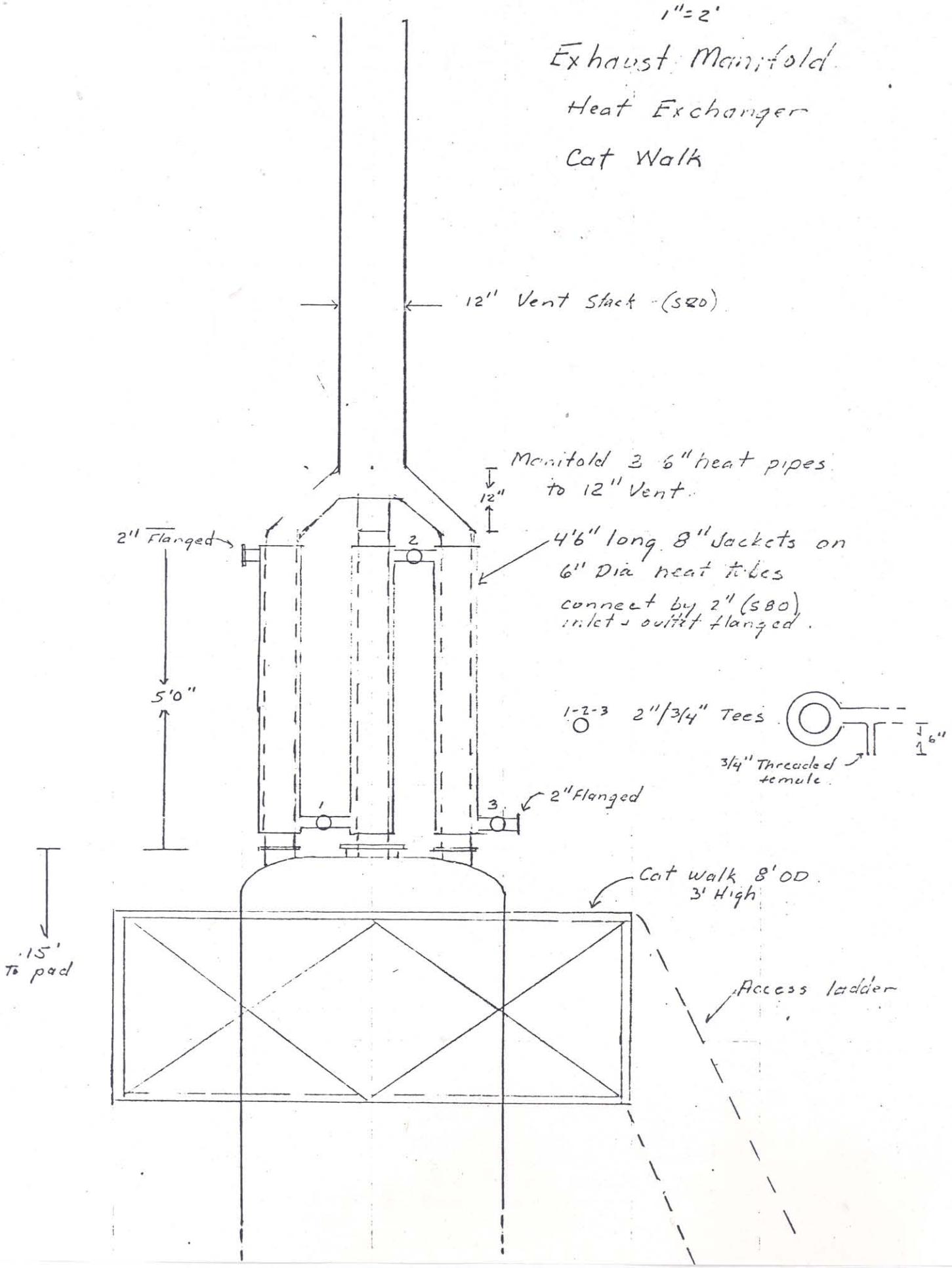
Date_____

BUTANE TANK CORPORATION
Dale C. Mathews/sc
Sales Engineer
By_____

ALL MATERIAL IS SOLD BY THIS CORPORATION SUBJECT TO THE CONDITIONS OF SALE PRINTED ON THE BACK HEREOF.

SUMMA CORP
Tonopah NV
D Project
1"=2'

Exhaust Manifold
Heat Exchanger
Cat Walk



Option: Electrolytic Cell
50 gpi. flow

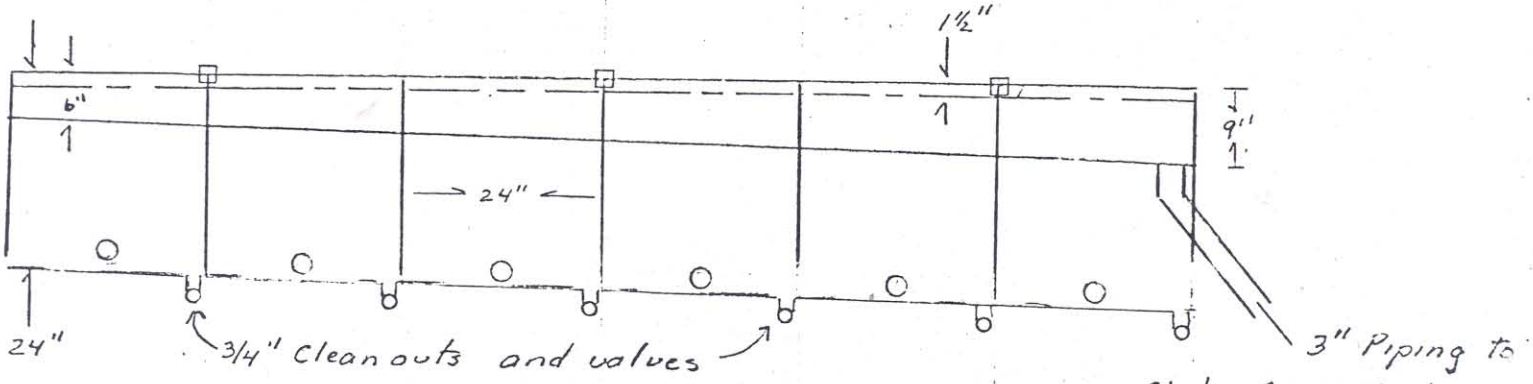
Summa Corp.
Tonopah. Nv.
D Pruett

1" = 2'

□ Buss bars.

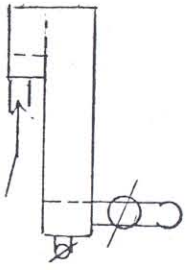
○ 1½ inlet piping

Side View

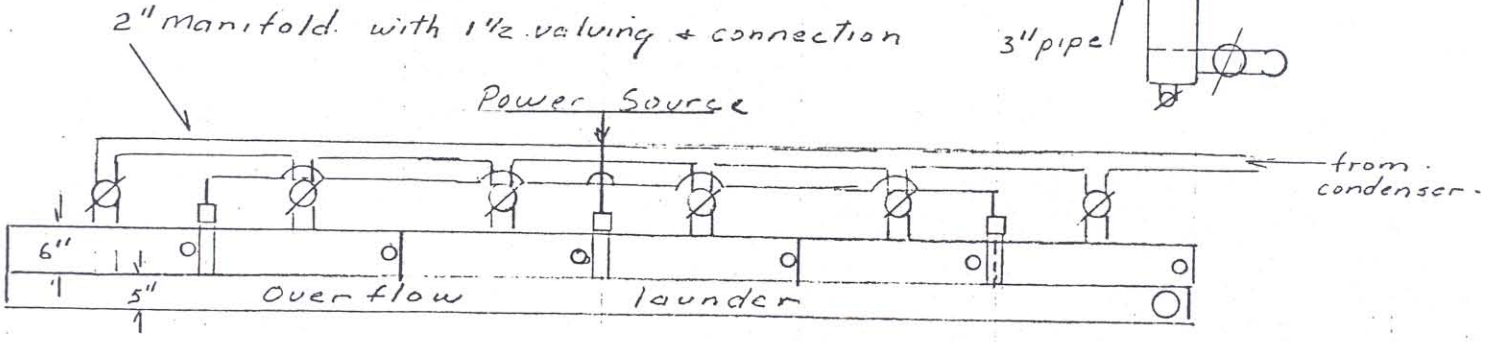


Overflow lip 1½ below cell top

End View



Top View



14 gal/cell Retention @ 8gpm 1min 45sec/cell.
(3 times present unit)

Steel wool Package 20" x 20" x 3"

200 Amp rectifier may be sufficient

Summa Corp.
Tonopah.
D. Pruett.

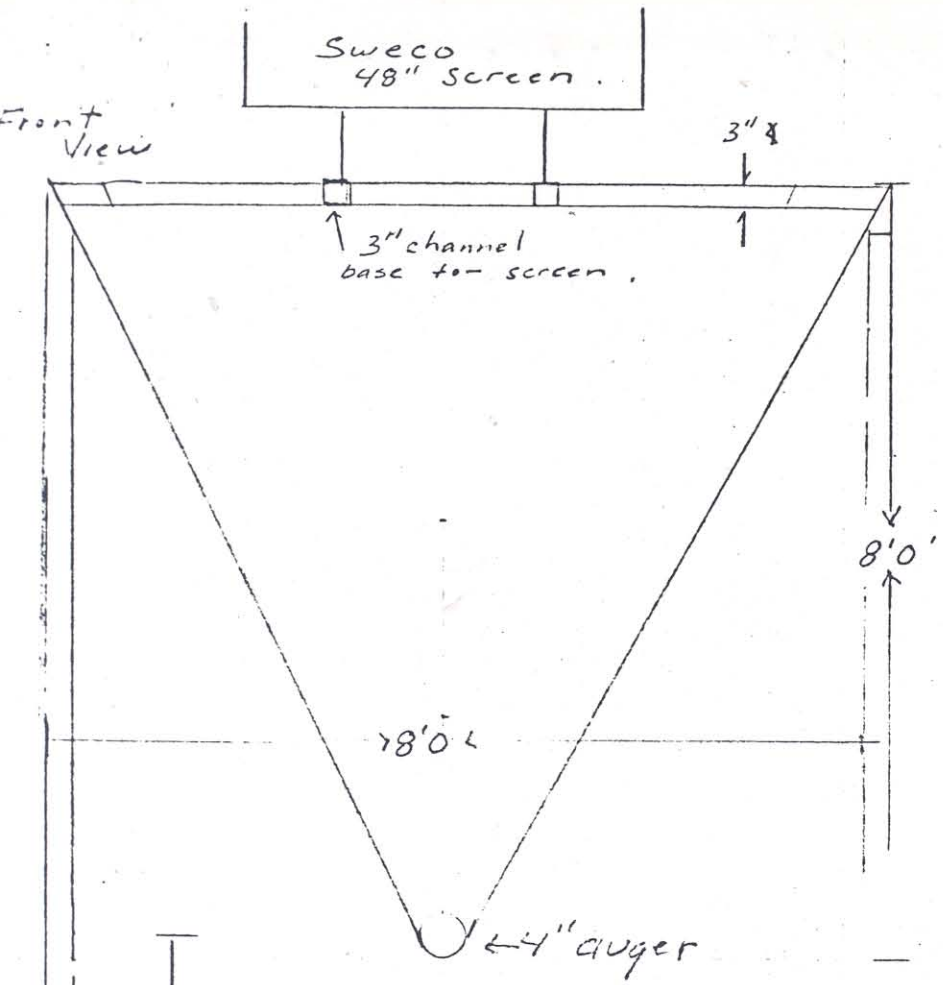
Carbon.

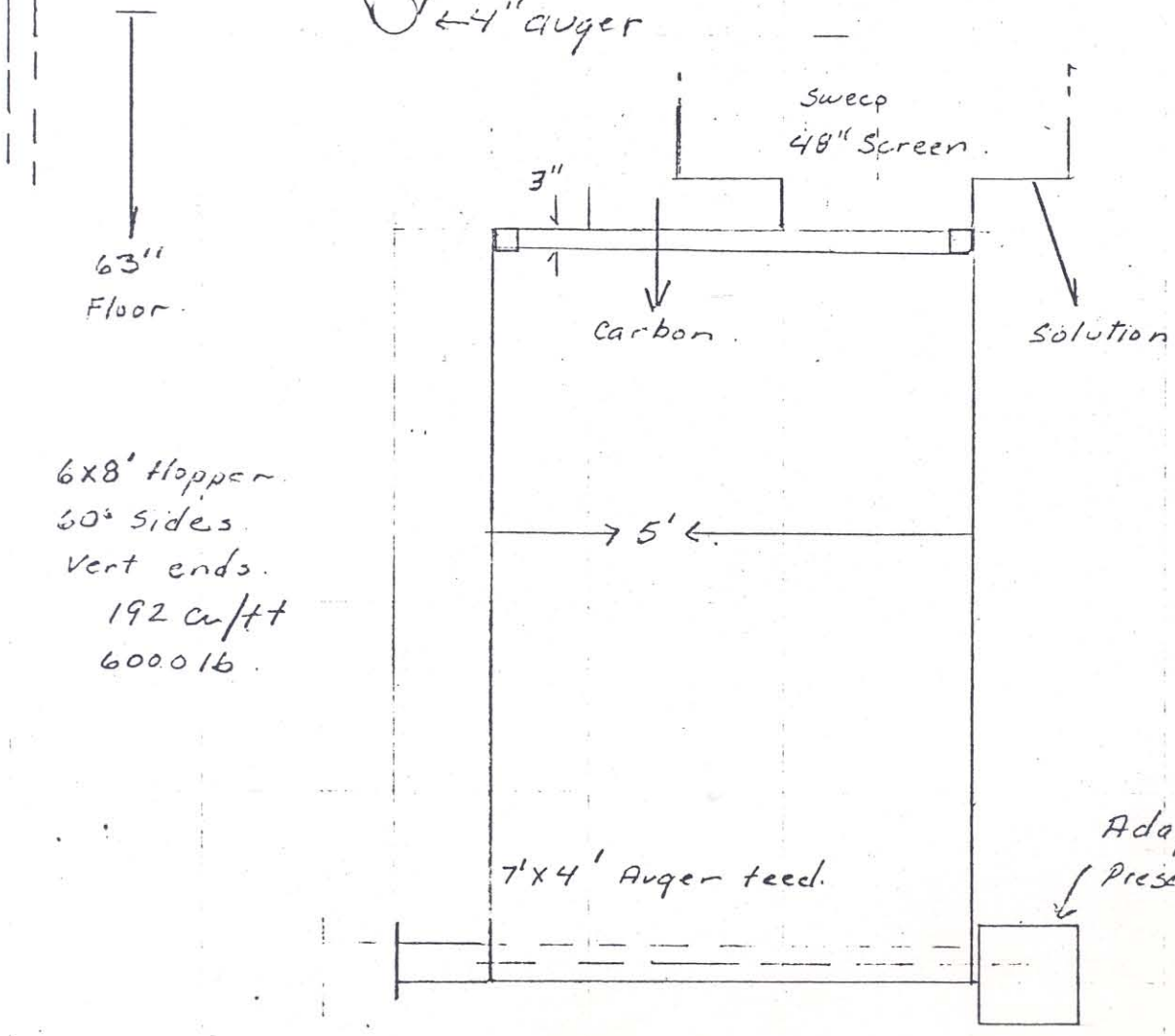
8' Height = 4800 lb.

6' Height = 2700 lb.

4' Height 1200 lb.

New Tank Casing
#1537





6x8' Hopper
60° sides
Vert ends.
192 cu/ft
6000 lb.

R.A. TRABERT CO., INC.

TELEPHONE (213) 678-4303 / POST OFFICE DRAWER "B" / 534 WEST MANCHESTER BOULEVARD / INGLEWOOD, CALIFORNIA 90301

February 10, 1976

Mr. Dave Pruitt
Summa Corporation
Post Office Box 1126
Tonopah, Nevada 89049

Dear Mr. Pruitt:

In answer to your telephone request of February 9, we are pleased to quote on Broughton Filters as follows:

One Model DUO 670B System - in carbon steel construction as illustrated on attached Bulletin F-4. Filter elements to be spiral wound design with .003" opening. Teflon seals. Six 2-way valves included.

Total price: $ 1,580.00

Alternate:

One Model DUO 670C System - as above.

Total price: $ 1,730.00

Prices are f.o.b. Glens Falls, New York.
Terms are Net 30 days.
Delivery would be 6 - 10 weeks.

The Model 770 Filter is also available in a dual arrangement as shown in Bulletin F-75, Diagrahm B, Page 3. Prices for equivalent sizes are approximately $100 higher than for the 660 models.

Let me know if you need anything further.

Very truly yours,

R. A. TRABERT CO., INC.

Robert A. Trabert

Robert A. Trabert, P.E.

RT:mj

Enclosure

Individually
770B #415 + 80 spiral screen. #20 ea filter-Teflon
1280
#515.00 per filter

(#600 ea. for piping)

LIGHTNIN
MIXERS AND AERATORS

MIXING EQUIPMENT Co., Inc.

135 Mt. Read Blvd. • P.O. Box 1370 • Rochester, New York 14603
Phone Area Code 7[illegible]

Address reply to: 9519 Teleg
Phone: 213

DUNCAN ENGINEERING & EQUIPMENT CO.
ENGINEERING REPRESENTATIVES
17910 Skypark Circle
P. O. Box 19099
Irvine, California 92713
Phones: 714-549-2914
213-944-6256
Telex 69-2489

DATE February 3, 1976

TO SUMA
3421 Las Vegas Blvd. South
Las Vegas, Nevada 89019

ATTN: Mr. Dave Pruitt

SUBJECT: Agitator for Fine Carbon Particles

YOUR REF. NO.
MIXCO REF. NO. 36-76-0027
DUNCAN REF. NO. 76-13379

ITEM NO.	QUANTITY	
1	1	MODEL N33G-200 LIGHTNIN

SPECIFICATIONS:

MOTOR H.P. 2 R.P.M. 1150 ENC. TE
Std. VOLTS 230/460 CYCLES 60 PHASE 3
SHAFT DIAMETER 1-1/2" MATERIAL C.S.
SHAFT LENGTH FROM MOUNTING BASE 120" SHAFT R.P.M. 233
IMPELLER NO. 3 DIA. 10.5" TYPE A-100 MATERIAL C.S.
STABILIZER _____ FINS ☐ _____ RING ☒ _____ MATERIAL C.S.
STEADY BEARING _____ MATERIAL _____ BUSHING MATERIAL _____
STUFFING BOX ☒ TYPE _____ WATER COOLED ☐ MATERIAL Teflon
MECHANICAL SEAL ☐ TYPE _____ WATER COOLED ☐ MATERIAL _____
ANGLE RISERS ☐ NOZZLE MOUNTING ☒ NOZZLE SIZE 8"
TEMPERATURE 250°F PRESSURE 50 PSIG
SERVICE CONDITIONS FOR PACKING OR SEAL _____
BAFFLES _____ SIZE 4 "WIDE NO. 4 OFF WALL DISTANCE _____
MIXER MOUNTING _____ ANGULAR OFF CENTER ☐ CENTRAL WITH BAFFLES ☒

SPECIAL DETAILS

TANK SPECIFICATIONS:

HORIZ. ☐ VERT. ☐ ROUND ☐ RECT. ☐ DIA. 48" STR. SIDE 108"
TOP 8" BOTTOM 8" OVERALL INC. MOUNTING 130"
BEAM HGT. _____ NOZZLE HGT. _____

SERVICE

To provide gentle suspension of coal particles

See attached terms and conditions of sale.

SHIPMENT 4 WEEKS*
APPROXIMATE WGT 320 LBS.
APPROXIMATE FRT. $20.32/cwt

PRICE $1,699.00 EACH

PRICES ARE F.O.B., FACTORY, ROCHESTER, NEW YORK TERMS NET 30 DAYS

BULLETIN NOS.
DRAWING NOS. L-16120, DS-T-71

*AFTER RECEIPT OF FORMAL ORDER OR FINAL APPROVED DRAWINGS. THIS IS STANDARD SHIPPING SCHEDULE.

SIGNED J. L. Wood/lb
J. L. Wood/lb
MIXING EQUIPMENT CO., INC.
DUNCAN ENGINEERING & EQUIPMENT CO.
AUTHORIZED REPRESENTATIVE

MIXING EQUIPMENT CO., INC.

CONDITIONS GOVERNING PROPOSALS

1. GUARANTEE.

We guarantee every LIGHTNIN Mixer to do the job for which we recommend it. We also guarantee the materials of which the Mixer is constructed to be as specified by the Buyer.

If the Mixer does not do the job for which we recommend it, we will provide without additional charge a Mixer which will do the job or refund the purchase price. We will repair or replace, at our option and at our expense, any part of the Mixer which contains defective material or workmanship within twelve months of first installation or within eighteen months of shipment from our factory, whichever period is shorter.

Although we will pay all transportation charges in connection with our obligation to repair or replace a Mixer, we cannot be responsible for removal, loading, installation, or similar expenses, nor for any special, indirect or consequential damages. There are no other guarantees or warranties, expressed or implied.

2. DELIVERY.

We will make every effort to meet the delivery date specified in the contract but we shall not be liable for delay which is beyond our control or which is due to an accident to our plant or equipment; riots; war or national emergency; labor disputes of every kind, however caused; embargoes; non-delivery by suppliers; delays of carrier or postal authorities; or governmental restrictions, prohibitions or diversions.

Our delivery of equipment to a common carrier or postal authorities f.o.b. Rochester, New York, shall pass title and risk of loss to Buyer.

3. PRICES.

Our prices do not include customs duties and sales, use, excise, retailer's occupation and/or other similar taxes payable by reason of this transaction.

4. PATENTS.

We agree to defend any claim, action or suit that may be brought as a result of this contract against Buyer for an infringement of any United States patent except a process patent. We also agree to pay such costs and damages as may be assessed against Buyor in such infringement proceeding. Buyer agrees that it will notify us promptly in writing and give us authority and assistance in the defense of the infringement proceeding.

If equipment supplied under this contract cannot be used by Buyer because of an injunction resulting from the infringement proceeding, we agree (a) to procure at our expense for Buyer the right to continue using the equipment or (b) to replace the equipment supplied with non-infringing equipment or (c) to refund the purchase price of the equipment and any transportation or installation costs incurred by Buyer in connection with such equipment.

This paragraph states our entire liability for patent infringement by our equipment.

5. ACCEPTANCE.

This proposal is not an offer. Buyer may within thirty days make this proposal the basis of an order which is subject to acceptance by us at Rochester, New York. Our terms and conditions shall apply to Buyer's order and if there is any inconsistency between our terms and conditions and that order, our terms and conditions shall prevail. Any contract arising out of our acceptance of Buyer's order shall be construed according to the laws of the State of New York.

6. AUTHORITY TO VARY CONTRACT.

None of our employees, agents or representatives has authority to make any agreement or representation which is not contained in this written contract.

QUOTATION TERMS

Shipment: 4 weeks from receipt of order.

Terms: Net 30 days, F.O.B. Rochester, N.Y. "Price(s) are firm for acceptance within 30 days from date of quotation."

Price and terms and conditions are subject to revision if manufacture and shipment is not released at time of order placement.

Purchase orders should be made out to:

Mixing Equipment Co., Inc.
c/o Duncan Engineering & Equipment Co.
Post Office Box CN
Irvine, California 92664

January 29, 1976

MACHINERY and EQUIPMENT CO.
INCORPORATED
P. O. BOX 3132 • SAN FRANCISCO • CALIFORNIA 94119
(415) 467-3400
TOLL FREE (800) 227-4544 • CALIFORNIA (800) 792-2975

Summa Corp.
P.O. Box 1127
Tonopah, NV 89049

Attention: David Pruitt

Dear Dave:

I would like to confirm the availability of the two reactors which I discussed with you on the telephone.

