MAR 19990001: BAD HEART SANDSTONE

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19990001

A REPORT AND BACKGROUND INFORMATION
PERTINENT TO THE EXPLORATION
AND ANALYSIS OF
THE BAD HEART SANDSTONE ON
PERMITS 9396110003 and 9396110004

1999 0001

JAN 21 1999

INDEX

January 14, 1999

Ronald T. Owens
[redacted]

Alberta Energy
Mineral Operations Division
Minerals Tenure Branch
9th Floor, 9945 - 108 Street
Edmonton, AB T5K 2G6

Att: Hazel Henson
Agreement Administrator

I hereby submit an Assessment Work Report to cover the required expenditures for sections nineteen through thirty-six of 6-09-079 of Metallic Minerals Permit No. 9396110003 and sections one through eighteen of 6-09-080 of Permit No. 9396110004.

These properties appear to have relatively small lenses of the Bad Heart Sandstone. Samples taken from eight locations, consisting of two pit areas, three drill holes and three outcrops have shown very low values of free gold.

Since 1990 I have submitted samples from numerous locations of the Bad Heart Sandstone to twenty-three different labs and quasi-labs. In the interest of brevity, I will not submit the results of the majority of these tests. They were generally inconclusive. I will submit the results of tests, pertinent to the fore mentioned land locations. The majority of samples tested have been taken from a pit on section twenty-six of permit 9396110003. The other sites tested, indicated a general similarity in content.

I have found it necessary to establish my own lab facilities, to be able to send portions of prepared samples, while retaining accuratly split portions for follow up analysis. The tests that show positive results, although done in most cases by persons with an impressive resume, are suspect, because they invariably have a "Recovery Process" to market.

I have found it necessary to travel extensively to meet in person most of the lab personnel that I have dealt with. Therefore I am submitting a portion of these travel expenses, as I considered it necessary in view of the wide-ranging results received.

I have come to the conclusion that some locations of the Bad Heart Sandstone contain a very complex asssemblage of minerals and elements. There is no doubt that there were errors caused by interferences in some of the instrumental analyses.

Ronald T. Owens

Alberta Energy
Mineral Operations Division

Att: Hazel Henson

Further work is required to identify and eliminate these interferences.

I respectfully submit the following list of expenses, test results, resumes of some of the test consultants and reference data that I consider useful to the exploration of the Bad Heart Sandstone.

Hewlett Mineral Management	$ 16,816.91
Process Research	1,200.00
Complex Metals Research [redacted]	2,409.36
Bahamian Refining	1,956.27
Metallurgical Research	1,205.95
Trinity Systems (Iron beneficiation)	451.96
Chemtron Labs	262.00
Lilbrat Mining	600.00
Loring Labs	346.00
50% of $27,000 cost of lab equipment	13,500.00
reagents, glassware & fusion expendables	3,400.00
Travel costs	4,000.00
Estimated 1000 hrs. @ $20.00	20,000.00
Total	$ 66,148.45

[redacted]