They are described as follows:

Reactor, 1,000 gallon, stainless steel, dimple jacket, turbo agitator, 100 PSI internal, 150 PSI jacket. Price, $8,000, Southern California location.

Reactor, 750 gallon, 347 stainless steel, resin kettle type, equipped with hot oil jacket, full vacuum internal, 54" diameter x 7' on the straight side, 18" double turbo agitator, 10 HP explosion proof motor to Falk gear reducer, 2-3/4" shaft center, 4" bottom outlet, 18" top manhole, site and light glass at top, 6", 2" & 4" top nozzles, equipped with a 12" diameter x 7' long shell and tube condensor. Price, $5,000, FOB our Los Angeles stock.

If you are interested in any of the above and would like more information or would like to set up an inspection, please feel free to contact me. I look forward to talking with you soon.

Very truly yours,

Michael R. Ebert
Sales Engineer

MRE/vf

QUOTATION

PLEASE REFER TO THIS NUMBER ON ALL INQUIRIES

NEVADA TANK & CASING

2500 DICKERSON ROAD • P. O. BOX 5206 • PHONE: 329-0866
RENO, NEVADA 89503

QUOTE NUMBER
Q- 1481
QUOTE DATE
11-26-75
F.O.B. POINT
Our Shop

QUOTE TO: Summa Corporation
P. O. Box 1126
Tonopah, Nevada 89049

ATTENTION: William O. Mollison

BID DATE:

JOB: Carbon Storage Tank

LOCATION:

OWNER:

A.I.A.:

M.E.:

QUANTITY	DESCRIPTION	PRICE	PER	AMOUNT	
1	Open top carbon storage tank, 60" dia. x 96" shell height with 45° cone bottom, supported on four 3" x 3" x 3/8" angle iron legs, per sketch attached. Primer coated exterior only.			1,232	00
	Sales Tax			43	12
				1,275	12

SUMMA CORPORATION
MINING DIVISION
Tonopah
DEC 1 1975
RECEIVED

RECEIVED
SUMMA CORPORATION
MINING DIVISION
DEC 4 1975

original DOG ✓
copy WOM
Tonopah files

TERMS % 10TH PROX., NET 25TH PROX.

QUOTES ARE HONORED FOR 30 DAYS ONLY FROM QUOTE DATE.
PRICES DO NOT INCLUDE ANY FEDERAL STATE LOCAL OR SALES TAXES.
DELIVERY SCHEDULES ARE ESTIMATED. SELLER IS NOT RESPONSIBLE FOR ANY FAILURE TO MEET SAID SCHEDULES DUE TO ANY CAUSE BEYOND SELLER'S CONTROL.

BY: Bill Walker
Bill Walker
TITLE: Manager

CUSTOMER COPY

612 East 2nd St.
Winnemucca, Nevada 89445

February 26, 1976

Mr. David K. Hamilton
Summa Corporation-Mining Division
P. O. Box 1126
Tonopah, Nevada 89049

Addenda:
Carbon Circuit Modification
dated 2-12-76

Dear Dave,

I may have created some misconception in the desorbtion circuit plans by not including more data on the electroplating option. Electrowinning was not intended to be shown an equally viable process to the aluminum precipitation. The following should be added to the report.

1. The time to desorb cycle as shown on page 3 does not apply to an electrowinning circuit. Even with a large cell and more time in electrical contact, as shown, the solution returns to the reactor with ± 25% of the overflow ppm. This leaves the kinetics of Au-Ag disassociation from the carbon in poor balance.

The cell used at Cortez has this problem as do all electrolitic units in current use. To terminate desorbtion cycle, Cortez returns 50 ppm Au solution to their cyanide circuit. This cannot be done at Tonopah.

2. Time to complete desorption with normal electrowinning will be in the ± 10 hour range for this unit as shown on plate 3. Add 12 hours to complete acid treatment and melting. This will take about ± 24 work hours.

3. An aid to this time factor and substitute for the Cortez method of dumping solution into the cyanide circuit would be to install a carbon adsorption cone in the building to strip low grade solution.

4. The wire baskets in this unit will contain approximately 10 times as much steel wool per cell as the current unit. This means a volume of approximately 15 gallons of compressed wool will have to be treated per 2,000 pound batch. This is in comparable ratio to the Cortez unit and the existing Summa Corporation unit. Approximately 20 gallons of acid will be required; 10% sulphuric is recommended for this unit rather than hydrochloric (avoid AgCl).

5. As shown by the recent health problems, the operation of the bullion room has been taken much to lightly. The acid treatment of a liquid containing cyanide is extremely dangerous. Larger volumes increase this risk. This chemical reaction produces cyanic gas and is instantly fatal if breathed. This reaction in an open room with doubtful exhaust over the immediate reactive pots could lead to a genuine tragedy.

Accidental breakage of glass acid bottles in the room pose real danger. The physical handling of the wool-cyanide caustic may lead to long term cumulative poisioning.

Dave, as long as you understand the problems and responsibilities of controlling this type of circuit, I have no objection to installing it. However, I feel an obligation to emphatically point out these potential problems.

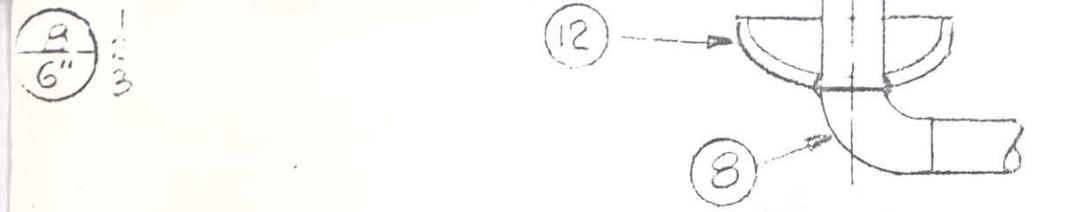
PLEASE initial a copy of this and return to me.

Sincerely,

D. L. Pruett

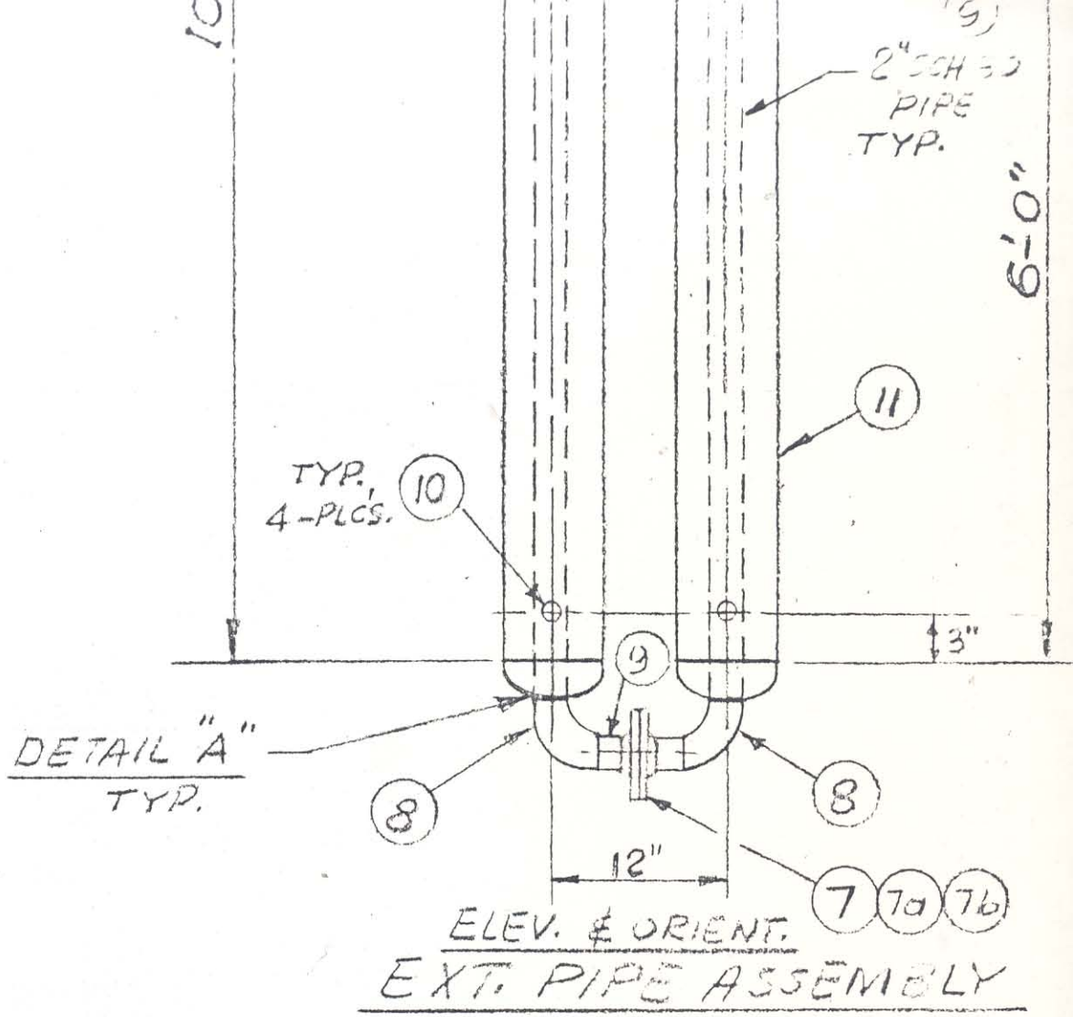
DLP:ln

xc: D. J. Gribbin



DETAIL "A"
TYP. (4) - PLC'S.

O.D., SOLID R
S ON 9½ ϕ B.C.
@ CTR. OF FLG.



ELEV. & ORIENT.
EXT. PIPE ASSEMBLY

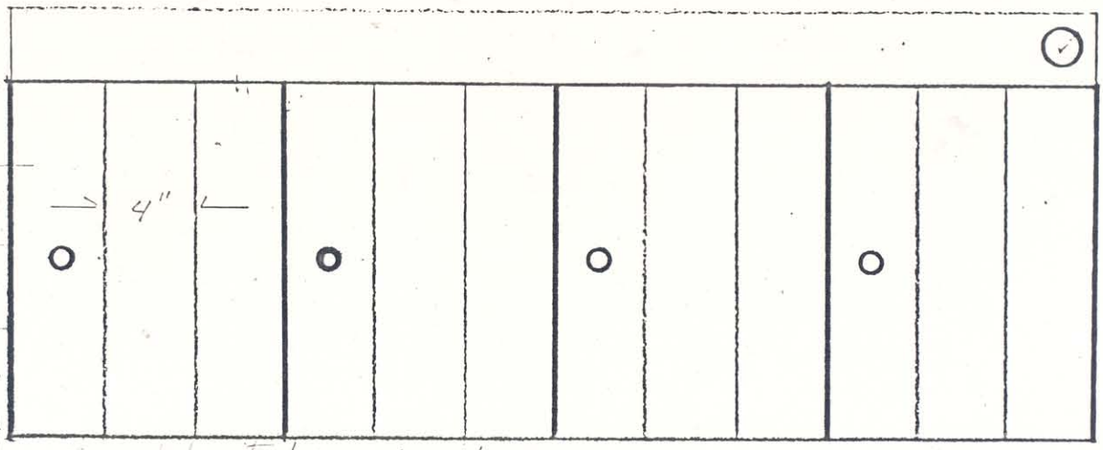
BUTANE TANK CORPORATION
LOS ANGELES, CALIFORNIA, U.S.A.

CARBON STRIP TANK

MAX. W.P. 150 P.S.I.	U	MFR'S W.O. No. 0677
TEMP. 650 °F		MFR'S SER. No. B-752
SHELL THICK 3/8" COND 7/16 IN.	W	OWNERS SER. No. —
HEAD THICK 3/8 MIN. IN.	CODE STAMP	STATE SER. No. —
SHELL C.A. 0 IN. HEAD C.A. 0 IN.	— —	DATE BUILT 1976

D-2 MANIFOLD

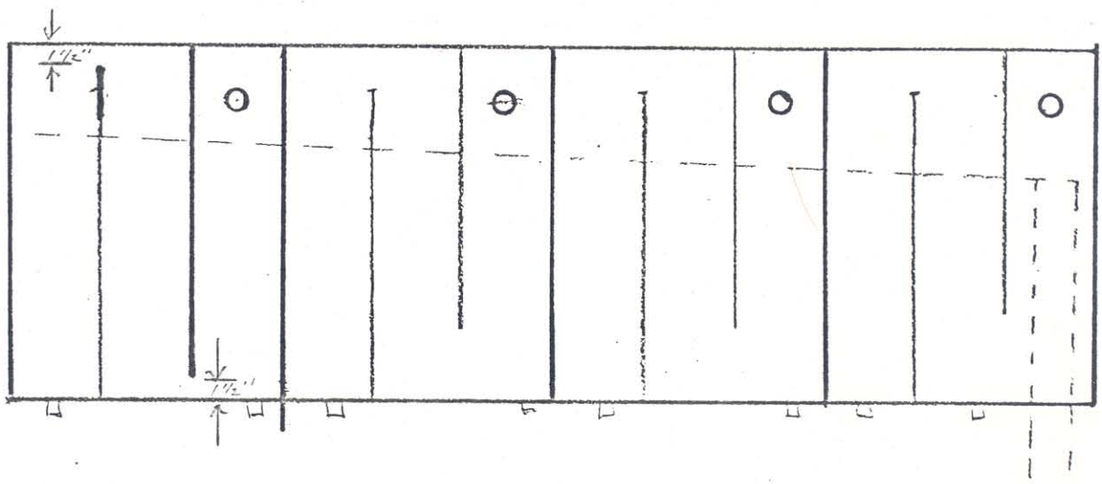
MK	NO. REQD.	SIZE	RATING	FACING	TYPE	BORE	SERVICE	NOZZLE WALL	NOZZLE O.D.	REINF. TH.	REINF. O.D.	
M	1	11"x	15"	—	—	—	MANWAY	—	—	—	—	
E	1	3"	150#	R.F.	S.O.	—		.300"	3½"	3/8"	8"	
D	3	2"	150#	R.F.	S.O.	—	INLETS	.218"	2⅜"	—	—	
F	1	8"	150#	R.F.	S.O.	—	MTG. FLG	.322"	8⅝"	3/8"	15	
B½	6	6"	PLATE	FLGS.			HEAT TUBES	.432"	6⅝"	3/8"	12"	W/
A½	2	2"	150#	R.F.	W.N	SCH 80	OUTLET	.218"	2⅜"	—	—	W/

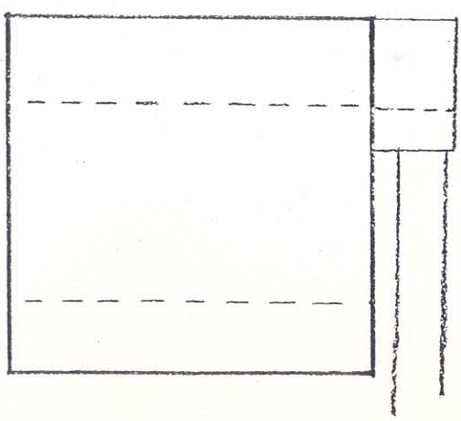


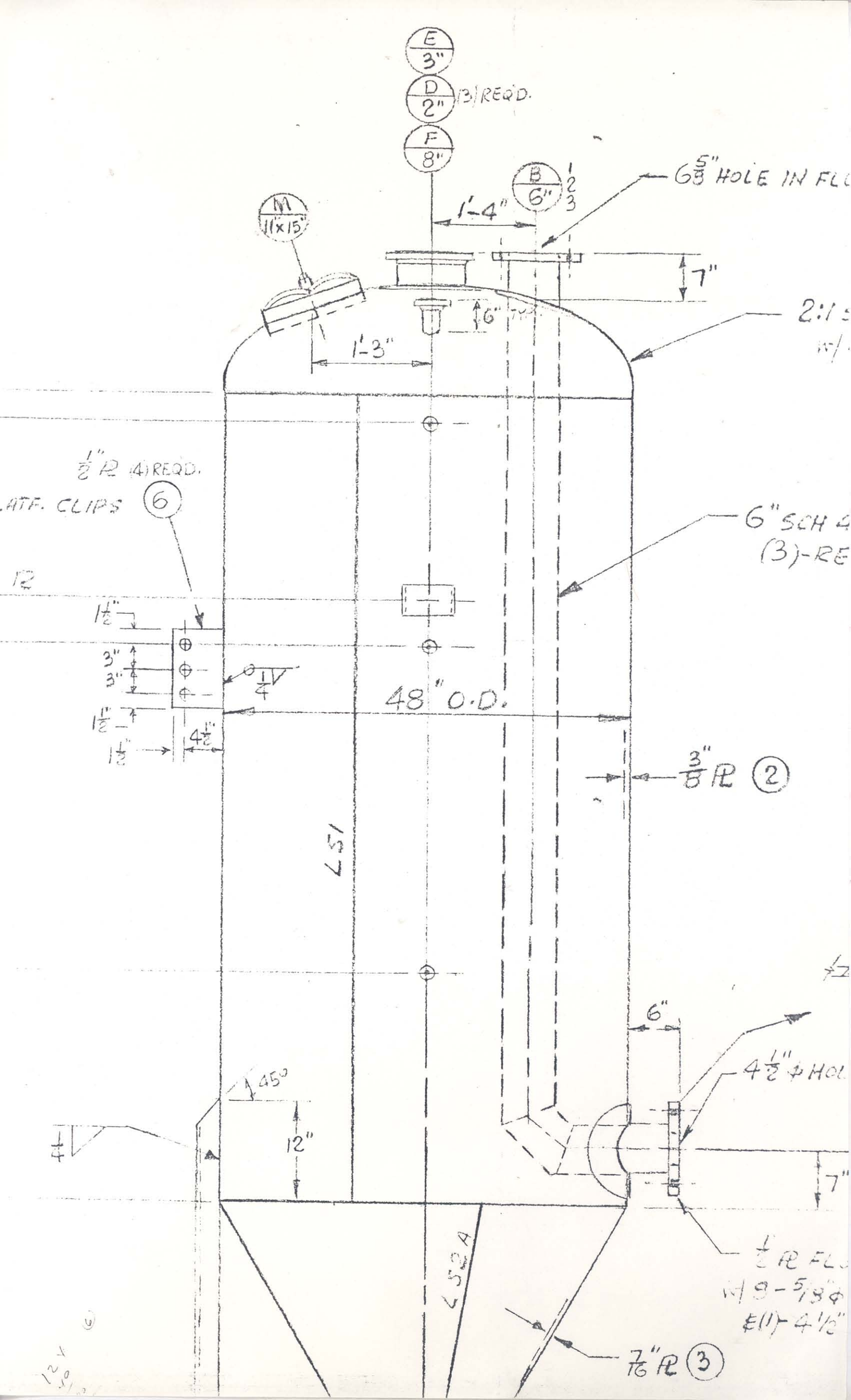
Current density / 3V - 8amp/sq ft

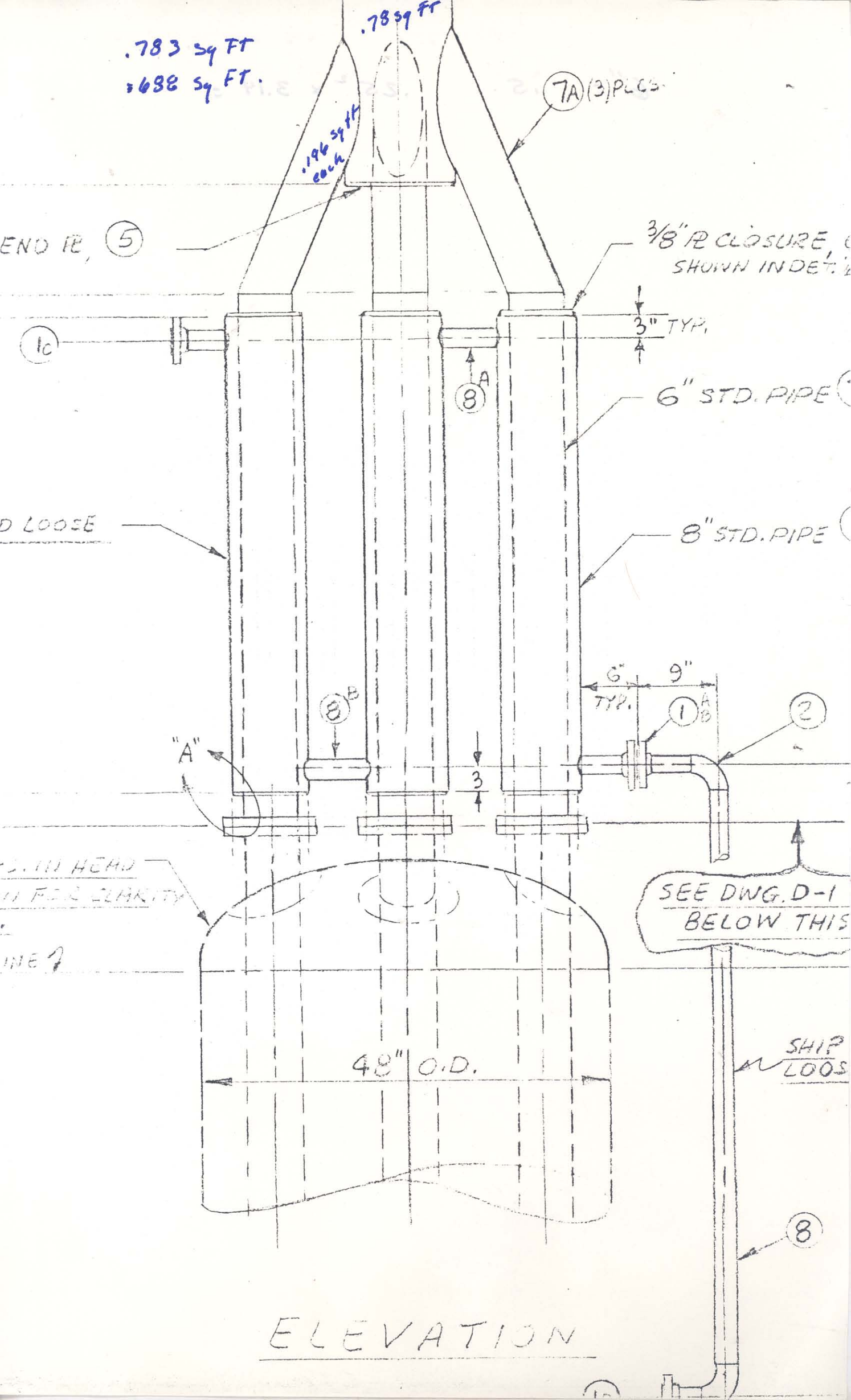
Widen cell to 6" to cut vert velocity

+ increase time

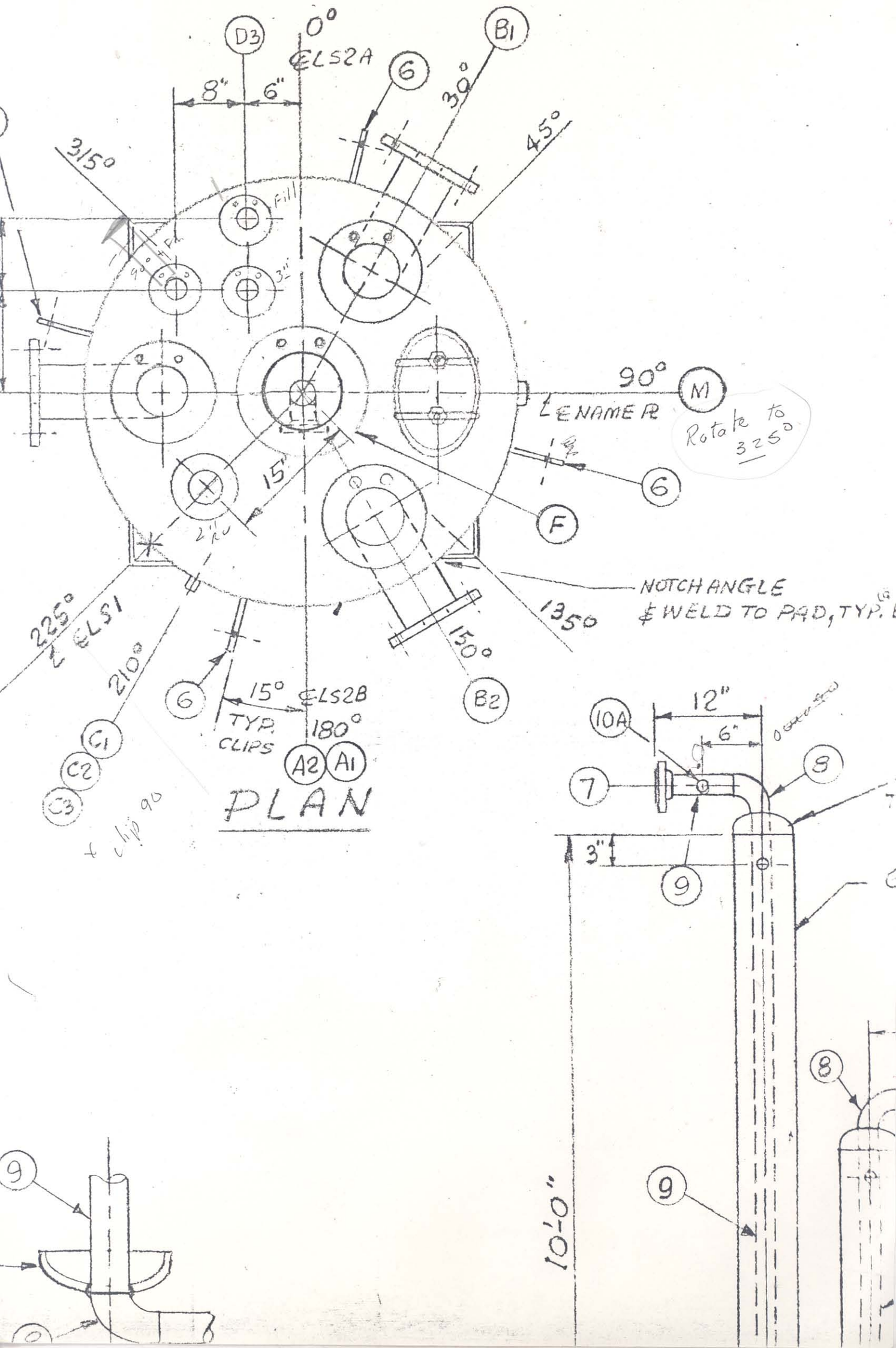


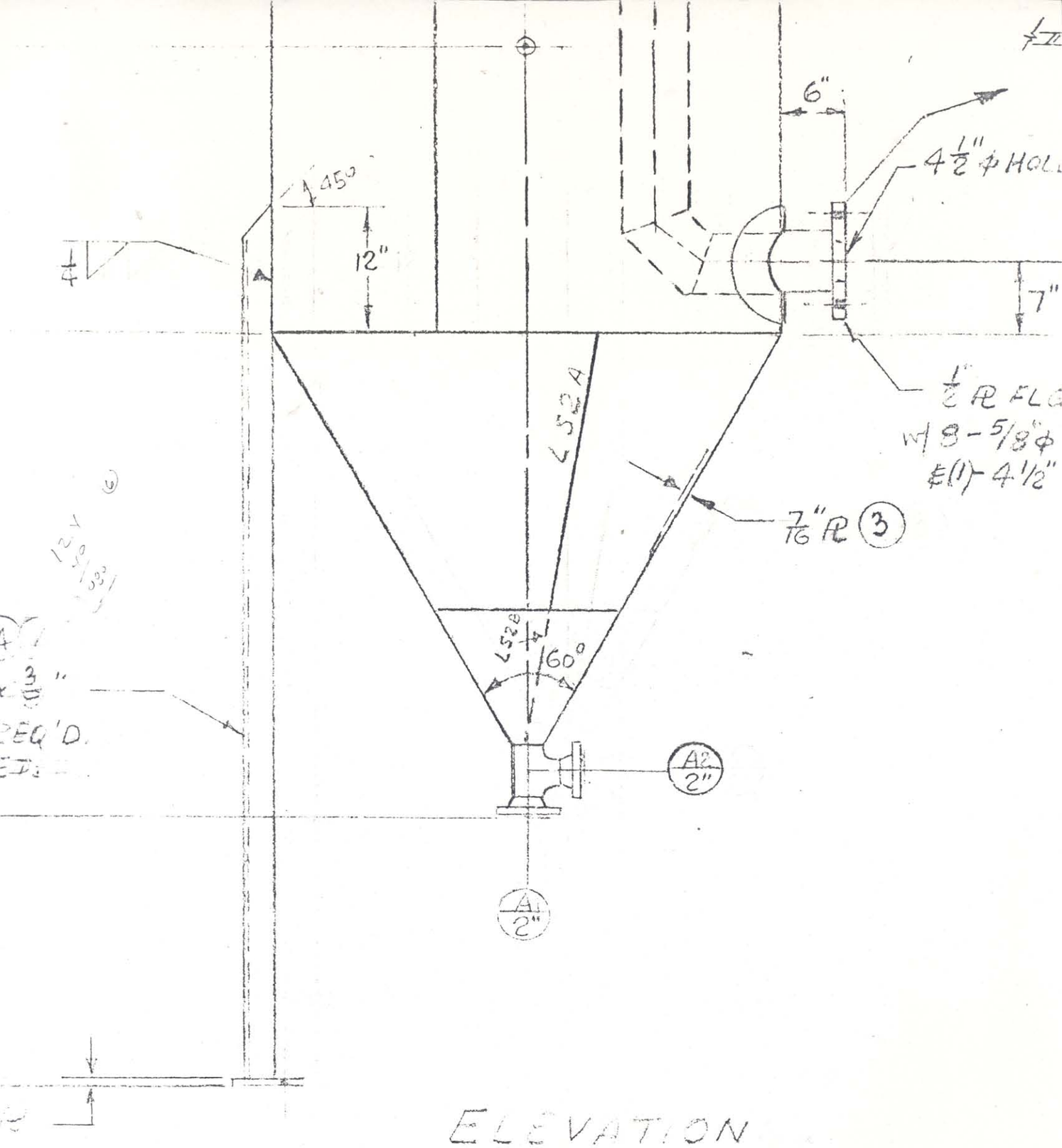



E
3"
D
2"
(3) REQ'D.
F
8"
B
6"
1
2
3
M
11"x15"
1'-4"
6 5/8" HOLE IN FL
7"
6"
1'-3"
2:1
1/2" R (4) REQD.
ATF. CLIPS
6
6" SCH
(3)-RE
1 1/2"
3"
3"
1/4
1 1/2"
4 1/2"
1 1/2"
48" O.D.
3/8" R (2)
L 51
6"
4 1/2" ø HOL
45°
12"
1/4
7"
L 52 A
1/2 R FL
(4) 9-5/8 ø
E(1)-4 1/2"
7/16" R (3)


.783 Sq FT
.698 Sq FT.
.783 Sq FT
.196 Sq FT each
7A (3) PLCS.
3/8" PL CLOSURE
SHOWN IN DET.
3" TYP.
6" STD. PIPE
8" STD. PIPE
6" TYP.
9"
"A"
3
SEE DWG. D-1
BELOW THIS
48" O.D.
SHIP
LOOS
ELEVATION

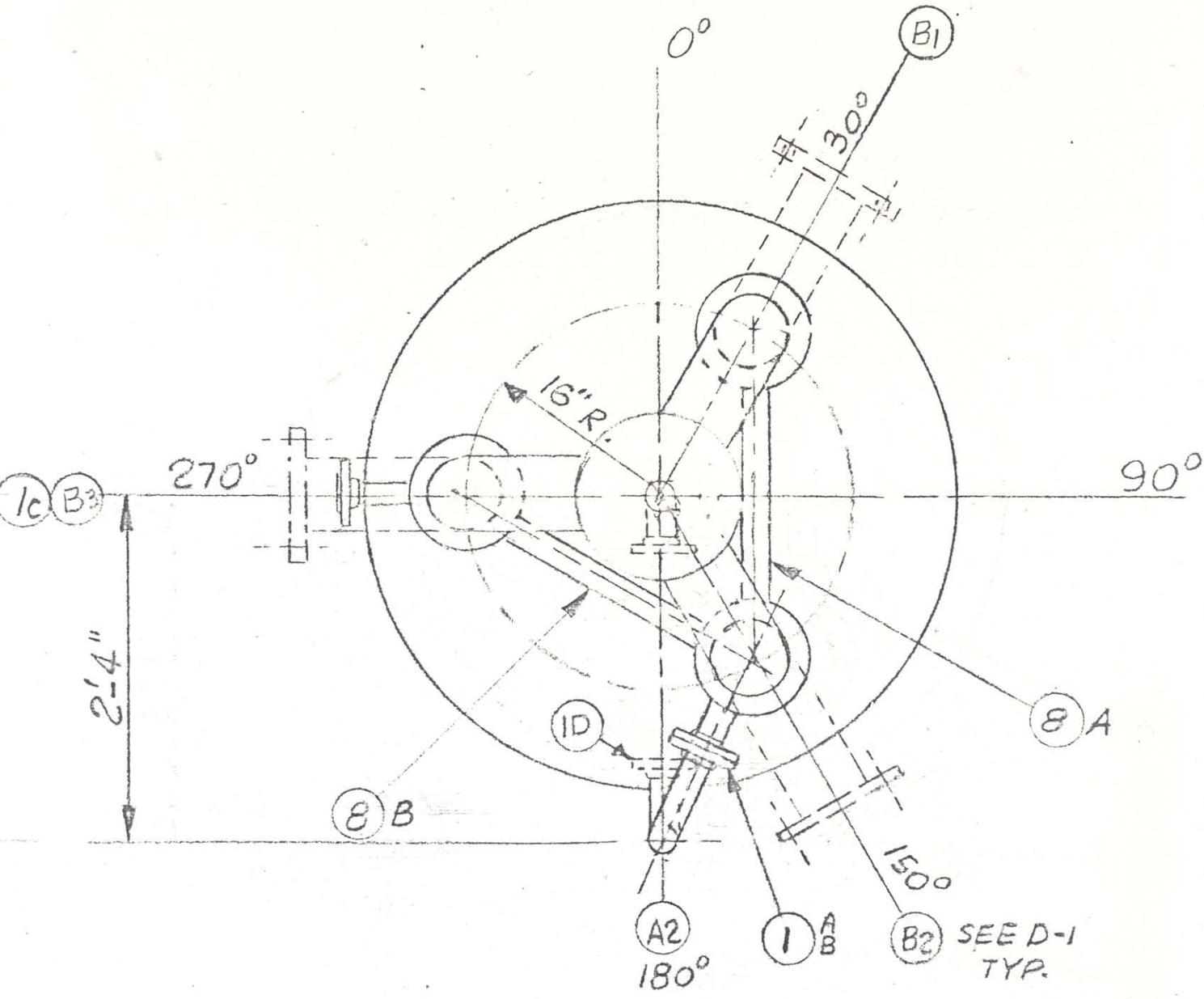
$6'' = .5 \cdot \quad .25^2 \times 3.14 =$


D3
0°
℄ LS2A
B1
8"
6"
6
30°
45°
315°
Fill
90°
M
℄ NAME R
Rotate to 325°
15°
F
NOTCH ANGLE & WELD TO PAD, TYP.
135°
225°
℄ LS1
210°
C1
C2
C3
6
15°
℄ LS2B
TYP. CLIPS
180°
A2
A1
B2
150°
+ clip 90
PLAN
10A
12"
6"
7
8
3"
9
10'-0"
9
8



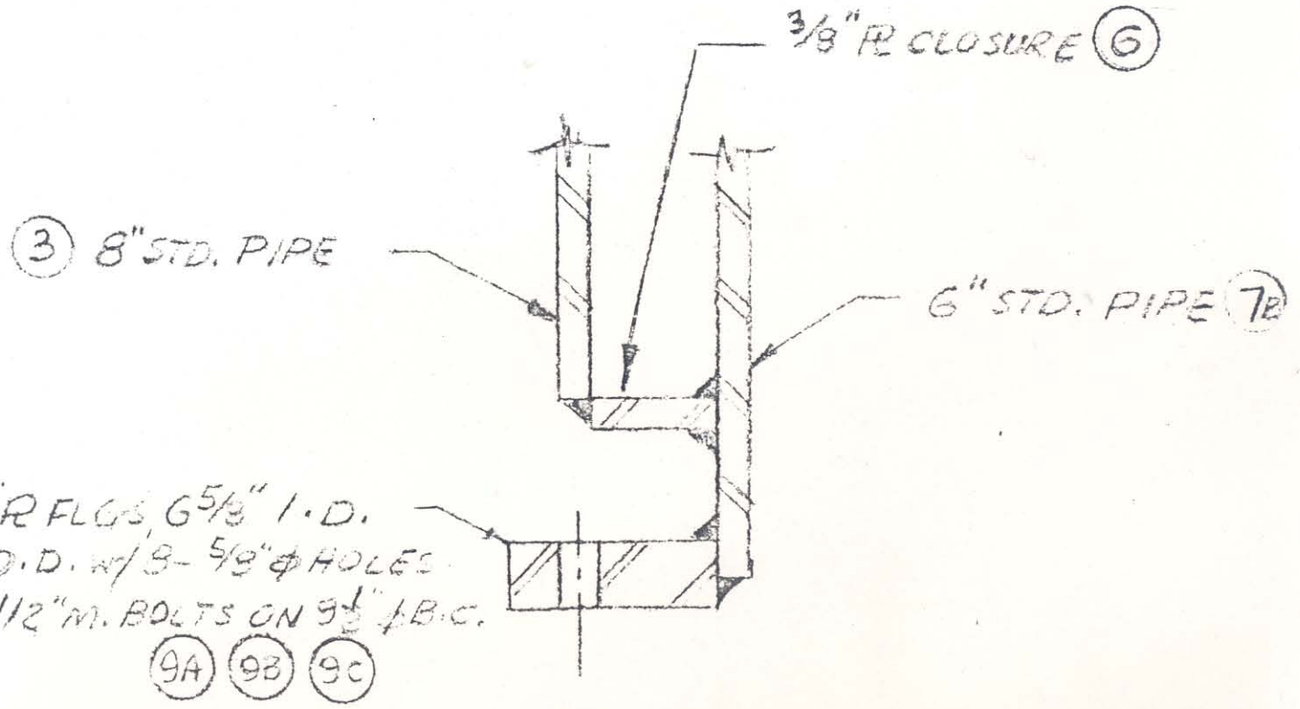
ELEVATION

SEE PLAN FOR ORIENTATION



PLAN

(SEE DWG. D-1 FOR ALL FLGS. NOT SHOWN HER



DETAIL "A"

MARK	REVISION	BY	DATE

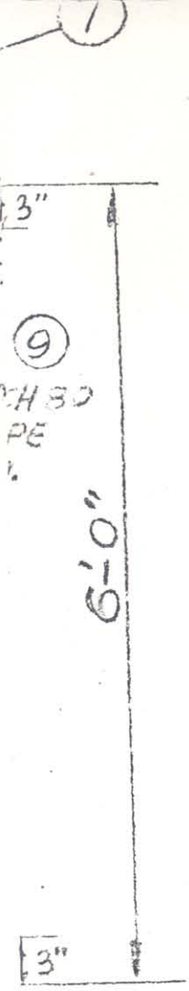
W.O. #0677 D-1

ITEM	QTY REQ'D.	BILL OF MATERIAL DESCRIPTION	MATERIAL	T	B/S	P.O. # W.O. # OTHER
1	1	48" O.D. x 3/8" MIN. 2:1 S.E. HD.	SA-285-C	T		
2	1	3/8" R 96" x 150" LG.	SA-285-C	T		
3	1	7/16" R 48" x 150" LG. (MAKES 2-R2)	SA-285-C	T		
4	4	L4x4x3/8" x 7'-0" LG.	SA-36	—		
5	4	1/2" R 6"x6" BASE R's	SA-36	—		
6	4	1/2" R 6"x9" PLATF. CUPS	SA-36	—		
7	7	2" 150# S.O. FLGS. R.F.	SA-181-I	T		
7a	4	5/8" ϕ x 3" LG. STUDS w/ 2-H NUTS	SA-193 B7	T		
7b	1	2" 150# J.M. 60 GSKT.	ASB.	—		
8	4	2" SCH 80 90° L.R. ELL.	SA-234	T		
9	1	2" SCH 80 PIPE x 21'-0" LG.	SA-53 B	T		
10	4	3/4" TANK FLGS.	SA-105	T		
10A	1	1/2" ✓ ✓	✓	T		
11	1	6" SCH 40 PIPE x 49'-0" LG.	SA-53B	T		
12	4	6" SCH 40 WELD CAPS	SA-234	T		
A½	1	2x2x2 SCH 80 WELD TEE	SA-234	T		
B	6	1/2" R x 11" O.D. FLGS.	SA-285-C	T		
B	6	3/8" R x 12" O.D. PADS	SA-285-C	T		
E	1	3" 150# S.O. FLG. R.F.	SA-181-I	T		
E	1	3" SCH 80 PIPE x 8" LG.	SA-53-B	T		
E	1	3/8" R x 8" O.D. PAD.	SA-285-C	T		
F	1	8" 150# S.O. FLG. R.F.	SA-181-I	T		
F	1	8" SCH 80 PIPE x 7" LG.	SA-53 B	T		
F	1	3/8" R x 15" ϕ PAD	SA-285-C	T		
M	1	11"x15" MANWAY COMPL. w/ 3" RING	C.S.			
	80#	WELD ROD	C.S.			
	1	NAME PL w/ BRKT.	C.S. S.S.			
A½	2	2" 150# W.N. FLGS. R.F. SCH 80 BORE	SA-181-I	T		

PIPE, TYP.

(7)

1974 DESIGN DATA				
1. CODE STAMP	A. S. M. E. ~~1965~~ SECT VIII LATEST EDITION			
2. OTHER SPECS.				
3. DESIGN PRESS. @ TEMP.	150	PSIG @	650	°F
4. OPER. PRESS @ TEMP.	—	PSIG @	—	°F
5. POST WELD HEAT TREAT.	NO			
6. RADIOGRAPHING	NO			
7. JOINT EFFICIENCY	SHELL 70 %	HEADS 80 %		
8. NOM. CORR. ALLOWANCE	SHELL 0	HDS. 0	NOZZ. 0	
9. MAX. ALLOW. PRESS. (New • Cold)	150 PSIG			
(a) MAX. ALLOW. PRESS. LIMITED BY	SHELL			
10. HYDROSTATIC TEST PRESSURE	225 PSIG			
11. MATERIAL SPECIFICATIONS				
(a) SHELL & CONE	SA-285-C			
(b) HEADS	SA-285-C			



2. OTHER SPECS.			
3. DESIGN PRESS. @ TEMP.	150 PSIG @ 650 °F		
4. OPER. PRESS @ TEMP.	— PSIG @ — °F		
5. POST WELD HEAT TREAT.	NO		
6. RADIOGRAPHING	NO		
7. JOINT EFFICIENCY	SHELL 70 %	HEADS 80 %	
8. NOM. CORR. ALLOWANCE	SHELL 0	HDS. 0	NOZZ. 0
9. MAX. ALLOW. PRESS. (New ● Cold)	150 PSIG		
(a) MAX. ALLOW. PRESS. LIMITED BY	SHELL		
10. HYDROSTATIC TEST PRESSURE	225 PSIG		
11. MATERIAL SPECIFICATIONS			
(a) SHELL & COVE	SA-285-C		
(b) HEADS	SA-285-C		
(c) INTERNALS (Except as noted)	SA-53 B		
(d) SUPPORT	SA-36		
(e) FLANGES	SA-181-I & SA-285-C		
(f) GASKETS	J.M. #60		
(g) COUPLINGS	SA-105		
(h) NOZZLE NECKS	SA-53 B		
(j) MANWAY NECK	C. STL.		
(k) BOLTS	SA-193 B 7		
(l) NUTS	SA-194 2H		
(m) INSULATION RINGS	—		
(n) FIREPROOFING LUGS	—		
(p) REINFORCING PADS			
12. FINISH: NONE			
13. INSPECTED BY: AUTHORIZED CODE INSPECTOR			
14. TOTAL ESTIMATED FAB. WT.	4500 #		

NOTES

(A) FLANGE BOLT HOLES STRADDLE VESSEL NATURAL CENTERLINES UNLESS OTHERWISE NOTED.

(B) VESSEL SHALL BE CLEANED INSIDE & OUTSIDE FREE OF ALL LOOSE SCALE, RUST, GREASE, WELD SPATTER AND OTHER FOREIGN MATTER.

(C) PROTECT ALL EXPOSED FLANGE FACES WITH WOODEN COVERS AND ALL THREADED CONNECTIONS WITH SUITABLE THREAD PROTECTORS FOR SHIPMENT.

O.D. NF.	REMARKS
—	
8"	
—	
15	
12"	W/ INT. PIPE
—	W/ 2×2×2 TEE

BUTANE TANK CORPORATION

3185 E. WASHINGTON BLVD. LOS ANGELES, CALIFORNIA 90023

48" O.D. × 14'-0" O/AL. CARBON STRIP TANK.

FOR SUMMIT CORP. P.O. SR043-04[illegible]-22[illegible]

DRN. BY	R.Z.K.	2-24-76
CHD. BY	D.M.	—
SCALE—	NONE	

WORK ORDER	SHEET NO.	REV.
0677	D-1	0

DRAWING NUMBER

MARK	REVISION	BY	DATE

BILL OF MATERIALS

0677 D-2

ITEM	QUAN.		MAT'L			
	4	2" 150# S.O. FLGS. R.F.	A-181-I	—		
	4	5/8" ϕ x 3" LG. STUDS w/ NUTS	C. STL.	—		
	1	2" 150# J.M. 60 GSKT. 1/16" THK.	ASB.	—		
	2	2" SCH 80 90° L.R. ELLS.	SA-234	—		
	3	8" STD. PIPE x 5'-0" LG.	A-53	—		
	1	12" STD. PIPE x 2'-6" LG.	A-53	—		
	1	3/8" R. x 12" O.D. x 6 5/8" I.D.	A-36	—		
	6	3/2" R x 8" O.D. x 6 5/8" I.D.	A-36	—		
	3	6" STD. PIPE x 2'-11" LG.	A-53	—		
	3	6" STD. PIPE x 5'-6" LG.	A-53	—		
	1	2" SCH 80 PIPE x 18'-0" (MAKES 7 PCS.)	A-53			
	3	1/2" R FLGS. 6 5/8" I.D. x 11" O.D.	A-36			
	8	1/2" ϕ NC x 2" LG. MACH. BOLTS w/ NUTS	C.S.			
	1	11" O.D. x 6 5/8" I.D. GSKT. w/ 2-3/8" HOLES	J.M. 41 GC			
	45#	WELD ROD	C.S.			