Ronald T. Owens

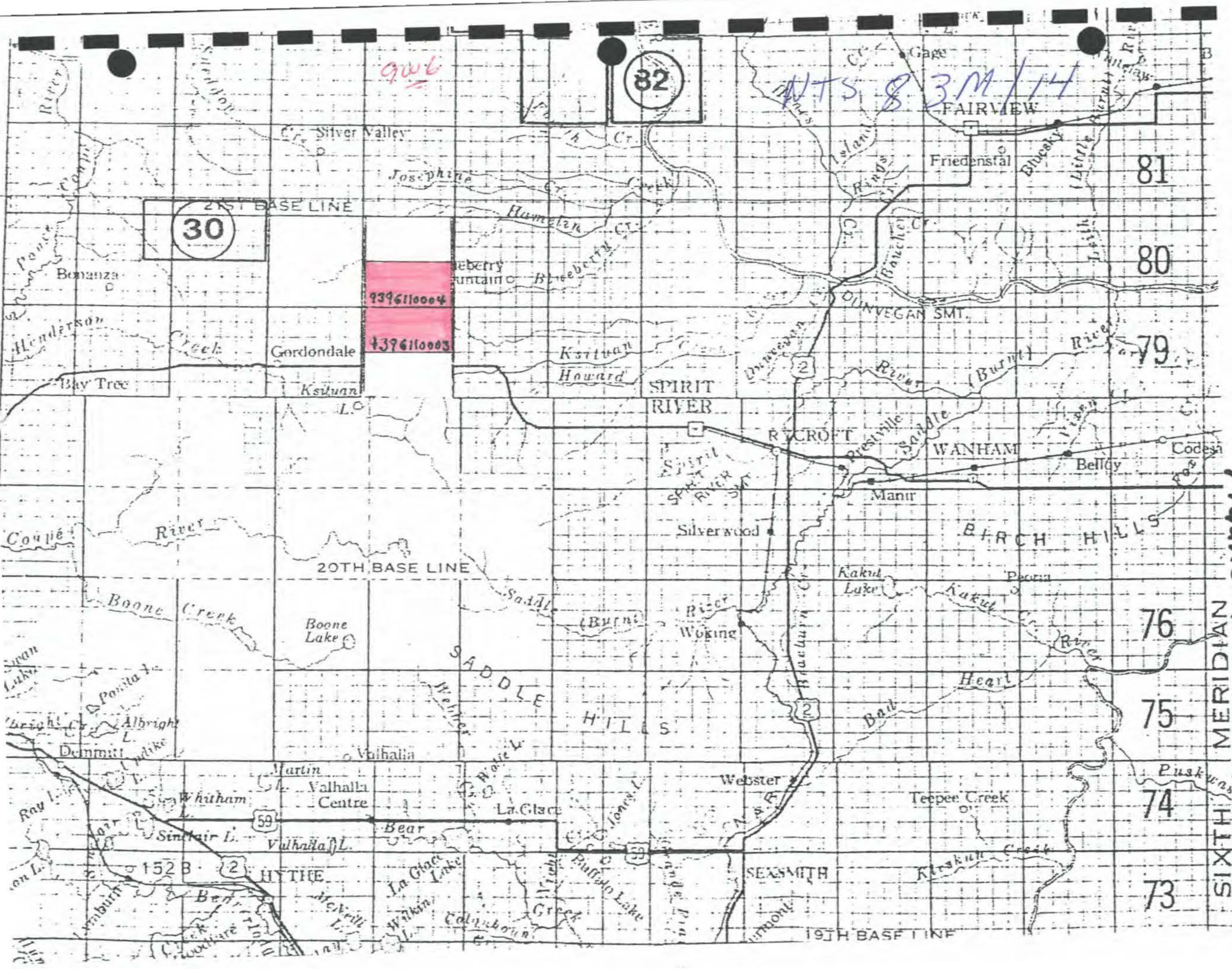

Figure #1 / 19970001

APPENDIX

TO

METALLIC AND INDUSTRIAL MINERALS PERMIT NO. 9396110003

COMMENCEMENT OF TERM:

1996 NOVEMBER 5

AGGREGATE AREA:

9 216 HECTARES

DESCRIPTION OF LOCATION:

6-09-079: 1-36

PERMITTED SUBSTANCES:

METALLIC AND INDUSTRIAL MINERALS

SPECIAL PROVISIONS:

NIL

APPENDIX

TO

METALLIC AND INDUSTRIAL MINERALS PERMIT NO. 9396110004

COMMENCEMENT OF TERM:

1996 NOVEMBER 5

AGGREGATE AREA:

9 216 HECTARES

DESCRIPTION OF LOCATION: NTS 83m/14

6-09-080: 1-36

PERMITTED SUBSTANCES:

METALLIC AND INDUSTRIAL MINERALS

SPECIAL PROVISIONS:

NIL

R. F. HEWLETT
(619) 384-2647 FAX (619) 371-4619

1539 N. China Lake Blvd., Ste. #153
Ridgecrest, CA 93555

HEWLETT MINERAL MANAGEMENT

January 10, 1996

Mr. Ronald Owens
[redacted]

Dear Ron:

Proposed is that I exclusively consult for you and your Peace River property.

The objective of these studies are to prove the ore reserves of the deposit, to define the processing flow-sheet, and the economics for the recovery of the precious noble metals, as well as the economic recovery of other elements.