DESIGN DATA

1. CODE STAMP ~~A.S.M.E. 1965 SECT VIII LATEST EDITION~~				
2. OTHER SPECS.				
3. DESIGN PRESS. @ TEMP.		PSIG @		°F
4. OPER. PRESS @ TEMP.		PSIG @		°F
5. POST WELD HEAT TREAT.				
6. RADIOGRAPHING				
7. JOINT EFFICIENCY	SHELL	%	HEADS	%
8. NOM. CORR. ALLOWANCE SHELL	HDS.		NOZZ.	
9. MAX. ALLOW. PRESS. (New ● Cold)		PSIG		
(a) MAX. ALLOW. PRESS. LIMITED BY				
10. HYDROSTATIC TEST PRESSURE		PSIG		
11. MATERIAL SPECIFICATIONS				
(a) SHELL				
(b) HEADS				

Goldfield, Nevada

March 29, 1976

Summa Corporation
P. O. Box 1126
Tonopah, Nevada 89049

Atttention: Messrs. William J. Robinson
David K. Hamilton

Re: Pilot Plant Cyanide and Flotation

Gentlemen:

With reference to our discussions concerning a pilot plant test facility, I make the following proposal to Summa Corporation:

That the equipment, listed in attachment, owned by Summa Corporation be installed at the ore processing facility I am constructing between Goldfield and Tonopah, Nevada. This equipment shall be separate from the main operational circuit and installed as a basic pilot plant with approximately the same flowsheet as the proposed layout, March, 1975. I will assume all installation and maintenance costs.

For test work pilot plant operation, Summa Corporation will have free use of the plant, but will bear its own labor, power, maintenance, reagent cost. Summa personnel may operate the plant.

Since test work is intermittant, we should have little problem scheduling for time with the unit.

Summa Corporation or Pruett may terminate this arrangement at will and the party asking termination will bear cost of returning equipment to Summa Corporation yard, Tonopah. Machinery will be returned on notice in equal or better condition, less normal wear.

The intent of the arrangement is to establish a modern flotation-carbon in pulp cyanide pilot plant usable by both Summa Corporation and Pruett. Pilot plant operation may be expected by June, 1976.

related info in Pruett Metallurgy folder

Goldfield, Nevada

March 29, 1976

Summa Corporation
P. O. Box 1126
Tonopah, Nevada 89049

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related info in Pruett Metallurgy folder

The equipment additions needed to complete the pilot plant would cost approximately $3,500. Summa Corporation may elect to supply these items. If supplied by myself, I would retain their ownership.

Since I intend to install a pilot circuit with or without your machinery, I request an answer to this proposal within the next ten days.

Sincerely,

David L. Pruett

Attachments

SUMMA CORPORATION EQUIPMENT

1	Deco Ball Mill #24 10 hp. Miscellaneous extra mill balls	HTC N000906
1	2" Wemco sand pump No motor	HTC N000903
2	1½" Wemco sand pumps 2 hp.	
1	2" Denver vertical pump 2 hp.	
1	15' Bucket Elev. 1½ hp. gear head motor	
1	10 x 16 Rodgers Saw Crusher with 25 hp. motor Delco B 786A	HTC N00916
4	Denver #12 Flotation Cells (71595-2)	
6	Denver #8 Lab Flotation Cells	
1	Denver Agitator Mechanism 10" prop. No motor (Manhattan)	
1	Wilfley standard table (Manhattan)	

DATE: February 5, 1975

TO: D. J. Gribbin and Walt Simmons

FROM: Dave Pruett

SUBJECT: Pilot Plant Functions

1. Use as a Sampling Plant: Sampling can be done much more accurately when done in bulk. The larger the sample, the better statistical base. The finer the sampled material is crushed, the more accurate the results of sampling. No sampling is completely unbiased but the mechanical type, if set upright can correct and average most error. The fine bin discharge from this pilot plant may easily be routed outside to a sampler if desired. Capacity: 3 tons per hour minus ¼".

The auto samplers provided will give good head and tail information of the ground and processed pulps.

2. Crushing and Grinding Evaluations: Observations of time, power requirements, size distribution, particle shapes may be correlated directly to requirements of full size machinery.

3. Carbon in Pulp-Cyanide Plant -- Gold and Silver Ores: This pilot plant is a direct copy of the most current gold-milling trend in gold extraction. The newest American plant using this technique is the Homestake Mill, Lead, South Dakota. Three new major plants in South Africa built within the last four years have been carbon in pulp. Ores not generally adaptable to heap leaching or of value too high to risk the losses inherent in heap leaching are often treated economically with carbon in pulp.

Advantages compared to conventional cyanide:

(1) Minimum amount of equipment.
(2) Ores with difficult settling or filtration character may be treated.
(3) Free cyanide need not be maintained in final

stages, a major problem in environmental waste.
(4) Fouled solutions do not retard precipitation.
(5) Low demands on operational personnel, technically and in time required.

This process is not experimental but a well proven method.

4. Flotation: Ores other than gold and silver may be run. Silver ores often treat better in flotation than cyanide. Methods are well established and operational costs low. Also, fine carbon can be treated by flotation and recovered. Cheaper carbons may be tested in conjunction with the cyanide plant. Flotation of carbon plant tails check carbon loss and abrasion property.

5. In-House Test Work: Although outside consultants may be necessary often, the advantages of test work by existing personnel are readily apparent. If a full-scale operation is envisioned many of the bugs can be eliminated by pilot operation. Experiments altering the flow of existing plant machinery can be run first in pilot.

6. Adaptable Ores: The carbon in pulp plant represents the probable flow sheet and treatment of Tonopah ore and the Tonopah Belmont Tailings. Checks can be run on Manhattan ore and others. The flotation plant could process the type of ore from the Lida area and the properties in Mineral County or any flotable ore. By acid proofing the agitators and pumps copper ore could be leach tested. Any bulk samples could be crushed and split.

7. Rates of Treatment:

Sampling	3 tons per hour	(-¼ inch)
Carbon in pulp	15 tons per day	(-35 mesh)
Flotation	10 tons per day	(-65 mesh)

(Higher for softer ores and coarser grinds.)

8. Labor Requirements:

1 crusher man	1 shift/24 hours
1 mill operator	1 per shift

Dave Pruett

SUMMA CORPORATION: TONOPAH, NEVADA

PILOT PLANT COSTS: FLOTATION, CARBON IN PULP

	Material	Labor
Feed Hopper 5' x 9'	$ 600.	$ 200.
Elevator Repair	100.	100.
2' x 4' Screen	600.	250.
6" x 16" rolls	1,500.	250.
Fine Ore Bin	750.	300.
Feeder Installation		150.
Screw Classifier	1,500.	200.
Pump Installation with catch	125.	100.
Reagent Feeders:		
2 Clarkson Wet	500.	30.
1 Dry	150.	80.
Auto Sampler	350.	150.
Conditioners Tanks (3)	1,800.	300.
Conditioners Motors	300.	100.
Carbon Screens (3)	1,500.	250.
Carbon and Pulp Air Lifts	250.	200.
Compressed Air -- 30 cfm.	1,750.	200.
Carbon Feeder	200.	50.
Auto Sampler	350.	150.
Tail Pump Installation	150.	100.
Pan Filter	300.	250.
Pan Filter Vacuum Blower	400.	150.
Flotation Machines	1,750.	200.
Supervision, Engineering, 40 days to complete		6,000.
	$14,925.	$9,860.
Set Amalgamation Unit	250.	250.
Set furnace	400.	250.
Carbon Bin	400.	250.
Drum Deck Wood	600.	300.
	$16,575.	$11,010.
Estimate plus Contingency	$27,575. + 2,425.	
TOTAL ESTIMATE		$30,000.

Goldfield, Nevada
April 2, 1976

Mr. Paul G. Reeve
General Manager
Summa Corporation
Mining Division
3421 Las Vegas Blvd. South
Las Vegas, Nevada 89109

Dear Mr. Reeve:

Dave Hamilton mentioned to me this morning that Summa is considering some test column leach tests done by Mountain States Engineering of Tucson.

I am involved in several other mining projects as a consultant that have a great deal of experience with Mountain States. One of these operations has a pending action in civil court against Mountain States. One of the charges is fraud.

From what I have seen, the charge is true. In addition, their work and recommendations are notoriously sloppy and ill directed. I would be highly disturbed if Mountain States has any access to the carbon stripping unit. The carbon stripping designs and engineering presented by Mountain States that I have seen, not only will not work effectively, but the 4000-pound units have a price tag of $175,000 to $200,000 plus a $40,000 engineering fee.

I don't believe these people should have the chance to copy our work.

I am not personally acquainted with these people, so that the above should not be construed as personal criticism. I can, however, provide you with the data upon which this criticism is based.

Sincerely,

David L. Pruett

cc: Francis Fillerup
William Robinson
David Hamilton

GEOLOGY
EXTRACTIVE SYSTEMS
MINE PLANT DESIGN

DAVID L. PRUETT
MINING GEOLOGIST
GOLDFIELD, NV 89013

May 7, 1976

Bill Robinson
Summa Corporation
3421 Las Vegas Blvd, South
Las Vegas, Nevada89019

Dear Mr. Robinson:

Here is the basic proposal and the main data sheets for equipment cost, installation cost, engineering fee, and operational costs. Our test work shows a 73% recovery to be possible with this arrangement.

I would require 50% of the estimate in advance, 25% on completion of the plant and turn key to you, and the balance on completion of the buildings. You may wish to write up some specifications for the work. I would be responsible for all costs, labor, and engineering supervision. If this goes thru we can hash out the details.

Sincerely,

David L. Pruett

David L. Pruett

cc: David K. Hamilton

also filed in Mt. const reports folder

GEOLOGY
EXTRACTIVE SYSTEMS
MINE PLANT DESIGN

DAVID L. PRUETT
MINING GEOLOGIST
GOLDFIELD, NV 89013

May 7, 1976

MANHATTAN CARBON-IN-PULP
Proposed Flow Sheet
50 Ton Per Hour Plant Feed

HP		COST
15	Wobble Feed Scalper	$ 9,500
	-8"	*2,500
	+8"	Summa
5	30" Conveyor	*2,500
		Summa
30	Hammer Mill	*2,500
		Summa
5	24" Conveyor	*2,500
	Stockpile	*1,500
5	Tunnel Load Conveyor	7,500
		*4,000
	Merric Weightometer	2,700
		*1,500
150	Ball Mill Scrubber	10,000
		*4,500
10	Screen-48 Mesh	4,500
		*1,500
	+48M Tail	
15	4" Pump Cyclone	3,200
		*1,000
25	Agitator #1	10,000
		*2,500

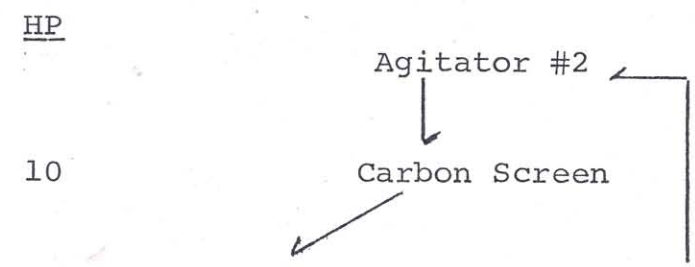
* Installation Cost

DSP

GEOLOGY
EXTRACTIVE SYSTEMS
MINE PLANT DESIGN

DAVID L. PRUETT
MINING GEOLOGIST
GOLDFIELD, NV 89013

Flow Sheet - Page Two



HP		COST
	Agitator #2	$ 10,000
		*2,500
10	Carbon Screen	*4,500
		* 500
1	Tailing	1,500
	Carbon Conveyor	* 500
290 Hp Total	Total Equipment	58,900
	Installation	*30,000
40 x 60 Building - New - Metal Insulated		25,000
20 x 30 Building		6,000
	Total Buildings	31,000
Tailing Pond by Summa		
	Subtotal	$119,900
	Engineering	25,000
	Total Project	$144,900

DLP

GEOLOGY
EXTRACTIVE SYSTEMS
MINE PLANT DESIGN

DAVID L. PRUETT
MINING GEOLOGIST
GOLDFIELD, NV 89013

May 7, 1976

Manhattan Carbon-in-Pulp
Direct Operating Cost
50 Ton Per Hour

	Cents/Ton
Electrical 225 KW $7.00/hour	$.15
Chemical	.50
Labor $280.00/shift 3 Men plus Supervisor 2 Crusher, 1 Plant Operator	.70
Maintenance and Wear	.10
General	.20
TOTAL	$1.65/Ton

DLP

GEOLOGY
EXTRACTIVE SYSTEMS
MINE PLANT DESIGN

DAVID L. PRUETT
MINING GEOLOGIST
GOLDFIELD, NV 89013

MANHATTAN CARBON-IN-PULP
Scrubbing Tests May 5 & 6 - Boundy and Pruett

Material: Big Pine Pit Screen Analysis #4 - Group 26

3 Tests on (-1"+ ½")(-½" + ¼") and (-¼")

100 Assay tons of each was placed in a 12" x 12" lab ball mill with 20 - 1" steel balls, 2 L. of water and tumbled for 10 minutes each.

Summary: 16.7% of total Au was rejected in the -1" + ¼" scrubbed material. 83.3% of total Au is contained in the -¼". This is an suitable product for carbon-in-pulp. If the Skinner work on scrubbing last year bears out 76% of this Au will be -48 mesh.

This test work should be continued.

DLP

GEOLOGY
EXTRACTIVE SYSTEMS
MINE PLANT DESIGN

DAVID L. PRUETT
MINING GEOLOGIST
GOLDFIELD, NV 89013

May 25, 1976

Mr. William J. Robinson
Summa Mining Division
P. O. Box 1126
Tonopah, Nevada 89049

Re: Manhattan Ore Treatment

Dear Mr. Robinson:

The test work just completed confirms the original estimates and recoveries stated in the May 7 letter to you. The data shows that by:

1. Crushing to minus one inch from Mine Run.
2. Scrubbing in a ball mill.
3. Screening to minus ten mesh will upgrade initial head assay by 2.5 to 1.

85% of the total gold will be contained in 32% of material on the average ore.

The minus 10 mesh fraction may be agitated in tanks with cyanide. 88% of the gold may be recovered from solution by activated cargon. These figures are averages.

The cost quoted ($145,000) will cover a plant with 500 ton per day minimum capacity. With some machinery exchange and additions, capacity can be raised to 1,000 ton per day - cost total $210,000. This decision should be made before construction start.

The basic data for the plant concept is uniform and with no surprises.

A summary of test work and the recent word data sheets is attached.

Sincerely,

David L. Pruett

DLP:ps

cc: David K. Hamilton

Also filed in Mtl. Corres & Reports re: Gr26

GEOLOGY
EXTRACTIVE SYSTEMS
MINE PLANT DESIGN

DAVID L. PRUETT
MINING GEOLOGIST
GOLDFIELD, NV 89013

SUMMARY: Screen Analysis

Agitated Cyanide Leaching

Big Four Mine

Big Pine Pit

Group 26

USGS Bulletin #723 - Ferguson

Coarse crushed ore (3 inch) screened to (- 5/8 inch)

(- 5/8 inch) 45% of total material contains 80% of gold. Deeper ore does not upgrade as well by screening.

March 12, 1975 - Skinner

Mine Run crushed to minus 1/4 inch; tumbled in lab ball mill 25 minutes.

		Assay	Percent Material
Material screened to	-48 mesh	.51 oz/ton	5.9
	+48 mesh	.01 oz/ton	94.1

Upgrading 13/1

76% of gold in 6% material.

Scrubbing and screening upgrades this ore.

March, 1976 - Boundy

Summary - Screen analysis Big Pine Pit

Group 26

+3 inch contains good grade - gold in fractures unreached by cyanide heap leach.

Screening alone does not upgrade Manhattan ore. Blasting recommended to open fractures.

GEOLOGY
EXTRACTIVE SYSTEMS
MINE PLANT DESIGN

DAVID L. PRUETT
MINING GEOLOGIST
GOLDFIELD, NV 89013

May 3, 1976 - Hamilton

Big Pine Pit Screen #3 and #4 - Boundy

-1/4 inch material in Bottle Roll Test.

72 hours. Recovery in solution 90% and 84.75% respectively.

Gold in -1/4 inch is soluble.

May 7, 1976 - Boundy & Pruett

Big Pine Pit Screen #4 (-1 inch composite)

Placed in lab ball mill with twenty 1 inch balls

3 liters water for ten minutes

83.3% of gold in 51% of material (-1/4 inch)

16.7% of gold in 49% of material (-1 inch + 1/4 inch)

May 12 through 24, 1976 - Pruett & Boundy

6 Scrub and cyanide tests were run on Big Pine Pit Mine Run crushed to (-1 inch).

The material was placed in a lab ball mill with twenty 1 inch balls for ten minutes.

Larger charges produced better results.

86% of gold - remains in (-10 mesh) 32% of total material. Agitated cyanide tests on this -10 mesh material shows:

88.6% Recovery	6 hours
92.5% Recovery	12 hours
95 %+ Recovery	24 hours

Base at 86% of gold in 32% of material gives an overall recovery of 76%, with 6 hour leaching.

CONCLUSION:

Big Four Big Pine Pit Group 26 ores are adaptable to upgrading by crushing, scrubbing and screening. Upgraded fines can be processed by cyanide in agitation tanks with much higher recoveries than heap leach.

ET NO. _____ C. _____

ASSAYER

Scrub Test #1

SUMMA CORPORATION
MINING DIVISION – TONOPAH, NEV.

LABORATORY REPORT

DATE 5/6 1976

B.P. Pit Screen Analysis #4 - Split B

DESCRIPTION	GROUP NO.	ozs. Au per ton	ozs. Ag per ton	Cu %	Pb %	Zn %					
"to -1" Heads	26	.026	} calc Head .073				}	11/1 upgrade.			
" to -1" Heads		.027									
from +½" to -1" Gravity Tails		.316	* Agitator Head					4/1. "			
" to -1" Scrubbed		.040									
" to -1" Gravity Concentrates		.897									
" to -½" Heads		.008	} calc head .1034								
" to -½" Heads		.006									
" to -½" Gravity Tails		.048	Agitator Head. TEST RESULTS								
" to -½" - Scrubbed		.022	—		2/1. upgrade. calc.						
" to -½" Gravity Concentrates		1.135									
" Heads		.072	calc Agit. head .172								
" Heads		.062				upgrade					
from -¼" Head Scrub Test		.157	←								
" Gravity Conc. from Scrub Test		2.036									
-1 + ½"	Scrubbed 10 min lab ball mill wet										
	Gravity concentrate panned from -¼										

Refer: DAVE PRUETT for

SHEET NO. 1 OF 1

B. ROBERTSON
ASSAYER

Sample Big Bne Pit Mine Run-Boundry Split 1A -(Screen Only) SCREEN ANALYSIS Date

Summa Corporati
Mining Division
Tonopah, Nevada

SIZE	WEIGHT (Grams)	ASSAY Tons	ASSAY Gold oz/T	Contained Gold Milligrams	% Total Gold	% Total Material	Cumulative % Gold	Cumulative % Material	Calculated Head oz/T	REMARKS
Mine Run			.188							Mine Run Crushed To (- 1")
+3"										Dry Screened. and weighed
-3"+1"										
-1"+½"	1381.9	47.38	.134	634.89	32.93	46.06	99.96 ~~100~~	99.52 ~~100~~		
-½+¼"	828.5	28.41	.107	303.99	15.76	27.62	67.03	53.46		
-¼+6m										
-6m+10m	235	8.04	.201	161.95	8.40	7.83	51.27	25.84		
-10m+35m	232.8		.424	338.43	17.85	7.76	42.87	18.01		
-35m+48m	43.2	1.48	.331 ~~.815~~	4902 ~~137~~ 20	254 ~~7.11~~	1.44	25.32	10.25		
-48m+80m	49.1	1.68	.815 ~~.410~~	137.20 ~~3.82~~	7.11	1.64	22.78	8.81		
-80m	215	7.38	.410	302.25	15.67	7.17	15.67	7.17		
									.188 - Good Sample	

* 1 assay ton = 29.166 grams
.01 ounce/ton = 1 milligram/assay ton

David J Puetz
ENGINEER

Sample Scrub Test #2 — Big Pine Pit Split TB Mine Run - Boundy — SCREEN ANALYSIS — Date

Summa Corporat
Mining Divisio
Tonopah, Nevad

SIZE	WEIGHT (Grams)	ASSAY Tons	ASSAY Gold oz/T	Contained Gold Milligrams	% Total Gold	% Total Material	Cumulative % Gold	Cumulative % Material	Calculated Head oz/T	REMARKS
Mine Run		100	.188							Mine Run crushed To -1" scrubbed
+3"										in Lab Ball mill 10 min 20 balls- 2c water
-3"+1"										Weight on screen tractions are dry weights
-1"+½"	1184	42 35	.189	767 34	42. 6		100	100	←	Poor Screening
-½+¼"										See- Screen test 1A
-¼+6m	721 5	25 81	.046	113 80	6 84		53 78	57 71		Bad Data - here "1+½"
-6m+10m	84	3	.202	58 18	3 5		46 94	31 90		
-10m+35m	81	2 90	.217	58 94	3. 54		43 44	28 90		
-35m+48m	69 2	2 48	.379	89 82	5 40		39 9	25 94		Test
-48m+80m	62 8	2 25	.454	97 61	5 87		34 5	23 46	.349	CN #1 Manhattan
-80m	593	21 21	.234	475 77	28 63		28 63	21 21		
	2795 5	100		1661 46					.173	

* 1 assay ton = 29.166 grams
.01 ounce/ton = 1 milligram/assay ton

ENGINEER

METALLURGICAL TESTING DATA

Summa Corporation
Mining Division
Tonopah, Nevada 89049

Type Test: Agitated Cyanide #1

Date: 14 May 76

SAMPLE DESCRIPTION	TIME	WEIGHT ORE / SOLUTION	ASSAY TONS	ASSAY GOLD oz./ton	CONTAINED GOLD mg.	ASSAY SILVER oz./ton	NaCN % / % NaOH	CONSUMPTION NaCN / NaOH	% GOLD RECOVERY	% SILVER RECOVERY
-35 Mesh Scrub #2 ~~Scrub #1~~ BPP Manhatten	0	300g. / 700ml	10.28	.378 calc	378					
Tailing-Residue	12 Hrs	300 gr.		.028	28.79					
Solution	12 Hrs	1200	41.14	.085	349.69				92.5%	

Some solution lost in 24 Hr. Pulp wash. - do not use results of 24 Hr Test

David E Pruett

Sample Scrub Test #4 (2c) Big Pine Pit Run Split 1c (1A-1B) SCREEN ANALYSIS Date

Summa Corporati
Mining Division
Tonopah, Nevada

SIZE	WEIGHT (Grams)	ASSAY Tons	ASSAY Gold oz/T	Contained Gold Milligrams	% Total Gold	% Total Material	Cumulative % Gold	Cumulative % Material	Calculated Head oz/T	REMARKS
Mine Run			.188							Mine Run Crushed
+3"										to -1". Scrubbed
-3"+1"										in Lab Ball Mill
-1"+½"	340 5	11 67	.028	38 44	1 75					10 minutes with 3c-water - 20 balls
-½+¼"										
-¼+6m	706 29	24 21	.101	244 52	11 17					Poor Split - Not enough coarse for
-6m+10m	165 80	5 68	.230	130 64	5 97					normal ratios however assays
-10m+35m	224 00	7 68	.322	247 29	11 30					per size range look good.
-35m+48m	62 -	2 12	.524	111 08	5 07					
-48m+80m	107 50	3 68	.406	149 40	6 83				.334	81% of Gold in 55% of Material
-80m	1239 -	42 48	.298	1265 90	57 87					