For example, the following results I obtained recently from Activation Laboratories in Ancaster, Ontario.

Sample #	Gold(T.oz./ton) Epithermal Neutron	Thermal Activation	Silver T.oz./T Fire Assay	Cu PPM	Pb PPM	Zn PPM	Ni PPM	Mn PPM
A80	0.452	0.402	0.207	19.	162.	206.	25.	95.

Cd PPM	V PPM	Ca %	P %	Mg %	F %	S PPM	Br PPM	As PPM	Co PPM
0.50	357.	2.14	0.527	0.58	0.12	490	21	380	18.

W PPM	Th PPM	La PPM	Ce PPM	SIO_2 %	Al_2O_3 %	Fe_2O_3 %	MnO %	CaO %	LOI %
45	7.4	25.4	54	42.63	2.30	34.50	0.14	3.11	12.79

Pt PPM	Pd PPM
8.7	23.9

Notice from the "Whole Rock Analysis" that the loss on ignition (LOI) was 12.79% and the silica+alumina was 44.93%.

Therefore, with magnetic separators and electro-static separators, the ore could be up-graded from 34.50% ironoxide to 62.65% Fe_2O_3 and by the heating/roasting in the R-N process, the LOI is removed and the Fe_2O_3 would be a theoretical 81.60% Fe_2O_3.

This could be accomplished as a sub-set of the recovery of the precious and noble metals and other economic elements.

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HEWLETT MINERAL MANAGEMENT

Proposed that I be retained for a fee of $10,000.00, which would allow me to up-date my facilities and to be able to control my direction of samples that I would pre-treat to various laboratories, with a view to provide reliable assay data and ore reserves that could be used for further funding, possibly by a public offering.

The thrust of this proposal is to provide the basic analysis required to finalize the process chemistry and flow-sheet and to then conduct large-scale testing at our facility. This would be the location for our bulk-sample test (200#-to-many tons).

+++

I have had experience with your ore as follows:

1) Results of various types of gravity concentration:
    - a) Diester vibrating tables
    - b) Dunham vibrating tables
    - c) Riechert Spirals
    - d) Knelson Rotary concentrators

2) Process Chemistry:
    - a) Fire Assay
    - b) Bromine/chlorine leaching in my reaction chamber
    - c) Neutron Activation:
        - a. Epithermal
        - b. Thermal

3) Mineralogy/Petrology:
    - a) "Free Gold" distribution by size fraction
    - b) "Locked Gold" by size fraction
    - c) Chemical composition

4) Secondary Concentration Testing:

   a) Eriez High-Intensity magnetic separator
   b) Eriez Low-Intensity magnetic separator
   c) Carpco Electro-Static separator

5) Geochemical Analysis:

   a) "Pathfinder" elements with/without statistical correlations
      a. "Free Gold"
      b. Oolitic (encapsulated) gold

6) Proposed Process:

   a) Unit Operations
   b) Process Chemistry
   c) Products:
      1. Gold-silver and Platinum Group Metals (remember that PCM's are not soluble in cyanide - they DO NOT form a complex with cyanide.) The process must recover all precious/noble metals economically and much faster than common leaching - as cyanide.

         My process allows for selective separation of the gold-silver-and-platinum group metals or they can be melted into one dore.

      2. Up-graded iron-ore.

      3. Other elements to consider (my process will recover them at very little additional expense.
         a. Bismuth
         b. Tellurium
         c. Base metals
         d. Nickel/Cobalt/Cadmium
         e. Rare Earth Elements
         f. Tungsten, etc.

Examples of the above results referred to are as follows:

1) Gravity concentration.
I have had very good success with a Dunham vibrating table - with 5-end riffles-- The highest-grade gold is in the #1 riffle, and the #2 riffle is only 23.42% of the #1 riffle grade, and the #3 riffle is only 1.11% of the #1 riffle grade.

Examples -- cont'd

No gold was found in the #4 - #5 riffles or in the tailings. (One-pass--another pass would have reduced the #2 grade.

I have made a simple concentrate with the Dunham table of over 10-troy ounces Gold/topn on a one-pass basis.