* 1 assay ton = 29.166 grams
.01 ounce/ton = 1 milligram/assay ton

David J. Rhuett
ENGINEER

Sample Heap #1 (+3" Leached) Scrubs SCREEN ANALYSIS Date

Summa Corporati
Mining Division
Tonopah, Nevada

SIZE	WEIGHT (Grams)	ASSAY Tons	ASSAY Gold oz/T	Contained Gold Milligrams	% Total Gold	% Total Material	Cumulative % Gold	Cumulative % Material	Calculated Head oz/T	REMARKS
Mine Run										+3" Material
+3"										from Heap #1.
-3"+1"										Crushed to (-1")
-1"+½"	2106	72 21	.024	173 30	25 91	36 26	100	100		Scrubbed in Lab Ball Mill - 3L H2O
-½+¼"										20 Balls
-¼+6m	1674	57 39	.026	149 21	22 31	28 82	73 77	63 74		
-6m+10m	460	15 77	.008	12 91	1 88	7 92	51 46	34 92		
-10m+35m	521	17 86	.008	14 28	2 14	8 97	49 58	27 -		
-35m+48m	97 5	3 34	.016	5 34	0. 79	1 68	47 44	18 03		
-48m+80m	120 -	4 "	.060	24 66	3 68	2 06	46 65	16 35		
-80m	830	28 46	.101	287 44	42. 97	14 29	42 97	14 29		
	5808 5	199 15	278 81	668 84					.0335	

* 1 assay ton = 29.166 grams
.01 ounce/ton = 1 milligram/assay ton

ENGINEER

Sample Scrub Test #6 Big Pine O.T mine Run Split 1D SCREEN ANALYSIS Date

Summa Corporation
Mining Division
Tonopah, Nevada

SIZE	WEIGHT (Grams)	ASSAY Tons	ASSAY Gold oz/T	Contained Gold Milligrams	% Total Gold	% Total Material	Cumulative % Gold	Cumulative % Material	Calculated Head oz/T	REMARKS
Mine Run	3920	134.4	.188				100	100		
+3"	Mine Run crushed to -1", placed in Lab Ball Mill									
-3"+1"	Dry and Tumbled for 10 minutes then screened and weighed									
-1"+½"	1597	54 75	.008	43 8	1.6	41				comparison to
-½+¼"										Scrub Test #4 shows 1 - Dry more effective
-¼+6m	900	30 86	.084	259 22	9 45	23 1	98 40	59 65		or 2 - Higher % Coarse
-6m+10m	140	4 8	.134	64 32	2 35	3 35	88 95	36 55		Scrubs better ? or 3 larger volume in ball mill scrubs better
-10m+35m										
-35m+48m										
-48m+80m										
-80m	1225	43 02	.552	2374 70	86 60	32 22	86 60	32 22		1140 grams to Test. CN #2 Manhattan
				2625725						

* 1 assay ton = 29.166 grams
.01 ounce/ton = 1 milligram/assay ton

Daniel J Pruett
ENGINEER

Summa Corporation
Mining Division
Tonopah, Nevada 89049

METALLURGICAL TESTING DATA

Type Test: Agitated Cyanide #2 Manhattan

Date: _____

SAMPLE DESCRIPTION	TIME	WEIGHT ORE / SOLUTION	ASSAY TONS	ASSAY GOLD oz./ton	CONTAINED GOLD mg.	ASSAY SILVER oz./ton	NaCN % / % NaOH	CONSUMPTION NaCN / NaOH	% GOLD RECOVERY	% SILVER RECOVERY
-10 m Scrub #6 Big Pine Pit Split 1D		1140 / 4600	3908	.552	2157 ±1		.1 / .1			
Residue	6 HRS.			.063	246.2				-11.4	
Solution	6 Hrs			.036					88.6%	
			Some solution spilled washing							
			filter cake - no solution cross check.							
			% looks average with 12 + 24 HR							
			Tests.							

Sample +3" BPP - Screen #1 Boundy — SCREEN ANALYSIS — Date — Mining Divisi Tonopah, Neva

Screen #9

SIZE	WEIGHT (Grams)	ASSAY Tons	ASSAY Gold oz/T	Contained Gold Milligrams	% Total Gold	% Total Material	Cumulative % Gold	Cumulative % Material	Calculated Head oz/T	REMARKS
Mine Run										
+3"	Sample from +3" pile in lab yard - Screen Analysis # 1 Boundy Big Pine Pit									
-3"+1"	crushed to minus 1" scrubbed 10 min									
-1"+½"	2461	84.38	.012	101.255	39.07	62.38	99.97	99.96		-very low head assay calc .019 Ratios holding
-½+¼"										
-¼+6m	606	20.78	.008	16.62	6.41	15.36	60.90	37.58		well despite lack of fines in head
-6m+10m	243	8.32	.042	34.99	13.50	6.15	54.48	22.22		sample. !!
-10m+35m	213	7.33	.035	25.56	9.86	5.39	40.98	16.07		
-35m+48m	38	1.3	.060	7.81	3.01	0.94	31.12	10.68	.0314	Calc. 54.48% Gold
-48m+80m	46	1.578	.080	12.62	4.86	1.16	28.11	9.72		22.2% material
-80m	338	11.589	.052	60.26	23.25	8.56	23.25	8.56		
	3945	135.241		259.15	99.96	99.96			.019	Calc Head.

* 1 assay ton = 29.166 grams
.01 ounce/ton = 1 milligram/assay ton

Dan J Runta
ENGINEER

GEOLOGY
EXTRACTIVE SYSTEMS
MINE PLANT DESIGN

DAVID L. PRUETT
MINING GEOLOGIST
GOLDFIELD, NV 89013

May 25, 1976

Mr. William J. Robinson
Summa Mining Division
P. O. Box 1126
Tonopah, Nevada 89049

Dear Mr. Robinson:

Agitated cyanide leach tests on the Belmont tailings samples, Group 24, were completed May 24th.

Recoveries	Silver	Gold
Carbon in pulp - 4 hours	66%	30%
Cyanide alone - 12 hours	73%	35%
Cyanide alone - 24 hours	67%	?

A series of carbon in pulp tests should be run with 6-10-12 hour times.

The silver is precipitating from solution with excess time. Carbon in pulp with correct time in agitation may run recoveries into the 80% range.

These tailings may be one of the best low capital intensive operations in the Summa inventory.

Sincerely,

David L. Pruett

DLP:ps

Also filed in Gro 24 General

METALLURGICAL TESTING DATA

Summa Corporation
Mining Division
Tonopah, Nevada 89049

Type Test: Agitated Cyanide #1 Belmont

Date: 22 May 75

SAMPLE DESCRIPTION	TIME	WEIGHT ORE / SOLUTION	ASSAY TONS	ASSAY GOLD oz./ton	CONTAINED GOLD mg.	ASSAY SILVER oz./ton	NaCN % / % NaOH	CONSUMPTION NaCN / NaOH	% GOLD RECOVERY	% SILVER RECOVERY
Belmont Tailings Boundy 21 May 75	—	6497 / 15110	222.76 / 518.09	.018	400.97	1.556 / 3466.46 mg	.15 / .15	.09 / .08		
Residue	12 Hrs		222.76	.012		.430 / 9578.7 mg			~~35%~~	~~23%~~
Solution	12 Hrs					.539 / 27,914 mg			35%	73%

Dave B. Pruett

METALLURGICAL TESTING DATA

Summa Corporation
Mining Division
Tonopah, Nevada 89049

Type Test: Agitated Cyanide #2 Belmont

Date: 22 May 75

SAMPLE DESCRIPTION	TIME	WEIGHT ORE / SOLUTION	ASSAY TONS	ASSAY GOLD oz./ton	CONTAINED GOLD mg.	ASSAY SILVER oz./ton	NaCN % / % NaOH	CONSUMPTION NaCN / NaOH	% GOLD RECOVERY	% SILVER RECOVERY
Belmont Tailing Boundy		6000 / 13958	205 T 478 ST	.010		1.66 ✱ / 34641	.15 / .15			
Residue ↓	24 HRS					.572 / 11722 ✱				
Solution	24		47857			.2940		.08 / .08		67 %
			The gold assay salted - ? - do not use.							
			Silver OK.							

Summa Corporation
Mining Division
Tonopah, Nevada 89049

METALLURGICAL TESTING DATA

Type Test: Carbon in Pulp - Cyanide #3 Belmont

Date: May 24. 76

SAMPLE DESCRIPTION	TIME	WEIGHT ORE / SOLUTION	ASSAY TONS	ASSAY GOLD oz./ton	CONTAINED GOLD mg.	ASSAY SILVER oz./ton	NaCN % / % NaOH	CONSUMPTION NaCN / NaOH	~~% GOLD RECOVERY~~	% SILVER RECOVERY
Belmont Tailings Boundy		300gm / 700ml.	10.28	.010	10.28	1.55 / 1593.			-	
Residue	4HRS.	300± /	10.28	.070	7.19	.583 / 599 3			-37 6 %	
Solution	4HRS	/ 1000ml.				.103 / 459 35			28.83 %	$
6-10m PBC 5 grams Carbon		/	.1714	.067		8.6 / 146 20			41.6 % / 9.17 %	
	4 Hrs.	/				/				66 40
		/				/				
		/				/				
		/				/				
		/				/				
		/				/				
		/				/				
		/				/				

David F Pruess

GEOLOGY
EXTRACTIVE SYSTEMS
MINE PLANT DESIGN

DAVID L. PRUETT
MINING GEOLOGIST
GOLDFIELD, NV 89013

June 16, 1976

Messrs. William J. Robinson
& David K. Hamilton
Summa Corporation
P. O. Box 1126
Tonopah, Nevada 89049

Gentlemen:

The Homestake Mill at Lead, South Dakota, as you know, is very impressive. Their carbon-in-pulp plant is simple and straight forward. The carbon reactivation unit is for all practical purposes identical to Summa's. They strip at atmospheric pressure in three one-ton vats. Time is plus 48 hours. Daily gold production from carbon is about 110 ounces, from zinc precipitation in their sand plant, 950 ounces per day. Their carbon desorbtion is efficient but very slow. The unit has more than $40,000 invested in valving alone.

Homestake is considering dropping its zinc circuit and using carbon columns for the sand leach solution. They are also building a 400 ton per day plant at Creede, Colorado to handle 6 ounce silver ore. Homestake is not current on the newer desorbtion developments. Cortez Gold refused to share data. Buckhorn, Windfall and Summa's Tonopah Plant are the only operational pressure units. None of their data has been published. Summa's is the most advanced of the group.

I have provided Homestake most of the basic details of the new plants. In return, Ken Hall, Metallurgical Superintendent, has offered to review and aid with plans for the carbon-in-pulp units here. Since Homestake's 2,000 tpd carbon-in-pulp circuit is the only major plant operational, their operating experience could not be purchased elsewhere at any price.

Homestake needs the newer desorbtion technology, their own process does not scale up very well. Their aid in carbon-in-pulp design would take away the experimental nature at Summa's possible projects and add confidence and experience.

I have relayed your invitation to the Homestake metallurgical staff to visit Summa's Nevada Mines Operation. You may expect a quick reply.

Sincerely,

David L. Pruett

DLP:ps

GEOLOGY
EXTRACTIVE SYSTEMS
MINE PLANT DESIGN

DAVID L. PRUETT
MINING GEOLOGIST
GOLDFIELD, NV 89013

July 6, 1976

Summa Mining Division
P. O. Box 1126
Tonopah, Nevada 89049

ATTENTION: Mr. William J. Robinson

Dear Bill:

Although the McCoy and Manhattan ores are very different metallurgically, some of the basic procedures apply to both. Crushing-grinding units, the classification of material by size, and the agitation in cyanide and the carbon in pulp, uses the same machinery for both ores. Time and chemical dosage will be different.

McCoy may require the 4-Denver #12 flotation cells in the warehouse yard added to the unit. The basic layout of the unit is on the attached sheet.

Sincerely,

David L. Pruett

DLP:ps

Attachment

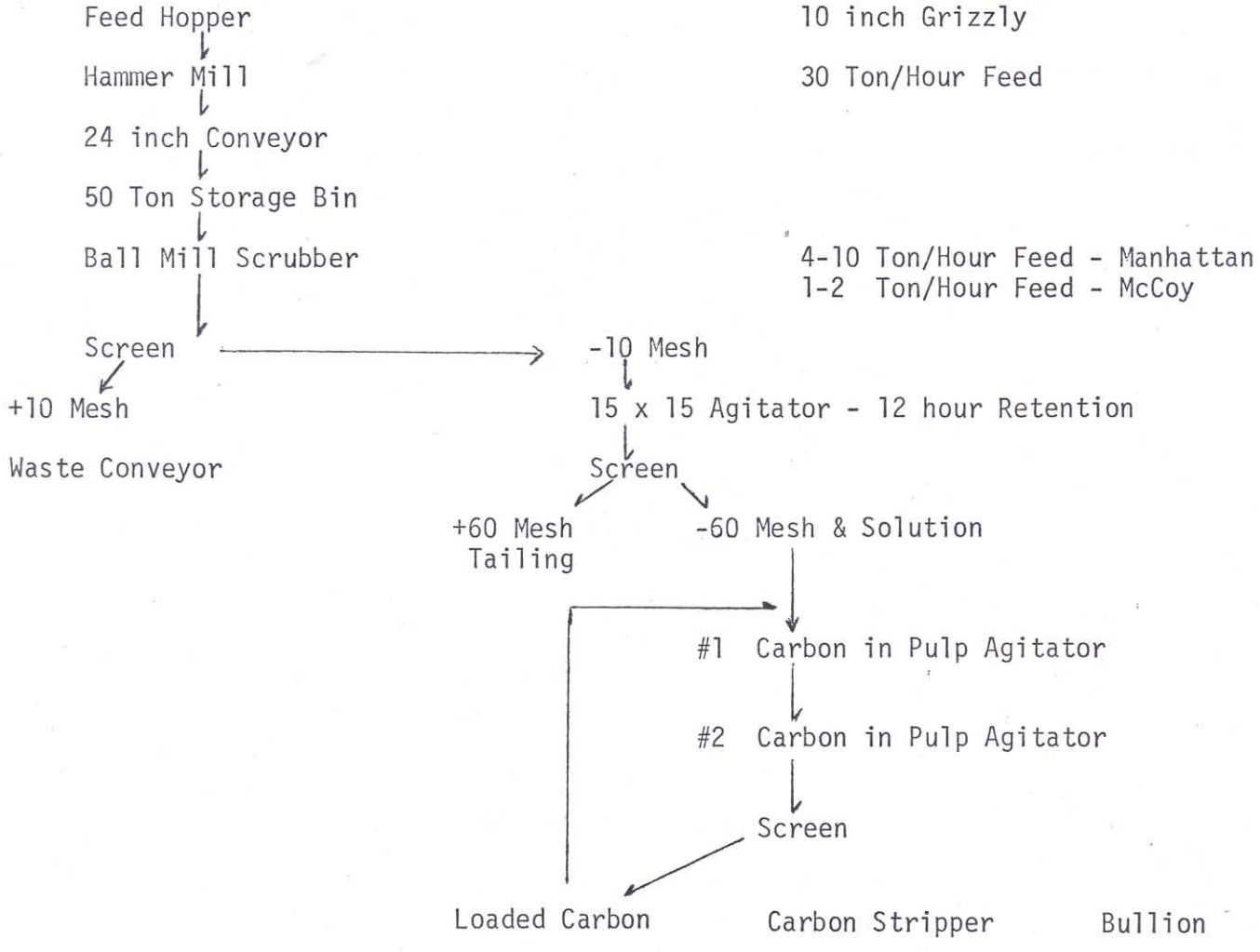
GEOLOGY
EXTRACTIVE SYSTEMS
MINE PLANT DESIGN

DAVID L. PRUETT
MINING GEOLOGIST
GOLDFIELD, NV 89013

July 6, 1976

FLOW SHEET - CARBON IN PULP PILOT

McCoy - Manhattan



This plant flow can be rearranged easily to fit other milling systems.

Manhattan ore with .10 oz./ton feed grade and 80% recovery would produce at

4 Tph feed	7.7 oz. day
10 Tph feed	19.2 oz. day

This pilot unit will certainly justify itself on the Manhattan ore.

GEOLOGY
EXTRACTIVE SYSTEMS
MINE PLANT DESIGN

DAVID L. PRUETT
MINING GEOLOGIST
GOLDFIELD, NV 89013

July 6, 1976

STATEMENT TO:

Summa Corporation
P. O. Box 1126
Tonopah, Nevada 89049

Professional Services McCoy Metallurgy	$4,000.00

David L. Pruett

Met.

General - Stripping Plant Improvements 130

HUGHES TOOL CO. 132
METALLURGY
GROUP 26, STRIPPING PLANT IMPROVEMENT

SUMMA CORPORATION

SUPPLIES AND LABOR TO IDAHO MINING

For carbon stripping in care of John Bennetts: 8/20 - 8/23/75

2	55 gallon drums methonal @ $74.25	$148.50
35	pounds NaOH @ $1.75	61.25
9.6	pounds copper screen 24 sq. ft.	60.48
10	6-pound HCl @ $30.55	305.50
34	hours labor W. O. Mollison	
4½	hours assay labor W. Robertson	

No charge for flux

Total excluding labor, flux:

$575.73

Ramyr MANUFACTURING CO.

Quality Processing Equipment backed by Dependable Service

INVOICE COPY

710 MONROE CIRCLE • P.O. BOX 608 • PLACENTIA, CALIF. 92670
(714) 521-8344 • (714) 996-4840

3120/3140

SALES ORDER NO.	DATE ENTERED	CUSTOMER ORDER NO.	DATE	SALESMAN
27921	9-4-75	Verbal-Bill Mollison		Kadel

INVOICE DATE	INVOICE NUMBER
9-10-75	16275

SOLD TO
Suma Corporation
P. O. Box 1126
Tonopah, Nevada 89049

SHIP TO

TERMS
NET 30 DAYS
or 1% 10

A SERVICE CHARGE OF 1½% PER MONTH WILL BE CHARGED ON ALL DELINQUENT ACCOUNTS.

DATE SHIPPED	SHIPPED VIA
9-10-75	CPU
WAYBILL NO.	

SHIP VIA	F.O.B.	TAXABLE	RESALE	CHARGE	C.O.D.	DROP SHIP	STOCK
Customer Pick Up	Placentia, CA.	NO					X

PARTIAL	COMPLETE	NO. CARTONS	SHIPMENT NO.
	X		

ITEM	QUANTITY ORDERED	DESCRIPTION	UNIT PRICE	BALANCE ON ORDER	QUANTITY SHIPPED	TOTAL AMOUNT
1	1	Ramyr Rectifier A-150-6, 115 volt input 121-009 03405	534.00		1	534.00
		Serial No. 06926				
2	30 ft.	PVC Screen 03405-413-002	2.00		30 ft.	60.00

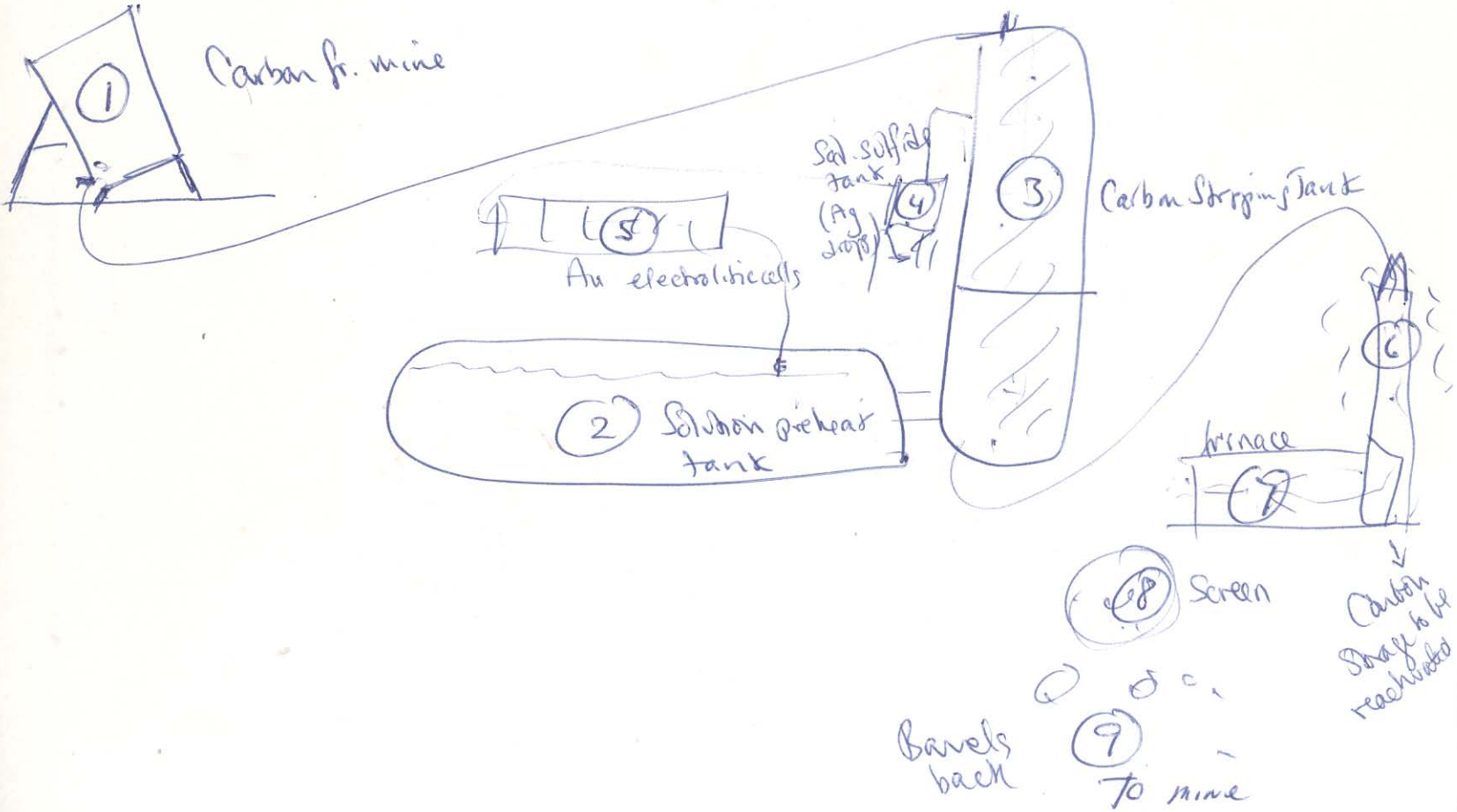
New unit for stripping unit (old one being repaired + will be chg to R/m.