2) Metallurgical Testing Results:

Slimes from the 6-26-79-9 pit were fire assayed and a split was analyzed using my bromine/chlorine chemistry. The results follow:

	FIRE ASSAY (t./oz./ton)		BROMINE (t.oz./ton)	
	GOLD	SILVER	GOLD	SILVER
Pit -50 slimes	0.008	0.003	0.203	0.056

Please note that this is the most dramatic increase in detectable gold values I have done. Reasons for this difference could be:

a) Excessive slimes (ferruginous) that cause a ferrous oxide layer at the interface of the metal phase (collector) and the "collected-gold" lead-sponge.

b) High Te, Bi, or Se. They form alloys with the gold that do not collect in lead (or other collector)--Au-Te/Bi/Se alloy goes intop the slag--speiss--or from the button into the cupel.

Also, Ir/Os have the same effect on gold as above.

(Note that As-Sb form "speiss" and they average 300 ppm As, and 20 ppm Sb with one high of 4900 pp, Sb.

(Also, that sample had 24 ppm Sn).

The Bromine/chlorine leach is "non-toxic" and much faster than cyanide or any other non-toxic leach. The costs are much less than cyanide and the process is so flexible that the operator is in control--instead of the process being in control of the operator.

3) Mineralogical Examination:

a) Pit Ore: Dunham Table concentrated. #01 riffle

+20 mesh	Oolites black/brown--minor qtz. (near top of Bad Heart fm).
+28 mesh	One native gold-Pt. Pt is in the center of an elongate gold grain.
+48 mesh	3-gold grains, 1 with Au-Pt.
+80 mesh	Tiny gold grains--one with silver alloy.

4) Secondary Concentration Testing:

I have used our low-intensity mag separator for separating ferromagnetics such as magnetite.

I have used (on a routine basis) the high-intensity mag separator to then separate sulfides.

Then, I have used the electro-static separator to separate such minerals as scheelite, silics, alumina, etc.

Also, the different intensity mag separators will give selective separating for iron minerals (siderite, geothite, notronite, ferruginous opal, hematite/limonite, etc.) when used with the electro-static separator.

5) Geochemical Analysis:

Elements with a high positive statistical association with gold are:

A) Silver
B) Antimony
C) Cobalt
D) Tungsten
E) Tin

By taking out the "free gold" the remaining chemical analysis describes the "complex-gold" or encapsulated gold. This aspect will be more precisely defined.

5) Geochemical Analysis--cont.

Geochemical criteria for "placer gold" and chemical precipitated gold would be expected to be determined.

6) The process chemistry used is economical and has all the advantages of being very flexible and it is controlled with ease.

I have made 300:1 concentrates with the Peace River ore and with a large-scale operation (500 tons/hour), that would be little over 2-tons/hr. of concentrate, or 13.3 tons of concentrate per 8-hr. shift.

This con. would go the following:

```
500-tons/hour head ore
!
Concentration (Spirals/rotary--later variable concentrator
!            !---tails
Cons.
1.67 tons/hr.)
!
Low-Intensity MAG Sepr.
!                    !
"Non-Mag"       Ferro-mag-----magnetite
!
High-Intensity MAG Sepr.
!                    !-----Non-Mag
ParaMag
Sulfides
```

6) continued

```
 Non-Mag      ParaMag(Sulfides)   Magnetite
      !             !                  !


      !       leach if economic   leach if economic
      !
      !
 "Non-Magnetic" iron oxides/carbonates/silicates/
               chamosite/W-Au
               !
        Electro-Static Separator
        !                     !
     conductors          non-conductors
     Iron cons./         Silica/alumina/W
     Au-Ag-PGM
     !
     !
     Selective magnetic separators
         !                !
     Iron conc.        Au-Ag-PGM conc.
                            !
                         Reaction Chamber
                         (Oxidation)
                           !
                          Br/Cl dissolution
                          of Au/Ag/Pt/Pd
                          and rest of PGM
                            !
                           ION-EXCHANGE
                           (Selective Chelating           COLUMNS OF
                           Ion-Exchange Resin-            3-cubic feet
                           "Loads" only precious/         resin each:
                           noble metals                   100 oz./cu. ft.
                           !                  !-------!
                           Elution--reuse resin;      !
                           after simple regeneration  !-------Effluent:
                           !                                  recover
                          Recover precious/noble metals       with other
                           selectively or one dore            1-X resins
                                                              other elements/
                                                              metals
```

By the time the process reaches the final leaching stage, the amount of concentrates is greatly reduced and one 8-hour shift precious/noble metal concentrates can be processed in that same period of time (8-hours).

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1539 N. China Lake Blvd., Ste. #153
Ridgecrest, CA 93555

HEWLETT MINERAL MANAGEMENT

April 29, 1997

Mr. Ronald T. Owens
[redacted]

Dear Ron:

After a long delay, I have received most of my assay analyses for your "Bad Heart Iron Formation Ore Deposit".

In summary, I have been spending my time analyzing each size fraction for four ore-types/locations. These were:

1) P-100: 6-26-79-9 pit pulverized by RTO to minus 100 mesh.
2) Top two (2) feet of 6-26-79-9 pit exposure.
3) General coarse samples from the pit.
4) Sample site some distance by road from the 6-26-79-9 pit.

Each size fraction was analyzed by numerous methods, such as:

1) Conventional fire assay
2) Bromine-leach
3) Bromine-leach-pre-treatment -- then fire assay
4) Neutron activation:
   a) Epithermal
   b) Thermal
5) Lead platinum group collection/mass spec.
6) Nickel sulfide platinum group collection/mass spec.

After analyzing each size fraction by various analytical methods and prior concentration, the best/most accurate results were used after computing the ore grades for the head-ore.

Previous results were as follows:

Sample ID	Troy oz./ton Fire Assay		Sample ID	Bromine leach/t.oz/ton	
	Gold	Silver		Gold	Silver
A-Glom	11.150	2.190	A-50slimes	0.203	0.056
A80	0.452	0.402	A-50	0.135	0.023
A+20 mesh	0.87	0.05	A-S	0.023	0.010
A+28 mesh	1.46	0.22	A-S	0.042	0.014
A+48	0.22	0.038	A-R	0.038	0.009
A-80	0.54	0.122			

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Ridgecrest, CA 93555

HEWLETT MINERAL MANAGEMENT

Ronald T. Owens: April 29 (2)

Following are the very recent analysis for your ore deposit:
(Platinum group will follow)

Sample ID		Gold (ppb)/ton	Weighted Average Gold/Ton (Converted to Head Ore)
P-100	+28 mesh	1,380.	
P-100	+48 mesh	204.	
P-100	+200 mesh	198.	
P-100	-200 mesh	8,730.	0.180 troy ounces Gold/Ton.
P-2-TOP	+28 mesh	344.	
	+ 48 mesh	36.	
	+200 mesh	290.	
	-200 mesh	26,200.	0.110 troy ounces Gold/Ton.
P	+28 mesh	4,890.	
	+48 mesh	34.	
	+200 mesh	94.	
	-200 mesh	37,000.	0.090 troy ounces Gold/Ton.
R-TOP	+28 mesh	753.	
	+48 mesh	9.	
	+200 mesh	149.	
	-200 mesh	18,700.	0.030 troy ounces Gold/Ton.

THESE RESULTS HAVE PROVEN TO DEFINE AN ECONOMIC ORE BODY--THAT REQUIRES CONCENTRATION/SCREENING/BROMINE LEACHING FOR AN ECONOMIC OPERATION.

SIGNED: [redacted]

RICHARD F. HEWLETT

[redacted] YL[redacted] LA[redacted] IN[redacted]
1775 W. Sahuaro Dr. • P.O. Box 50106
Tucson, Arizona 85703

JOB NUMBER WQR 004B
January 12, 1995
Sombrio Fine Gray Mags
PAGE 5 OF 8 PAGES

HEWLETT MINERAL MANAGEMENT
Attn: Mr. Richard F. Hewlett
1539 N. China Lake Blvd., Suite 153
Ridgecrest, CA 93555