Stripping unit at Ind. Park

CUSTOMER INSTRUCTIONS:

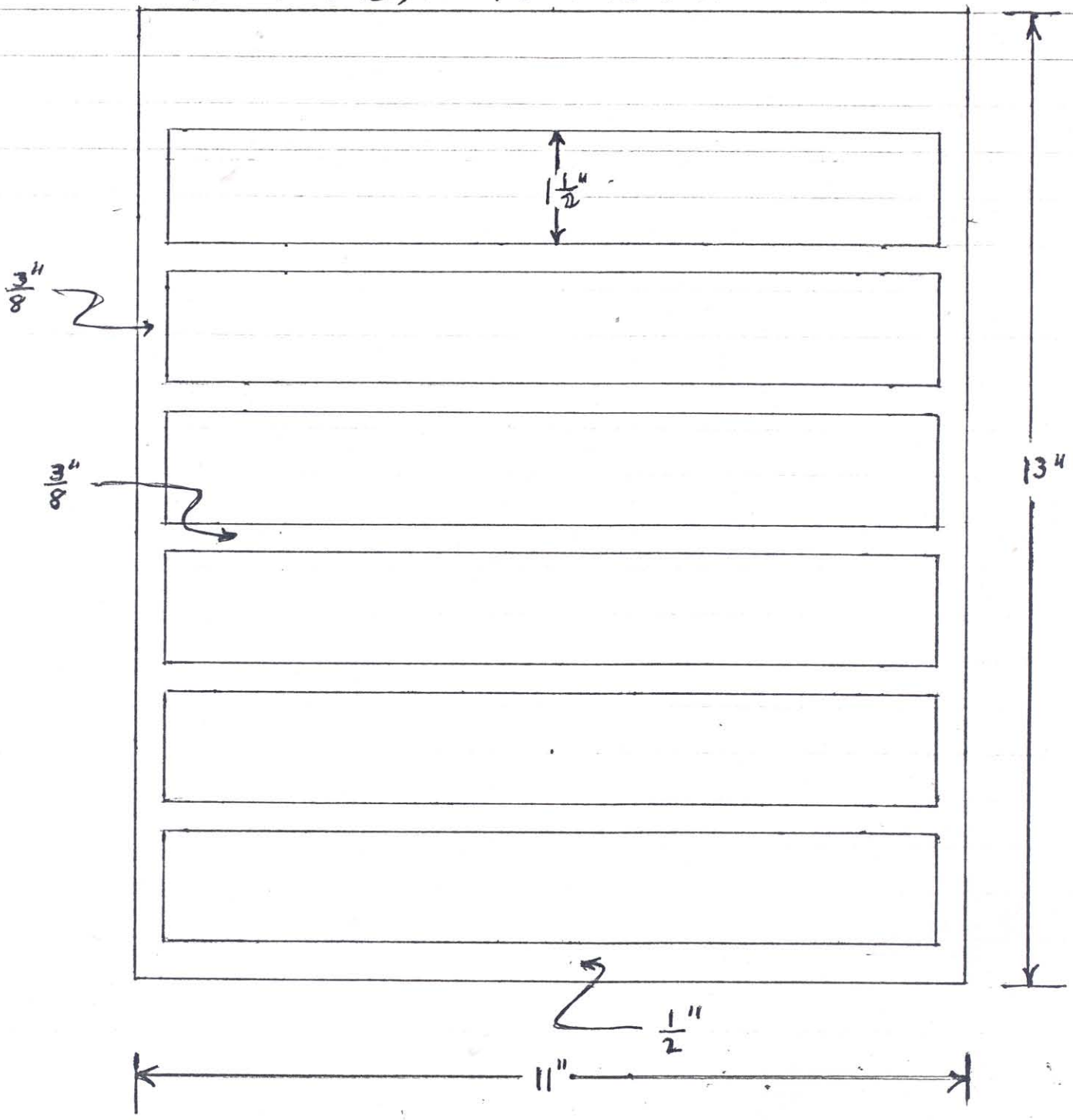
SALES TAX	$
FREIGHT	$
TOTAL	$ 594.00

"WE HEREBY CERTIFY THAT THE ARTICLES AND SERVICES COVERED BY THIS INVOICE WERE PRODUCED AND PERFORMED IN COMPLIANCE WITH ALL APPLICABLE REQUIREMENTS OF SECTIONS 6, 7 AND 12 OF THE FAIR LABOR STANDARDS ACT, AS AMENDED, AND OF REGULATIONS AND ORDERS OF THE UNITED STATES DEPARTMENT OF LABOR ISSUED UNDER SECTION 14 THEREOF."


1
Carbon fr. mine
Sol. sulfide tank
(Ag ppt)
4
5
Au electrolitic cells
3
Carbon Stripping Tank
2
Solution preheat tank
6
furnace
7
8
Screen
Carbon Strap to be reactivated
Barels back
9
To mine

Electrolytic Baskets

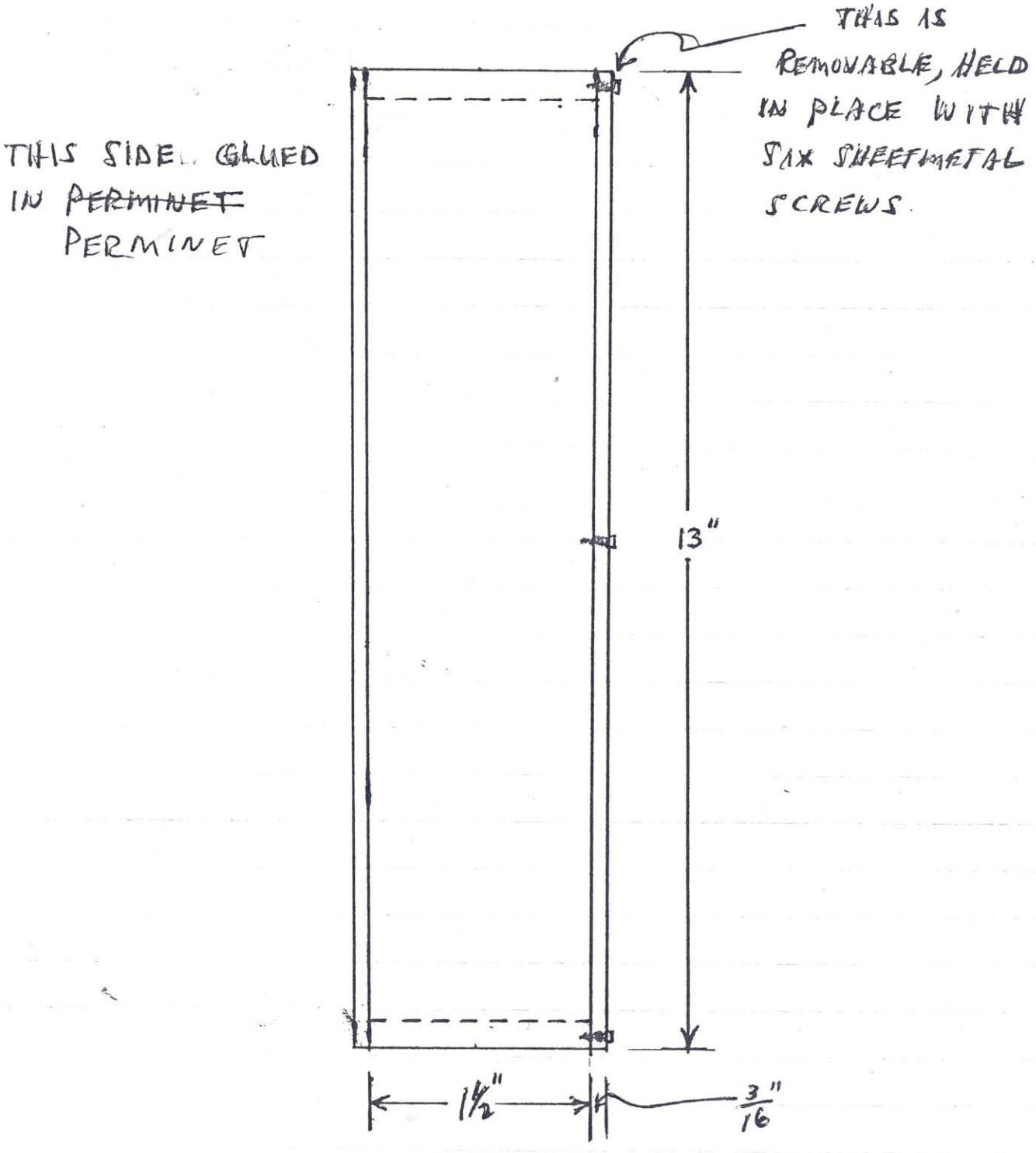
BOTH FRONT AND BACK ARE BUILT THE SAME, THE ~~EXCEPTION~~ ONLY DIFFERENCE BEING THE BACK IS WELDED PERMINENTLY TO THE FRAME, THE FRINT IS HELD ON WITH SCREWS.



P.M.S. 4-22-75

SCALE 1/2 actual SIZE

Electrolytic Basket

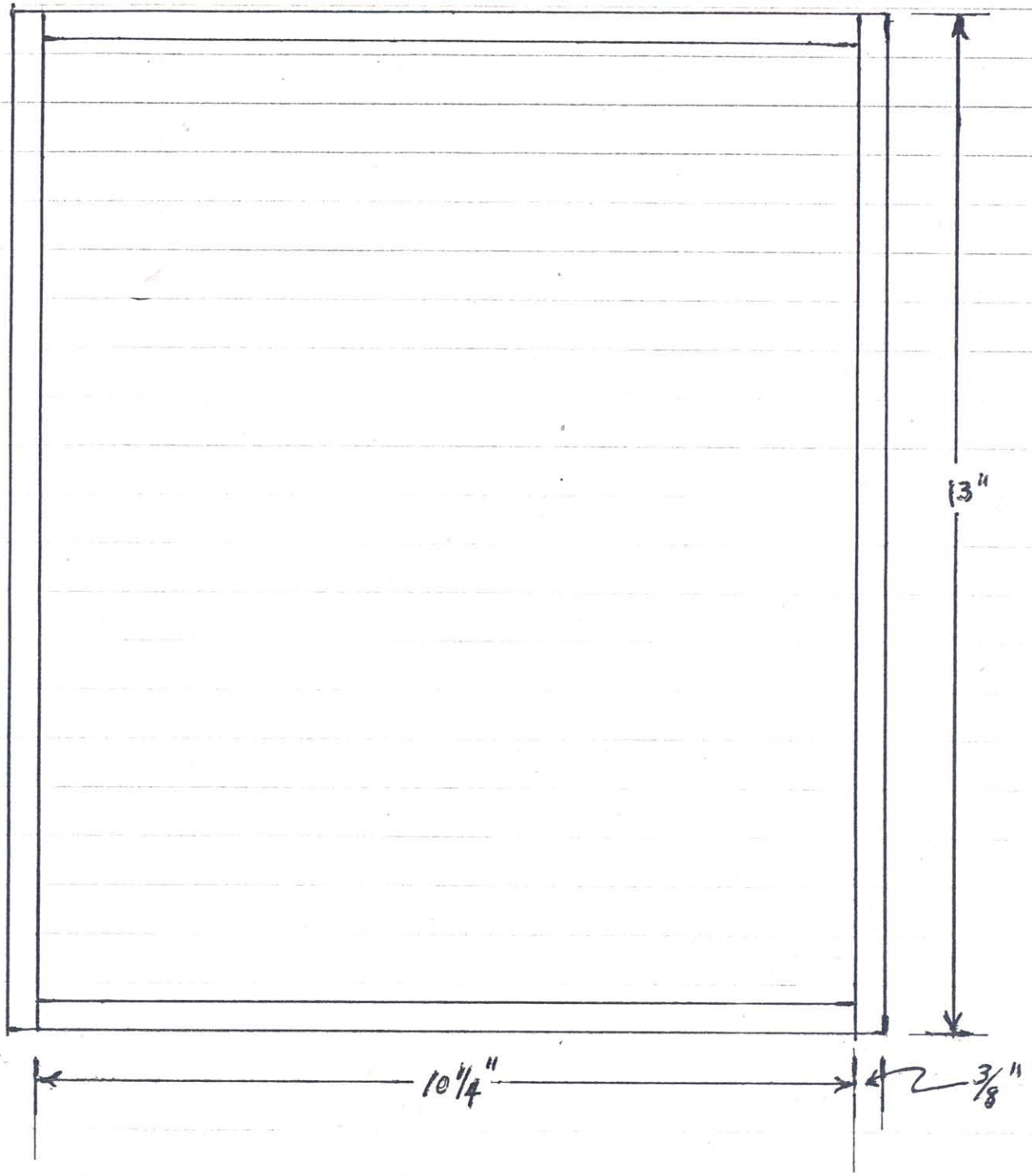


SIDE VIEW OF FRAME WITH FRONT AND BACK PLATES IN PLACE.

P.M.S. 4-22-75

SCALE = 1/2 ACTUAL SIZE.

Electrolytic Baskets



FRONT VIEW FRAME

SCALE = 1/2 actual size

P.M.S. 4-22-75

ESTIMATED BUDGET FOR STRIPPING AND CARBON REACTIVATION

1975 - 1976

Based on 82 tons of carbon 200 oz./ton assay:

174	barrels Methanol	$10,287
14	barrels flake caustic soda	1,173
160	gallons Hydrochloric acid	186
82	gallons Ammonium hydroxide	108
48	crucibles	345
24	cases steel wool	460
	Filters	70
	PVC screen (approx.)	80
	Safety supplies	125
	Oil for preheat	1,625
	Propane for carbon reactivation	2,275
	Repair and maintenance estimated	4,000
		$20,734 *

* Does not include freight, wages, or consulting fees.

Prices used are as of 9/26/75

Ken O. Mathison

To: DJG

FROM: WOM W.O.M

RE: Carbon to Idaho Mining 10-23-75

6 Barrels 200 pounds net each picked up by Idaho Mining 10-23-75

Total: 1200 pounds carbon

See attached.

ORIGINAL INVOICE

CALGON CORPORATION

CALGON

CALGON CENTER BOX 1346, PITTSBURGH, PA. 15230 (412) 923-2345

ASSIGNED MANUFACTURER'S NO. 681001 D-U-N-S 431-9910

INVOICE NUMBER
301851

PLEASE REMIT TO: BOX 34045P, PITTSBURGH, PA. 15230. REFER TO THIS NUMBER →

VIA	CUSTOMER PURCHASE ORDER NO.	DATE SHIPPED	INVOICE DATE	F	C	TERMS
TRUCK	MT-74-7068	09 30 74	09 30 74			NET 30 DAYS

SOLD TO

SUMMA CORP
PO BOX 1126
TONOPAH NEVADA
89049

SHIP TO

SUMMA CORP
TONOPAH, NEVADA

SHIPPED FROM MCKEES ROCKS PA

RELEASE NO. H008661

QUANTITY	UNIT	CODE	DESCRIPTION	UNIT PRICE	AMOUNT
4000	LB	55362	PCB 12X30 200# DRUM	.89	3,520.00
			3.50% NEV SLTX & .5% NYE CTY		123.20
			DELIVERY CHARGES TO FOLLOW		

SUMMA CORPORATION
MINING DIVISION
Tonopah
OCT 7 1974
RECEIVED

NET BY ► DATE 10 30 74 3,643.20

THANK YOU FOR USING CALGON PRODUCTS - SEE REVERSE SIDE FOR TAX REGISTRATION NUMBER.

CONSIGNEE'S COPY

DELTA LINES

California's Finest

P.O. BOX 2081 • OAKLAND, CALIFORNIA 94604

FREIGHT BILL NO. 906768

SUMMA CORP
TONOPAH NEVADA

TO BLOCK 33

L 38 387598
L GCN CORP M MCKEES ROCKS PA

FM. BLOCK 13

DATE 10 9 74 e27

CHECK	DELTA EXPRESS	EQUIP. NO.	OUR REVENUE	C. L. PAYABLE	ROUTING AND C. L. REFERENCE NUMBERS
X					

NO.	DESCRIPTION OF ARTICLES SHIPPED	WEIGHT	RATE	FREIGHT	SUR. CHG.	TOTAL CHARGES

Bill 111-

(Solution fr. Bennetts Carbon ?)

SUMMA CORPORATION
MINING DIVISION – TONOPAH, NEV.

LABORATORY REPORT

DATE 9-17- 197 5

DESCRIPTION	GROUP NO.	ozs. Au per ton	ozs. Ag per ton	Cu %	Pb %	Zn %					
?TCH #10 Head Soln	26	.3186									

SHEET NO. _____ OF _____

Bill Roberson
ASSAYER

Copy.
B. Mollison

SUMMA CORPORATION
MINING DIVISION – TONOPAH, NEV.

LABORATORY REPORT

DATE 8-26- 197 5

DESCRIPTION	GROUP NO.	ozs. Au per ton	ozs. Ag per ton	Cu %	Pb %	Zn %					
J.B. Stripped Carbon		19.624	N.I								
J.B. Stripped Carbon		20.880	N.I								

SHEET NO. _____ OF _____

Bill Robertson
ASSAYER

B. Mollison
Vegas
Bennetts

SUMMA CORPORATION
MINING DIVISION – TONOPAH, NEV.

LABORATORY REPORT

DATE 8-25- 197 5

DESCRIPTION	GROUP NO.	ozs. Au per ton	ozs. Ag per ton	Cu %	Pb %	Zn %					
B. BAR #1							.817	Au	fine		
BAR #2							.860	Au	fine		
BAR #3							.896	Au	fine		

SHEET NO. _____ OF _____

Bill Roberson
ASSAYER

Copy.
B. Mollison ✓ Bennetts
W. Simmons

SUMMA CORPORATION
MINING DIVISION – TONOPAH, NEV.

LABORATORY REPORT

DATE 8-21- 197 5

DESCRIPTION	GROUP NO.	ozs. Au per ton	ozs. Ag per ton	Cu %	Pb %	Zn %					
Bennetts CARBON #2		900.96	46.24				} 403 oz Dore at 100%				
Bennetts CARBON #3		7.4.36	57.68								

SHEET NO. _____ OF _____

Bill Robertson
ASSAYER

Copy.
B. Mollison
W. Simmons ✓

SUMMA CORPORATION
MINING DIVISION – TONOPAH, NEV.

LABORATORY REPORT

DATE 8-21- 197 5

DESCRIPTION	GROUP NO.	ozs. Au per ton	ozs. Ag per ton	Cu %	Pb %	Zn %					
Bennetts CARBON #2		900.96	46.24								
Bennetts CARBON # 3		7.4.36	57.68								

SHEET NO. _____ OF _____

Bill Robertson
ASSAYER

Copy.
B. Mollison
Vegas
Bennetts

SUMMA CORPORATION
MINING DIVISION – TONOPAH, NEV.

LABORATORY REPORT

DATE 8-25- 197 5

DESCRIPTION	GROUP NO.	ozs. Au per ton	ozs. Ag per ton	Cu %	Pb %	Zn %					
J.B. BAR #1							.817	Au	fine		
BAR #2							.860	Au	fine		
BAR #3							.896	Au	fine		

SHEET NO. _____ OF _____

Bill Robertson
ASSAYER

SUMMA CORPORATION

SUPPLIES AND LABOR TO IDAHO MINING Stripping

For carbon stripping in care of John Bennetts: 8/20 - 8/23/75

2	55 gallon drums methonal @ $74.25	$148.50
35	pounds NaOH @ $1.75	61.25
9.6	pounds copper screen 24 sq. ft.	60.48
10	6-pound HCl @ $30.55	305.50
34	hours labor W. O. Mollison	
4½	hours assay labor W. Robertson	

No charge for flux

Total excluding labor, flux:

$575.73

Original DJG
cc. WAM
Idaho Mining file in Tonopah Office

Copy.
B. Millisen
W. Simmons ✓

SUMMA CORPORATION
MINING DIVISION – TONOPAH, NEV.

LABORATORY REPORT

DATE 8-21- 1975

DESCRIPTION	GROUP NO.	ozs. Au per ton	ozs. Ag per ton	Cu %	Pb %	Zn %					
Bennetts CARBON #2		900.96	46.24								
Bennetts CARBON #3		714.36	57.68								

SHEET NO. _____ OF _____

Bill Robertson

Copy.
B. Mollison
Vegas
Bennetts

SUMMA CORPORATION
MINING DIVISION – TONOPAH, NEV.

LABORATORY REPORT

DATE 8-25- 197 5

DESCRIPTION	GROUP NO.	ozs. Au per ton	ozs. Ag per ton	Cu %	Pb %	Zn %					
J.B. BAR #1							.817	Au	fine		
BAR #2							.860	Au	fine		
BAR #3							.896	Au	fine		

SHEET NO. _____ OF _____

Bill Roberson
ASSAYER

Internal Communication

summa

Date: February 10, 1976

To: William J. Robinson

From: David K. Hamilton DKH

Subject: Improvements in Stripping Plant

The primary function of the stripping plant is the recovery of the gold carried as a cyanide complex on the surface of activated carbon. The present system, while functional, is inefficient in time and consumption of energy. Working with David Pruett on the stripping unit and William Mollison on the building and process problems I have the following suggestions for making this plant a more efficient operation. We can also look forward to lower operating costs and the virtual elimination of downtime.

1. Trailer-Mounted Carbon Tank: Presently the carbon is transported to the plant from the Manhattan Mine via flatbed truck. The truck is used, at other times, for general haulage purposes, i.e. for fuel and reagents. I suggest replacing the currently-used system with a permanently mounted tank on a trailer, used only for mine-to-plant carbon transport. Inexpensive brake controls could be installed in several trucks, any of which could

pull this unit. The tank would carry the induction pump and fittings for unloading and would have a capacity of 2000 pounds of carbon. This tank would be a rolling storage facility and, in combination with an improved stripper unit, would eliminate the need for additional storage for loaded carbon at the stripping plant.

2. Improved Stripping Unit: The condition and efficiency of the present stripping unit leaves much to be desired. Downtime, loss of solutions, and slow heating are all detrimental to the operation. A concensus design for a replacement unit has been prepared and manufacturers contacted concerning the cost of fabrication. With this new unit, we can expect a heating time reduction of at least 50%, and elimination of solution losses and much of the lost time. Further, we estimate that the complete stripping cycle can be completed in on 8-hour shift, making scheduling and security simpler.

3. Parallel Circuit Electrowinning Unit: Increased solution flows from the stripping unit, which are required to meet production schedules, make replacement of the present electrowinning cell imperative. We are proposing flow

rates of 20 GPM, while the capacity of the present unit is approximately 10 GPM. A new cell design has been proposed that will handle the higher flow rates and also provide much greater efficiency. Only a new tank will have to be fabricated since many components are common to both units. An aluminum precipitation system is being considered as an alternative to electrowinning, but until the capital and operating cost estimations and metallurgical testing have been completed and the advantages of the system shown, I suggest staying with a proven process.

4. Carbon Reactivating Unit: Much of the inefficiency of the present unit rests in inadequate storage capacity and poor slurry dewatering. At present, some 40% of the time is spent preheating the unit for the processing of a normal load of carbon. The dewatering system does not function well and is a major cause of spillage and pluggage. I suggest that a two-ton storage tank be installed in the plant to replace the present units, designed with an efficient, proven dewatering system. Drier slurries as feed to the kiln and a longer

production run will both tend to reduce costs in this process.

5. General Plant Improvements: Among the basic needs of the plant to insure an efficient operation are the following:

   A. A new furnace for fluxing
   B. Improved laboratory facilities
   C. A new floor with a sump in the solution handling area
   D. The removal of all extraneous machinery
   E. Bins for supplies
   F. Rewiring and repiping of energy supplies
   G. Standardization of valves, guages, and piping systems

Although bids, as of this time, are not finalized, rough estimates of the cost for the preceeding replacement equipment items and activities are as follows:

1.	Trailer for carbon tank	$ 500
2.	Carbon Tank (Mounted)	1,200
3.	Stripping Unit	8,000
4.	Electrowinning Unit	600
5.	Dewatering Unit	1,100
6.	Carbon Storage Tank	2,000
7.	Fluxing Furnace	2,300
8.	Laboratory Facilities	2,500
9.	Plant Floor	2,000
10.	Bins	400
11.	Valves, guages, instruments, etc.	1,200
	TOTAL:	$21,800

Upon approval of this project, a period of 8-10 weeks would be required for completion. Immediate approval would allow this work to be done while weather is hampering mine-mill operations.

Financial justification can be made by showing a tripling of refinery capacity and an 8-10% increase in plant recovery, while significantly reducing operating costs.

David K. Hamilton
Metallurgy and Planning

Distribution: WS
WOM
DKH

Files: Stripping Plant
DKH rf

summa

Internal Communication

Date: February 11, 1976

To: D. J. Gribbin

From: W. J. Robinson WJR

Subject: Refinery Improvements

Attached please find a completed Request for Major Expenditure accompanied by Mr. Hamilton's memorandum concerning refinery improvements.