REPORT OF ANALYSIS

Analysis of 37 Rock Chip Samples

ITEM	SAMPLE NO.	FIRE ASSAY Au (oz/t)	Ag (oz/t)	Sb (ppm)	As (ppm)	Bi (ppm)	Te (ppm)	Hg (ppm)
24	BG 8 - CC Qtz	.055	< .01	< 2.	38.	< .1	< .1	.01
25	BG 8 - Black (Qtz)	6.120	.62	< 2.	44.	1.2	.1	.12
26	A +20m	.870 ✓	< .01	15.	300.	.5	.3	.07
27	A +28m	1.460 ✓	.22	3.	165.	.1	.1	.10
28	A +48m	.220 ✓	< .01	7.	190.	.6	.3	.06
29	A +80m	.670 ✓	.02	7.	245.	.2	.2	.44
30	A -80m	.540 ✓	< .01	9.	290.	.2	.1	.09
31	M 1	.110	< .01	4.	125.	.8	.7	.05
32	M 2	1.000	.15	6.	125.	.3	.1	.06
33	M 3	7.270	1.72	10.	410.	.5	.1	.07
34	M 5	.780	.16	9.	155.	.6	< .1	.06
35	M 9	.135	< .01	4.	70.	.7	.2	.03
36	M - Mag	.100	< .01	10.	160.	2.2	.2	.03
37	M - 4 Non-Mag	.215	< .01	8.	110.	2.0	.2	.04

Charles E. Thompson   William L. Lehmbeck   James A. Martin

[redacted]
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JOB NUMBER WQR 004B
January 12, 1995
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PAGE 8 OF 8 PAGES

HEWLETT MINERAL MANAGEMENT
Attn: Mr. Richard F. Hewlett
1539 N. China Lake Blvd., Suite 153
Ridgecrest, CA 93555

REPORT OF ANALYSIS

Analysis of 37 Rock Chip Samples

ITEM	SAMPLE NO.	Cr* (ppm)	Mg* (%)	Ba* (ppm)	Ti* (%)	B* (ppm)	Al* (%)	Na* (%)	K* (%)	W* (ppm)	Be* (ppm)
24	BG 8 - CC Qtz **	638	.01	16	.01	18	.08	.01	.01	<2	.4
25	BG 8 - Black (Qtz) ++	505	.01	37	.01	83	.06	.01	.01	<2	.4
26	A +20m	49	.88	520	.02	<5	1.52	.10	.34	<2	6.0
27	A +28m ++	67	.64	478	.01	111	1.23	.08	.28	44	2.9
28	A +48m **	83	.72	638	.03	76	1.17	.07	.24	23	5.0
29	A +80m **	96	.53	527	.02	64	1.01	.06	.20	25	4.2
30	A -80m	130	.44	348	.03	<5	.93	.05	.20	38	2.7
31	M 1 **	160	.19	172	.11	<5	.79	.05	.15	11	1.8
32	M 2 **	151	.19	209	.27	<5	.87	.10	.22	52	2.0
33	M 3 ++	117	.15	217	.19	14	.62	.07	.15	110	.1
34	M 5 **	152	.19	257	.27	<5	.93	.10	.23	10	1.9
35	M 9 **	134	.23	197	.16	<5	.96	.07	.17	5	2.3
36	M - Mag	132	.20	206	.15	<5	.39	.05	.03	7	2.4
37	M - 4 Non-Mag	226	.20	208	.38	<5	1.20	.17	.42	108	1.6

"*" Denotes partial digestion in aqua regia of elements commonly found in geological materials.
+NOTE: Greater than normal geochemical range. Please advise if fire assay is needed.
++NOTE: Samples analyzed on WQR004-A.
**NOTE: Samples analyzed on WQR004-B.

Charles E. Thompson
William L. Lehmbeck
James A. Martin

JOB NUMBER WQR 004B
January 12, 1995
Sombrio Fine Gray Mags
PAGE 7 OF 8 PAGES

HEWLETT MINERAL MANAGEMENT
Attn: Mr. Richard F. Hewlett
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REPORT OF ANALYSIS