The entire project can be considered normal replacement and should be started immediately. The only design changes, as far as the established process, are in the carbon stripping tank and the electrowinning unit. These changes have been examined by the consultant, Mr. Pruett, and the local Metallurgist, Mr. Hamilton. There appears to be no risk in changing the original design, only time and efficiency improvements to be gained.

Please inform me of approval or changes you desire. The final plan will be included in the Existing Facility Budget now being prepared.

Attachment

WJR:sfm

Dist: DKH
WJR
WJR rf
Stripping Plant file ✓

} without DKH memo

REQUEST FOR MAJOR EXPENDITURE

Year 197 | Schedule A1-09 Page 1

☐ ORIGINAL ☒ REVISION ☐ NOT IN ORIGINAL BUDGET

Land Exploration & Mining DIVISION | NO.

(DOLLARS ONLY)

DEPARTMENT NAME	NO.	PREPARED BY	DATE	BUDGET YR
Metallurgy		David K. Hamilton	2/11/76	1976

1. DESCRIPTION OF ITEM OR PROJECT: Improving facilities of the carbon stripping and carbon re-activating plants in Tonopah, Nevada.

2. REASON FOR EXPENDITURE:

(TAG NO. OF ITEM REPLACED)

☒ Normal Replacement ☐ Increased Volume ☐ Cost Control

☐ Early Replacement ☐ New Service ☐ Safety

☐ Profit Improvement ☐ Appearance ☐ Other (Explain)

3. JUSTIFICATION FOR EXPENDITURE: ☒ Mandatory ☐ Desirable

"See attached memorandum"

Present system is uncoded and unsafe; we may be shut down by MESA-OSHA before we can upgrade the plant.

4. OTHER ALTERNATIVES CONSIDERED: " See attached memorandum"

COST INFORMATION	CAPITAL (1)	EXPENSE (2)	TOTAL (3)
5. Outside Fees and Services		--	--
6. Outside Labor Costs		--	--
7. Purchases		21,800	21,800
8. Division Payroll		3,200	3,200
9. SUB TOTAL		25,000	25,000
10. Contingency (.....%)		3,750	3,750
11. TOTAL COST			

12. Cost Data is... ☒ Estimated ☐ Based on Bid

13. Date Required. February 1976

14. Shutdown Period from 10 days ~~xx~~

15. Revenue Loss During Shutdown $ - 0 -

16. Less -- Expense Reduction in Period $ - 0 -

17. NET REVENUE LOST $ - 0 -

RETURN ON INVESTMENT (Complete this section if reason for expenditure is Profit Improvement and append a worksheet to support entries on Lines 20 and 21.).

18. Cost $ ..... Salvage $ ..... Net

19. Estimated Useful Life ..... (Years)

EFFECT ON PROFIT

20. Increase in Annual Revenue

21. (Increase) Decrease in Annual Expense

22. Incremental Annual Profit

TO BE COMPLETED BY COMPTROLLER:

23. Payback Period _____ Years

24. Return on Investment _____ %

APPROVALS

L C and D K Hamilton (SUBMITTED)	2/11/76 (DATE)	(GENERAL MANAGER)	(DATE)
William T. Robinson (ENGINEERING)	2/11/76 (DATE)	(CORPORATE)	(DATE)
(COMPTROLLER)	(DATE)	APPROP NO.	

SPEED MEMO

To Mr. Richard Z Klosinski At BT Corporation

Date: March 8, 1976

Subject: Attached

Returned herewith is your drawing with approval and revisions.

Date Signed David L. Pruett

B.T. CORPORATION (BUTANE TANK)

LETTER OF TRANSMITTAL

3185 EAST WASHINGTON BLVD., LOS ANGELES, CALIFORNIA 90023 • (213) 261-5118

VIA

DATE March 2, 1976

TO Summa Corporation
Box 1126
Tonopah, Nevada 89049

YOUR P.O. No. SRO-43-048-220

ATTENTION Dave Pruett

OUR WORK ORDER 0677

GENTLEMEN:

• ATTACHED ARE:

_____ PROGRESS REPORTS
_____ MILL TEST REPORTS
_____ MFG. CODE REPORTS
_____ STRESS-RELIEF CHARTS
3 PRINTS EACH OF THE FOLLOWING DRAWINGS:
_____ TRANSPARENCY
_____ NAMEPLATE RUBBINGS
_____ WELDING PROCEDURES
_____ QUALIFICATION TESTS

• STATUS	• PLEASE NOTICE	• SENT FOR YOUR
_____ PRELIMINARY	_____ REVISIONS	X APPROVAL
_____ UNCHECKED	_____ ADDITIONS	X COMMENT
_____ CHECKED	_____ OMISSIONS	_____ USE
_____ FINAL	_____ CORRECTIONS	_____ FILES

Dwg. 0677-D1 48" O.D. x 14'0" oa Carbon Strip Tank
Dwg. 0677-D2 Exhaust Manifold Detail

• YOUR ATTENTION IS DIRECTED TO THE FOLLOWING:

X PLEASE RETURN ONE PRINT WITH YOUR APPROVAL OR COMMENTS NOTED THEREON.
_____ URGENT PLEASE YOUR APPROVAL OR COMMENTS.
_____ MANUFACTURE IS PROCEEDING IN ACCORDANCE WITH THESE DRAWINGS.
_____ PRODUCTION SCHEDULE REQUIRES OUR RECEIPT OF APPROVAL BY
_____ DESIGN-CHANGES OR APPROVAL-DELAY WILL EXTEND SCHEDULED SHIPPING DATE.
X THE ATTACHED ARE SENT AS REQUESTED.

VERY TRULY YOURS,

B. T. CORPORATION

Richard Z. Klosinski

Richard Z. Klosinski

RZK/sc

Land Exploration and Mining Division

Post Office Box 1126
Tonopah Nevada 89049
702 482 3584

A Division of
Summa Corporation

March 9, 1976

Mr. Bill Walker
Nevada Tank & Casing
2500 Dickerson Road
Reno, Nevada

Dear Bill:

Enclosed is a sketch of the electrowinning cell that I will want your company to build for us.

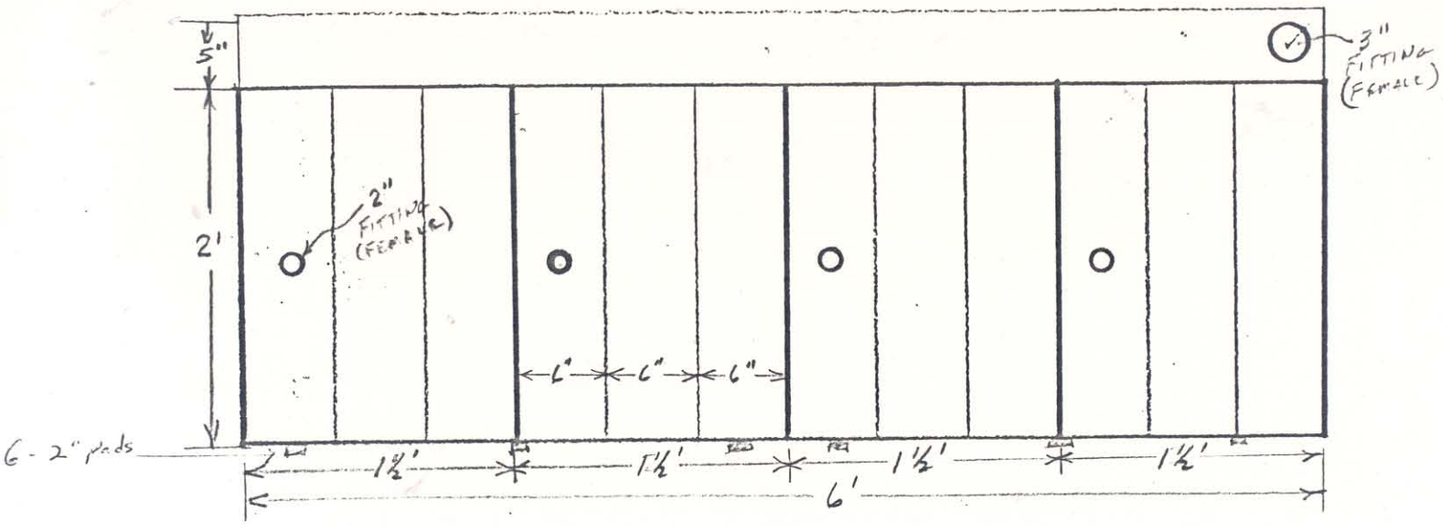
After you study this, please call and we will discuss. I still do not know the length of legs necessary to hold this but it will have to have legs.

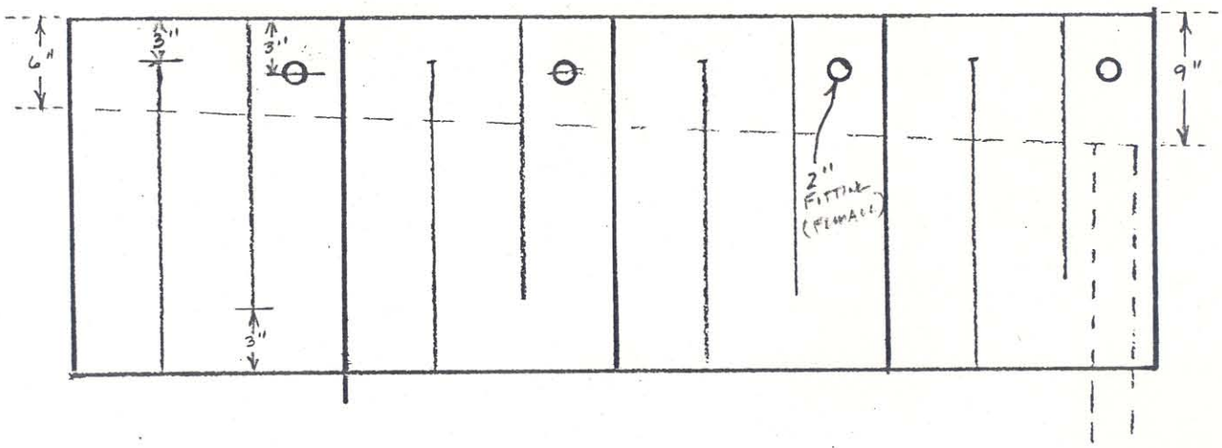
Sincerely,

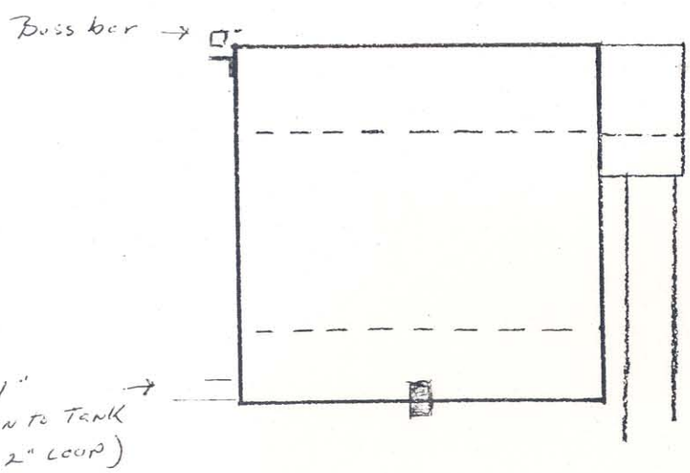
Wm O. Mollison

William O. Mollison
Refinery Superintendent

Enclosure







SUMMA CORPORATION
TONOPAH, NV

ELECTROWINNING
CELL

50 GPM CAPACITY

3-9-76

summa

Internal Communication

Date: April 14, 1976

To: David K. Hamilton

From: William O. Mollison

Subject: Stripping Plant Progress

Realizing that you all are under internal pressures, I am at the present time trying (with extremely limited help) to construct and finalize the gold stripping unit designed by David Pruett.

With the help of only one person it is impossible to meet your projected completion date of 4/19/76, especially with modifications required.

With stated deliveries from our purchasing agent and present availability of a competent welder, helper, etc., I would estimate that as of 4/14 completion would require 30 working days.

If adequate and competent help is available immediately, I'm sure this time could be halved. This half-time is also based on assured competency of outside contractors -- Logan, Skanovsky, Perchetti -- and their stated dates of completion.

Copies 9-4-75

Bill Wilson (1)
Idaho Mining
Grand & Anchan Colo.
81507

John Bennetts
Idaho Mining (2)
Box 328
Eureka, Nevada.
89316

Linda

TRIPLICATE FREIGHT BILL

DELTA LINES

3

California's Finest

P.O. BOX 2081 • OAKLAND, CALIFORNIA 94604

FREIGHT BILL NO. 3893008

CONSIGNEE
IDAHO MINING
TONOPAH NEVADA SUMMA CORP

O/T & sent to Bennetts at Eureka 10-6-75

TO BLOCK 33

3893008

SHIPPER
SIERRA CHEM 42098
RENO NEVADA

FM. BLOCK 18

DATE 091175C28

DELTA LINES	DELTA EXPRESS	EQUIP. NO.	OUR REVENUE	C.L. PAYABLE	ROUTING AND C.L. REFERENCE NUMBERS
XX					

NO. PCS.	DESCRIPTION OF ARTICLES SHIPPED	WEIGHT	RATE	FREIGHT	SUR. CHG.	TOTAL CHARGES
3	METHANOL FLAMMA LE LIQUID	1170				
3	HYDROCHLORIC ACID CORROSIVE	72				
6	TOTAL	1242	330	4099	XO COL	

I.C.C. & P.U.C. REGULATIONS REQUIRE PAYMENT OF THIS BILL WITHIN SEVEN DAYS.
REQUEST FOR INSPECTION ON CONCEALED DAMAGE OR SHORTAGE MUST BE MADE WITHIN 15 DAYS AFTER DELIVERY.
FOR PROPER CREDIT **RETURN** ONE COPY OF THIS BILL WITH YOUR REMITTANCE.

Thank You for Shipping VIA **DELTA LINES**

Met.
General - Technical Information
133

HUGHES TOOL CO. 133
METALLURGY
GENERAL TECHNICAL INFORMATION

Gold Bullion Refining

PRE-MELTING

Each deposit is melted separately in a silicon carbide crucible; the turbulence caused by the metal following the flow of the field of the high frequency current, causes an homogenous melt to be effected. The homogeneity is necessary for best results in assaying of the resultant bar. Some deposits--retorted gold, jewelers' scrap, sweepings--are melted under borax and nitre. Two small dips are taken and immediately quenched in water to prevent segregation. The lift coil is raised and the pot which rests on a carriage is moved away to the pouring table. The molten metal is poured into hot iron moulds, which have been coated with a mixture of kerosene and lampblack to prevent adhesion of the metal to the mould. The surface of the metal is immediately covered with powdered charcoal to prevent oxidation of the base metals. When the metal is set, the moulds are dumped and the ingots quenched in water, cleaned with a brush and dried. The crucible, when cold, is thoroughly scraped; the scrapings are ground and sieved and any beads of metal added to the weight of the deposit after melting.

TOUGHENING

The bullion is melted in clay crucibles and a little nitre (potassium nitrate) or sodium nitrate is thrown on to the surface of the metal. Violent bubbling at once ensues, as heat converts the nitrate into nitrite with evolution of oxygen. The nitre is pressed down with a stirrer and the nitrite and undecomposed nitrate oxidize some of the base metals. Some iron and zinc can be removed by this method. If large amounts of base metals are present, then partial refining by chlorine is suggested.

Toughening with chlorine follows the same procedure as refining with chlorine, except the gold is only brought up to 85 to 90 per cent. The bullion is melted in a clay crucible and chlorine is slowly bubbled into the molten bullion. If large amounts of

Copy - L Mallion 9-3-15

nickel or zinc are present, the addition of silver helps to start the metals making chlorides. The metal is tested from time to time by inserting a chlorine tube. Some of the gold will be forced up in the tube and the core, when cold, can be hammered. When the gold has lost its brittleness and becomes ductile, the chlorine is shut off. The melt is allowed to cool and the crucible broken away. The slag is remelted with sodium carbonate and a gold silver alloy button recovered.

The partially refined gold and gold silver alloy button are then melted together and assayed.

REFINING

The bullion is refined by the Chlorination Method. This process is based on the fact that chlorine attacks the base metals in their order as they appear in the electromotive series. The various metals present, with the exception of gold which is low on the electromotive table, combine with the chlorine at rates depending upon the heats of formation of the chlorides with a certain amount of overlapping, depending upon relative concentrations present. Lead or zinc is volatilized rapidly; soon the white fumes of lead, if present, are replaced by darker fumes of cuprous chloride; copper and iron are only partly volatilized and the formation of liquid silver chloride starts soon after. The early silver chloride is more cuperiferous than that formed toward the end of treatment— if the temperature is raised above the critical point, some gold chloride is formed. A small amount of free gold is carried over in the volatilization of the base metals.

REFINING OPERATION

The tilting furnaces reserved for rough gold melting are charged with a mixture of rough gold deposits of about 15,000 ounces, sufficient to supply the 24 furnaces; and, when molten, a red hot chlorine pot, which has previously been annealed in an electric furnace,

is taken from the refining furnace and filled to about two-thirds its capacity (about 650 ounces). It is then placed in a graphite guard pot resting on a portion of fire brick on the bottom of the red hot refining furnace. The guard pot catches any leakage which might result from a defective pot. Sufficient powdered borax glass, which has been melted in the chlorine pot before filling, forms a layer of about 3/4 inch on top of the molten metal; the lid is placed on the pot and a pipe stem, which has previously been heated to a red heat, is inserted into the molten metal through the slot in the lid and the hole in the furnace cover. The stem is pushed down to the bottom of the pot and held there by a metal clamp attached to the wall of the chamber immediately above the furnace.

The chlorine is admitted by slowly opening the control valve until the gas is passing through the molten metal in a steady stream. This can be observed by the pulsation felt by holding the rubber tubing between the thumb and fingers.

Volatile chlorides of zinc, iron, lead, copper, etc., (metals which are nearly always present), are formed and dense fumes of these chlorides are immediately evolved. As they pass off, the supply of chlorine is gradually increased until the full pressure is reached without any danger of splashing, for the silver absorbs the chlorine completely, no bubbles of gas escaping from the molten metal.

Toward the end of the operation when nearly all the silver has been converted into chloride, the supply of chlorine is gradually reduced to a mere trickle to prevent splashing while the last traces of base metals which remain to the end are being removed. The finishing point is gauged from experience by the colour of the gas issuing from the slot in the lid of the pot or, if in doubt, by the characteristic canary yellow stain formed on a piece of cold pipe stem held in the gas leaving the pot and the smell of free chlorine. If continued after this point, gold is attacked and gold chloride escapes with the chlorine.

When the refining is finished, the flow of gas is stopped, the furnace opened, clay lid and pipe stem removed and silver chloride containing base chlorides floating on the top of the gold is baled out with a small triangular clay crucible into iron moulds twelve inches square, two inches deep, while the pot is in the furnace. If the rough gold being refined contains a large amount of silver, it is necessary to stop the flow of gas before refining is finished and bale off some of the chloride of silver to prevent overflowing from the pot due to the silver almost doubling its bulk on conversion to chloride. The surface of the gold is cleaned by an addition of a little bone ash which forms a paste with the remaining chlorides and is scraped off with a flat stirrer.

The pot is then removed from the furnace and the molten refined gold, approximately 500 ounces, is poured into the tilting furnace reserved for fine gold melting, which is kept at a little above the melting point of gold. The chlorine pot is immediately filled with another charge of molten rough gold from the rough gold tilting furnace and refining continued.

This furnace has been recharged and the charge melted while refining is taking place in the twenty-four furnaces, so there is a constant supply until the last round of refinages for the day. By having a constant supply of molten rough gold, refining is practically continuous, only being stopped during the time taken to bale off the chlorides, etc. Also, by mixing deposits containing much silver and little base with those containing much base and little silver, a more suitable bullion is obtained for refining. Two refinages can be made in the same pot.

When the fine gold tilting furnace has received the refined gold from one round of refinages (24 furnaces) - approximate weight about 10,000 ounces, varying with the quality of rough gold being refined - the molten gold is thoroughly stirred with a spade graphite stirrer.

The pouring table is wheeled into position and the gold poured into iron moulds resting on the table, each holding 400 ounces. The weight of the fine gold ingots (trade bars) must not be less than 390 and not more than 410 ounces. The man operating the tilting furnaces judges the amount by the eye and from practice becomes very expert, less than one per cent being rejected for being out of remedy. As one mould is filled, the table is revolved slowly until the next mould is under the lip of the crucible. A natural gas flame is immediately played on the surface of the gold as it sets to prevent oxidation of traces of base metals remaining in the gold and to retard the surface crystallization of the gold, thus producing a bright surface on the ingot. When the gold is set, the moulds are dumped and the ingots quenched in water containing a little sulphuric acid in a copper tank washed with water, dried and stamped with their respective serial numbers.

Small sample bars of about twenty ounces, from which cuts are subsequently taken for assay, are poured (1) after stirring and before casting the first ingot, and then after every tenth until just before casting the last ingot when the final sample bar is poured.

ELECTROLYSIS OF GOLD

The electrolytic process used in producing fine gold at the Royal Canadian Mint was developed by Wohwill. It was first used at the Norddeutsche Refinery, Hamburg, in 1878.

The process consists in electrolyzing gold anodes in a hot acid solution of gold chloride. At the refinery, the solution is kept at approximately 60°C., contains seven to eight per cent gold chloride and 100 grns. per litre of free HCl. A thorough stirring of the electrolyte is necessary and this is accomplished by agitating the solution with compressed air under a pressure of 10 lbs./sq. inch. Each cell contains 23½ litres of H_2O, 2½ litres of $AuCl_3$ and 1½ litres of hydrochloric acid. The current density in each cell is 100 amperes per square foot of anode surface.

There are six gold cells in a unit in the refinery. The cells are of white porcelain and the size is 12" x 12" x 16". The cells are heated by means of an oil bath (white oil #48, 160°F.) which surrounds them. Each cell is contained in a cupboard arrangement with lift windows over each cell. In order to prevent corrosion of the electrical connections, they are placed outside the cells at the front of the cupboard. The fumes from the cells are removed from the cells by a forced draught.

The cathodes and anodes are supported in each cell by sterling silver rods and hooks. These rods and hooks must be cleaned every day in order that a proper contact can be maintained. As it is necessary to use D.C. current in the cells, selenium rectifiers have been installed in the electrolysis room.

The cathodes in each cell consist of thin strips of rolled gold produced from fine gold assaying 999.9. The anodes are relatively pure (over 99%) and are cast with a hole in the top in order that they may be suspended in each cell by means of the sterling silver hooks mentioned above. If a relatively pure anode is not used, anode passivity will occur owing to the build-up of an adherent film of silver chloride on the anode surface and, subsequently, there will be a rapid depletion of gold from the electrolyte.

It should be mentioned that when a current is passed through any liquid conductor, decomposition or electrolsis takes place. The conducting liquid is called the electrolyte and the plates or poles used to lead the liquid in and out are called electrodes. The positive electrode at which the current enters the liquid is called the anode and the negative plate is called the cathode. The electrolyte is said to be decomposed into ions. In the Wohlwill process, Au, Cu, Pb, Pt and Pd dissolve in the electrolyte; silver and the other platinum metals (Ir, Rh, Ru) remain chiefly in the slime--the former as AgCl. It is possible to recover the gold from the electrolyte by reducing it with sulphur dioxide. The platinum and palladium can be recovered by precipitation with ammonium chloride.

The cathodes are removed from the electrolyte when they reach a reasonable weight and average 999.9 fine. They are washed thoroughly by water under pressure in order to eliminate any silver chloride. The washings are added to the cells again to take care of the evaporation. After washing the cathodes are dried and then melted into bars weighing not over 410 oz. troy, and not under 390 oz. troy.

Present practice at the Refinery is to change the electrolyte periodically. The cells are, however, cleaned out every 1,000 hours running time. The slimes obtained are dried and melted into ingots from which cuts are taken for assay. These scrap bars assay 91 per cent gold approximately. They undoubtedly contain some metals of the platinum group.

When the electrolytic cells are in operation 24 hours a day, considerable evaporation takes place. It is, therefore, necessary to add acid and gold chloride every morning. The operator makes these additions until he obtains a voltage of one and a current of 150 amperes in each cell. Also, when the cells are run for 24 hours a day, short circuits may develop. It is, therefore, imperative to make certain each morning that no bridging takes place between the anode and the cathode and also that the slimes do not build up at the bottom of each cell until they touch the anode or the cathode.

CHLORIDES FROM REFINAGE

Physical Characteristics:

The chlorides when bailed off from the refinages are cast, forming a dense crystalline tough mass. They are usually dark coloured externally, but with a lighter coloured fracture which is rapidly affected by light. On exposure to air, carbon dioxide, moisture, etc., they become green and are frequently deliquescent. The chloride slag is found to contain gold in the form of minute crystals of gold reduced from chloride of gold and large amounts from the bailing off the chlorides.

Treatment:

There are four distinct operations necessary in the treatment of chlorides:

(a) Removal of the gold

(b) Removal of the chlorides of the base metals

(c) Reduction of the silver chloride

(d) Melting of the silver

(a) Removal of the Gold:

There is no difficulty in separating the bulk of the gold; indeed some of it may be separated by simple fusion. The silver chloride contains about 8 per cent gold, varying with the fineness of the rough gold chlorinated. The cakes of chloride are broken up and melted in 50 lb. lots in a Morgan #45 crucible. The temperature of the melt is a little above the melting point of silver. To this fusion, we slowly scatter on the surface about 5 lbs. of sodium carbonate. The pot is allowed to stand in the furnace until the gold-silver alloy button has become solid. The silver-gold alloy button is then chlorinated. The crucible is taken from the furnace and the dross, consisting of borax and slags, is skimmed from the top of the chloride with a flat iron skimmer. The crucible of silver chloride is then emptied into a graphite pot with a 5/8 inch hole set on 25 gallon jar filled with water. The grain size of the granulated silver chloride is regulated by the height of the crucible above the water and the opening in the crucible.

$$4\ AgCl + 2\ Na_2\ CO_3 - 4\ Ag + 4\ NaCl + 2\ CO_2 + O_2$$

(b) Removal of the Chlorides of the Base Metals:

1. The mixture of chlorides obtained from the chlorination room is treated with bleaching powder $Ca\ OCl_2$ - this compound is a very strong oxidizing agent and converts the copper (cuprous) chloride to soluble cupric chloride.

2. The action of the bleaching powder with the solution of chlorides evolves nascent chlorine. This nascent chlorine, at the moment of its liberation by a chemical reaction, shows much greater chemical activity than ordinary chlorine gas. There are various

theories as to why nascent chlorine is more active chemically than gaseous chlorine.

One theory is that gaseous chlorine consists of atoms combined into molecules; therefore, work has to be done in converting these molecules into atoms before chemical reaction can take place. Nascent chlorine is, presumably, liberated in the form of atoms, and these need no work done on them but are at once ready to react. It is also true that the whole of the energy liberated in the reaction producing chlorine does not appear as heat but a part of it increases the chemical energy of the chlorine atom as liberated.

After the soluble cupric chloride and the lead chloride have been removed by washing, it remains to reduce the AgCl with the use of iron plates.

TREATMENT OF THE FLUE GASES

The flue gases, carrying the gold and silver and other metals in suspension from the gold and silver refining, are passed through the Cottrell electrical precipitator.

The installation consists of two units of Cottrell precipitators, one above the other, operated in parallel--known as the rod curtain type.

The flow of gases from the main duct is divided into approximately two equal streams by a damper in the distributing chamber to the units. Each unit consists of an air-tight steel shell divided into four sections in series. At the entrance of the first and third sections are perforated steel plates with one inch holes through which the gas flows to obtain a more even distribution. In each section the electrodes and collecting electrodes are suspended from an insulated frame.

The units are energized by the high voltage current—the discharge electrodes being negative and the collecting, positve. With a negative discharge, a higher current can be used without danger of a disruptive discharge or arcing. The particles of suspended matter in the gas in the intervening space between the discharge and collecting electrodes receive a charge of electricity of the same

polarity as the

discharge and, being repelled, flow to the collecting electrode where the charge is neutralized and there they accumulate.

The units are put in operation before starting the furnaces in the morning and shut down when all melting has ceased at the end of the day. A damper in the outlet flue is closed immediately after shutting down to prevent moisture entering. The collecting electrodes are then rapped by means of mechanical rappers operated by motors, and the accumulated particles drop into the chambers at the bottom of each section. These chambers are cleaned out once a month and the precipitate mixed and sampled. The guaranteed efficiency is an extraction of not less than 98 per cent of all suspended matter in the gases with a rated capacity of 25,000 cubic feet per minute.

BY-PRODUCTS

Gold and Silver:

The gold sweep consists of discarded crucibles, furnace linings and dust collected by the Cottrell Precipitator. The sweep is crushed to $\frac{1}{4}$ inch in a jaw crusher and passes down through a 6 inch pipe to a storage bin and then fed into the Hardinge ball mill by a weigh machine, the feed load being about 100 lbs. per hour.

The ball mill is a 2 foot size cone crusher containing 800 lbs. of steel balls. On the end of the crusher is a conical shaped screen of 40 mesh. The minus 40 mesh passes through the screen into a carriage and the oversize is caught in a trough at the end of the screen.

The sweep is then fed into the mixer which is a steel drum 5 feet in diameter by 2 feet, with a conical-shaped extension. The interior of the cylindrical portion is fitted with steel blades radiating from the axis and set at such an angle that the sweep is projected to the cone and gravitates inwards to the blades again. The mixer is filled with 3,000 lbs. of sweep and mixed for about 16 hours.

Below the mixer in the iron pipe is an automatic sampler, driven by a small motor which takes a portion of the whole stream all the time the mixer is discharging into the drums. The sample is assayed for gold and silver, and the sweeps are sold to smelters.

HEAT CALCULATIONS

Present Desorbtion Unit:

30" diameter x 10' long x 5/16" walls

6" heat exchanger x 10' long

Burner 1.65 gal./hr. @ 135,000 btu./gal. = 222,000 btu./hr.
Electrical strip heat 30 kw. = 102,000 btu./hr.
Total = 324,000 btu./hr.

Temperature raise 60°F to 200°F (140°F/3 hrs.)

Radiant surface area 13 sq.ft./pipe
5 sq.ft./elec. A = 18 sq.ft.

Input heat 324,000 btu./hr.
Radiation loss 140,000 btu./hr. (calculated)

Btu -- applied 18,000 btu./sq.ft. exchange area
-- exchanged 10,000 btu./sq.ft.

Overall coefficient heat transfer = .57

550 gal. of solution + carbon/975,000 btu/1 hr./140°F (rise)
or 1772 btu./gal.

Planned Unit:

Using the heat transfer figures from the present stripper, 1,950,000 btu./140°F (rise)/hr. would be required for a unit with 1150 gal. of solution + 2000 lb. carbon.

The efficiency of this unit will be much greater.
1,950,000 btu. = 14.4 gal. of fuel oil.
3 2.5 gal./hr. burners = 7.5 gal./hr.
2 hours would be required for preheat.
Improved heat exchange and insulation should cut this to estimated 1 hour 20 minutes.

HEAT CALCULATION CONVERSIONS

1 hp. = 0.745 kw. = 42.4 btu./min. = 2544 btu./hr.

1 boiler hp. (bhp.) = 33.475 btu./hr.

1 kw. = 1.34 hp. = 56.88 btu./min. = 3413 btu./hr.

1 btu. = .029 kw./hr.

1 cu.ft. water = 62.4 lb. @ 60°F.
1 gal. = 8.34 lb. water @ 60°F.
1 cu. ft. = 7.48 U. S. gal.

fuel oil = 135,000 btu./gal.

propane = 87,000 btu./gal. liquid

Solution Conversion

Gals.	
1 =	.004167 Tons
2	.008334 ✓
3	.012501 ✓
4	.016668 ✓
5	.020835 ✓
10	.04167 ✓
25	.104175 ✓
50	.20835 ✓
100	.41670 ✓
150	.62505 ✓
175	.72922 ✓
200	.83340 ✓
240	1.00000 ✓
300	1.2501 ✓
400	1.6668 ✓
500	2.0835 ✓
600	2.5002 ✓
700	2.9169 ✓
800	3.3336 ✓
900	3.7503 ✓
1000	4.1670 ✓

Example: Solution Assay .02 Au

at 500 Gal. min. = 2.0835 tons

2.0835 x .02 = .04167 oz. Au min.

.04167 x 60 = 2.5002 " " hr.

2.5002 x 24 = 60.0048 " " 24 hrs.

60.0048 x 30 = 1800.144 " " 30 Days

Solution 8.3333 lbs. per Gal.

FLOW OVER 60-DEGREE TRIANGULAR NOTCH WEIR

Head in Ft.	Head in Inches	Flow Cu.Ft/Sec.	Flow Gal/Min.	Tons/Min.
0.13	1 9/16	.008	3.6	.015
0.14	1 11/16	.010	4.5	.019
0.16	1 7/8	.012	5.4	.022
0.17	2 1/16	.016	7.2	.030
0.18	2 3/16	.019	8.5	.036
0.20	2 3/8	.022	9.9	.041
0.21	2 ½	.026	11.7	.049
0.22	2 5/8	.031	13.9	.058
0.24	2 7/8	.035	15.7	.066
0.25	3	.040	18.0	.075
0.26	3 1/8	.046	20.6	.086
0.28	3 3/8	.052	23.3	.097
0.29	3 ½	.058	26.0	.109
0.30	3 5/8	.065	29.2	.122
0.32	3 13/16	.072	32.3	.135
0.33	3 15/16	.080	35.9	.150
0.34	4 1/16	.088	39.5	.165
0.36	4 5/16	.096	43.1	.180
0.37	4 7/16	.106	47.6	.198
0.38	4 9/16	.115	51.6	.215
0.39	4 11/16	.125	56.1	.234
0.41	4 7/8	.136	61.0	.255
0.42	5 1/16	.147	66.0	.275
0.43	5 3/16	.159	71.4	.298
0.45	5 3/8	.171	76.7	.320
0.46	5 ½	.184	82.6	.344
0.47	5 5/8	.197	88.4	.369
0.49	5 7/8	.211	94.7	.395
0.50	6	.226	101.0	.423
0.51	6 1/8	.240	108.0	.449
0.53	6 3/16	.256	115	.479
0.54	6 ½	.272	122	.509
0.55	6 5/8	.289	130	.541
0.57	6 7/8	.306	137	.573
0.58	6 15/16	.324	145	.607
0.59	7 1/16	.343	154	.642
0.61	7 5/16	.362	162	.678
0.62	7 7/16	.382	171	.715
0.63	7 9/16	.403	181	.754
0.64	7 11/16	.424	190	.794
0.66	7 15/16	.445	200	.833
0.67	8 1/16	.468	210	.876
0.68	8 1/8	.491	220	.919
0.70	8 3/8	.515	231	.964
0.71	8 ½	.539	242	1.009
0.72	8 5/8	.564	253	1.056
0.74	8 7/8	.590	265	1.104
0.75	9	.617	277	1.155
0.76	9 1/8	.644	289	1.206
0.78	9 3/8	.672	302	1.258
0.79	9 ½	.700	314	1.310
0.80	9 5/8	.730	328	1.367
0.82	9 13/16	.760	341	1.423
0.83	9 15/16	.790	355	1.479
0.84	10 1/16	.822	369	1.539
0.86	10 5/16	.854	383	1.599
0.87	10 7/16	.887	398	1.660
0.88	10 9/16	.921	413	1.724
0.89	10 11/16	.955	429	1.788
0.91	10 15/16	.991	445	1.855
0.92	11 1/16	1.03	462	1.928
0.93	11 1/8	1.06	476	1.984
0.95	11 3/8	1.10	494	2.059
0.96	11 ½	1.14	512	2.134
0.97	11 5/8	1.18	530	2.209
0.99	11 7/8	1.22	548	2.284
1.00	12	1.26	566	2.359

Apparently granular 16-30 mesh carbon

SUMMA CORPORATION

METALLURGICAL TEST DATA SHEET

TEST NO. 1 DATE: 9-11-75

Objective: To determine the weight of 1 gallon of Dry Carbon, Let Soak for 48 Hours and determine Wet weight

Product	Weight or ml.	% Weight	Assay Oz./Ton Au	Assay Oz./Ton Ag	Content Au	Content Ag	Distribution Au	Distribution Ag
DRY CARBON	1 gallon = 3.786 Litres 1561 Grams (3.441#)							
WET CARBON	1 gallon 3816 grms 8.412#							
moisture content	4.971 #/gallon							

Procedure: 3.786 Litres = 1 gallon
453.6 grams = 1 pound

3,786 Litres of Carbon (DRY) were weighed out by Volume to establish a 1 gallon point. The dry gallon of Carbon having a weight of 1561 grams or 3.441 # was then saturated completely in distilled water and left to sit for 48 Hours.

Conclusion: 1 gallon dry Carbon Weighs 3.441 pounds — After dry Carbon is Soaked for 48 hours the Weight of the Carbon is 8.412 # — Therefore 1 gallon of dry Carbon will Absorb 4.971 # moisture —

Information, For general use.

At the present time C.P. Hydrochloric Acid is being used in the refinery to treat steel wool This acid is 36% to 37% HCl

Muriatic acid also Hydrochloric Acid is 36% HCl It should be used instead of the C.P. acid.

Cost basis.

1 - 6lb bottle C P acid - $8.85 = $1.475/lb.

Muriatic Acid in carboys = $0.1466/lb.

There is a deposit on the carboys so they must be returned. Deposit is $25.00 for 15 gallons

~~Hold~~

c.c. - Walt Simmons ✓
Bill Mullison
Bill Robertson
Purchasing Dept

General Information

Call from George Potter -
U.S. Bureau of Mines
Salt Lake City.
Re — Carbon Loading

Potter supplied the following information -

1.- Using reactivated carbon
Loading was 300 oz/ton - Doré
Barren Solutions of 0.0002 oz/ton Au were obtained
They used -10 + 20 mesh reactivated carbon
4 stages of loading recovered the gold
8 stages were needed to recover the silver
They used a 5 column in series set-up
Flow rate of 15 gallons/sq ft/min gave a suitably expanded bed using this carbon.
The total length of the expanded bed in the column was 4 ft.
This gave a retention time of approximately 2 minutes per column for a total of 10 minutes
5% of the orifice plate is holes

Reference Book -
R. J. Davidson
The mechanism of gold adsorption
Journal of South Institute of Mining - Metallurgy.

Zerox copies -
Mollison -
Siklenga
Simmons ✓
Baker
Robertson -
File

TECHNICAL INFORMATION July 14, 1975

Theoretical amount of Na_2S needed to precipitate silver as silver sulfide (Ag_2S):

Molecular Weight Ag = 108
Molecular Weight S = 32

$2Ag^+ + Na_2S \rightarrow Ag_2S + 2Na^+$

$2 \times 108 + 78 \rightarrow 248 + 46$

Therefore, 78 units of Na_2S will precipitate 248 units of Ag_2S or 1 pound of Na_2S should precipitate 3.18 pounds Ag_2S.
Ag_2S is $\frac{216}{242}$ = 87.1% Ag

So, 3.18 x 87.1% = 2.77 pounds of Ag is precipitated by 1 pound Na_2S

Or, conversely, $\frac{1}{2.77}$ = 0.369 pounds of Na_2S should precipitate 1 pound of silver or 14.57 troy ounces.

Then convert avoirdupois pounds to troy ounces:
1 pound Av. = 14.57 troy ounces

So, 1 pound Na_2S should precipitate 40.36 ounces Ag.

Please confirm and check. This figure is probably a minimum.

R. E. Baker

Dist: R. E. Baker
W. O. Mollison
Tech. Info file

Lab Reports

July 15, 1975

TO: Walt Simmons

FROM: R. E. Baker

SUBJECT: Laboratory Equipment -- Bottle Rolls

It is felt that bottle roll testing in cyanidation work offers the most rapid method of obtaining preliminary and necessary information.

A set-up whereby a multiplicity of tests can be run at the same time has the following advantages:

1. Several samples can be cut at one mixing. If irregular free gold content is a problem, it will be indicated by irregular assays, as well as non-conforming metallurgical balances (computed head assays). Take averages.

2. Assuming no free gold problems, then a series of tests can be run varying time of leach.

3. Another series should be run varying cyanide strengths at optimum time.

4. Another series should next be run in which the caustic (or lime) strength is varied at previously determined optimum conditions.

5. Only after the conditions have been determined should column tests be started.

6. The leaching of coarser ore, in a column will involve a longer time to effect extraction; so, while this is being done, tests on another sample of the same

ore from a different section of the deposit should be run under optimum conditions which have been previously determined.

7. Screen analyses of residues from the above tests will determine the nature of the ore and the gold occurrence. Titrations on each test will show reagent consumption.

8. It should be possible for one man to manipulate six bottle roll tests during the course of a day, so it is felt that a set of bottle rolls to accommodate six bottles should be obtained. The labor cost and time to obtain information will easily justify the cost of the equipment.

9. The laboratory staff is endeavoring to maintain cleanliness. However, there is still some of the equipment that should be cleaned of residual carbon that might or might not have precious metal values adhering to it. This can be gone over as time allows. Laboratory equipment should be cleaned both before and after use. The ore samples are, in general, low grade and it does not take very much gold to make a salt that would render the test unreliable.

It is only with tedious and repetitive work that design criteria can be established.

As time goes on, more testing equipment might be suggested, but it is felt that at present ample bottle roll facilities are of paramount importance.

Respectfully submitted,

Robert E. Baker

REB:sfm

Dist: DJG
WOM
CS
Files: Lab Reports, R. E. Baker

Bill - Clarence.

1. A detailed instruction on NaOH Na$_2$S titration is located in The bulletin. on. Hydrometallurgy of Mercury - Nevada Bureau of Mines .. by Botts - & Fyte.

2.

FLOW THROUGH V NOTCH IN PREG.- POND - MEASURED FROM

-BOTTOM OF BOX TO TOP OF WATER-

HEAD	FLOW IN GALLONS PER MIN.
7-3/16"	3.6
7-5/16"	4.5
7-7/16"	5.4
7-9/16"	7.2
7-11/16"	8.5
7-13/16"	9.9
7-15/16"	11.7
8-1/16"	13.9
8-3/16"	15.7
8-1/4"	18.0
8-3/8"	20.6
8-1/2"	23.3
8-5/8"	26.0
8-3/4"	29.2
8-7/8"	32.3
9"	35.9
9-1/8"	39.5
9-1/4"	43.1
9-3/8"	47.6
9-1/2"	51.6
9-5/8"	56.1
9-3/4"	61.0
9-13/16"	66.0
9-15/16"	71.4
10-1/16"	76.6

GAUGE CHART

10,000 Gallon, 95" x 336" Horizontal Storage Tank, Pacific Coast Standard.

Calibrated in Inches

Inches	Gallons	Inches	Gallons	Inches	Gallons
1	19.0	33	3184.0	64	7388.0
2	54.0	34	3316.0	65	7517.0
3	98.0	35	3449.0	66	7645.0
4	149.0	36	3582.0	67	7772.0
5	208.0	37	3716.0	68	7897.0
6	273.0	38	3852.0	69	8021.0
7	341.0	39	3987.0	70	8143.0
8	417.0	40	4124.0	71	8263.0
9	495.0	41	4260.0	72	8383.0
10	579.0	42	4398.0	73	8500.0
11	666.0	43	4534.0 (?)	74	8616.0
12	755.0	44	4672.0	75	8730.0
13	848.0	45	4811.0	76	8842.0
14	945.0	46	4948.0	77	8951.0
15	1045.0	47	5086.0	78	9057.0
16	1147.0	47½	5155.0	79	9163.0
17	1252.0	48	5224.0	80	9265.0
18	1359.0	49	5362.0	81	9364.0
19	1468.0	50	5499.0	82	9461.0
20	1580.0	51	5638.0	83	9555.0
21	1694.0	52	5775.0	84	9644.0
22	1810.0	53	5912.0	85	9731.0
23	1927.0	54	6049.0	86	9815.0
24	2046.0	55	6186.0	87	9893.0
25	2167.0	56	6322.0	88	9969.0
26	2289.0	57	6453.0	89	10,037.0
27	2413.0	58	6598.0	90	10,102.0
28	2538.0	59	6727.0	91	10,160.0
29	2665.0	60	6861.0	92	10,212.0
30	2793.0	61	6993.0	93	10,256.0
31	2922.0	62	7126.0	94	10,291.0
32	3052.0	63	7258.0	95	10,310.0

Sent: 7-11-75

Mr. Gribbin —

The Lab Counter Top is 24 feet long and 24 inches wide

The Horizontal Back where the Gas Valves + ect Are is 12½ inches Tall — 24 feet in length

The Flat Top Portion is 11½" Wide and 24 feet long.

Bill

ASSAY GROUP CLASSIFICATIONS

AU - GOLD

AG - SILVER

CU - COPPER

PB - LEAD

ZN - ZINC

Au Residue Flux

75%	Precipitate
45%	Borax
45%	Niter
25%	Silica
5%	Fluorspar

4/24/75

GOLD AND SILVER CONVERSION CHART

ppm	Oz/T	ppm	Oz/T	ppm	Oz/T
1....	.029	34....	.992	67....	1.95
2....	.058	35....	1.02	68....	1.98
3....	.088	36....	1.05	69....	2.01
4....	.117	37....	1.08	70....	2.04
5....	.146	38....	1.11	71....	2.07
6....	.175	39....	1.14	72....	2.10
7....	.204	40....	1.17	73....	2.13
8....	.233	41....	1.20	74....	2.16
9....	.263	42....	1.23	75....	2.19
10....	.292	43....	1.25	76....	2.22
11....	.321	44....	1.28	77....	2.25
12....	.350	45....	1.31	78....	2.28
13....	.379	46....	1.34	79....	2.30
14....	.408	47....	1.37	80....	2.33
15....	.438	48....	1.40	81....	2.36
16....	.467	49....	1.43	82....	2.39
17....	.496	50....	1.46	83....	2.42
18....	.525	51....	1.49	84....	2.45
19....	.554	52....	1.52	85....	2.48
20....	.583	53....	1.55	86....	2.51
21....	.613	54....	1.58	87....	2.54
22....	.642	55....	1.60	88....	2.57
23....	.671	56....	1.63	89....	2.60
24....	.700	57....	1.66	90....	2.63
25....	.729	58....	1.69	91....	2.65
26....	.758	59....	1.72	92....	2.68
27....	.783	60....	1.75	93....	2.71
28....	.817	61....	1.78	94....	2.74
29....	.846	62....	1.81	95....	2.77
30....	.875	63....	1.84	96....	2.80
31....	.904	64....	1.87	97....	2.83
32....	.933	65....	1.90	98....	2.86
33....	.963	66....	1.93	99....	2.89
				100....	2.92

ppm x 0.029167 = troy oz/ton

1 troy oz/ton = 34.28 ppm

12/10/74