Analysis of 37 Rock Chip Samples

ITEM	SAMPLE NO.	Mn (ppm)	Fe (%)	Sr (ppm)	Cd (ppm)	Bi (ppm)	V (ppm)	Ca* (%)	P* (%)	La* (ppm)
24	BG 8 - CC Qtz **	37	1.22	3	.5	<3	9	.01	.01	4
25	BG 8 - Black (Qtz) ++	373	1.20	2	.5	<3	22	.01	.01	1
26	A +20m	2180	29.99	20	1.0	<3	225	2.99	1.00	216
27	A +28m ++	1661	20.36	205	2.5	<3	279	2.65	.85	18
28	A +48m **	1842	25.16	192	3.2	23	499	3.30	.74	20
29	A +80m **	1336	21.07	174	2.9	10	431	2.84	.74	20
30	A -80m	1009	17.69	20	2.0	5	355	1.81	.55	149
31	M 1 **	775	10.19	55	.5	<3	319	.70	.07	22
32	M 2 **	1284	11.68	78	.5	<3	386	1.21	.09	39
33	M 3 ++	1115	12.62	58	.5	<3	449	.81	.12	30
34	M 5 **	904	11.54	80	.5	<3	379	.93	.07	40
35	M 9 **	950	13.33	66	.5	<3	447	.82	.08	27
36	M - Mag	1906	35.49	24	1.5	<3	1029	.78	.29	44
37	M - 4 Non-Mag	672	7.87	42	.8	<3	222	1.13	.08	113

Charles E. Thompson
Arizona Registered Assayer No. 9427

William L. Lehmbeck
Arizona Registered Assayer No. 9425

James A. Martin
Arizona Registered Assayer No. 11122

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January 12, 1995
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PAGE 6 OF 8 PAGES

HEWLETT MINERAL MANAGEMENT
Attn: Mr. Richard F. Hewlett
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REPORT OF ANALYSIS

Analysis of 37 Rock Chip Samples

ITEM	SAMPLE NO.	Ag (ppm)	As (ppm)	Sb (ppm)	Cu (ppm)	Pb (ppm)	Zn (ppm)	Mo (ppm)	Ni (ppm)	Co (ppm)
24	BG 8 - CC Qtz **	.1	29	<2	5	4	13	2	20	3
25	BG 8 - Black (Qtz) ++	21.9	38	<2	8	11	17	1	19	35
26	A +20m	1.8	315	45	15	210	235	<1	29	24
27	A +28m ++	.9	167	16	14	76	255	5	38	74
28	A +48m **	1.3	212	15	22	124	346	7	44	95
29	A +80m **	4.3	265	8	17	102	299	4	42	69
30	A -80m	4.2	267	24	15	151	239	6	26	42
31	M 1 **	1.2	70	10	36	75	81	2	19	32
32	M 2 **	4.1	129	8	27	89	69	3	16	44
33	M 3 ++	27.4	543	11	29	334	76	3	10	35
34	M 5 **	4.6	154	9	25	75	76	4	14	38
35	M 9 **	1.1	67	3	34	46	78	3	20	45
36	M - Mag	1.6	80	34	25	126	154	<1	5	39
37	M - 4 Non-Mag	2.0	100	13	36	39	50	3	10	25

Charles E. Thompson Arizona Registered Assayer No. 9427	William L. Lehmbeck Arizona Registered Assayer No. 9425	James A. Martin Arizona Registered Assayer No. 11122

HEWLETT MINERAL MANAGEMENT

R. F. HEWLETT
(619) 384-2647 FAX (619) 371-4619

1539 N. China Lake Blvd., Ste. #153
Ridgecrest, CA 93555

RESUME

EDUCATION:

1957: B.S. in Chemical Engineering; Iowa State University.

1960: M.S. in Mining Engineering; University of Arizona.

1967: Course work and Dissertation completed for D. Sc. in Mining Engineering and a Ph. D. in Geological Engineering while on the teaching faculty of the Colorado School of Mines and the University of Arizona.

EXPERIENCE:

1980 to date: *Acquisition of extensive California Placers and GOLDFIELD MINES. Completed exploration, development, and feasibility studies on all of these mining properties.*

Consultant for numerous precious metal operations; milling and heap leaching[development and metallurgical design of various ore treatment systems for varied and complex ores].

Consultant for numerous gold placer mines[development, evaluation and metallurgical design of recovery systems utilizing advanced gold recovery technology].

1979 to 1980: *General Partner; Gold-Silver development and heap-leach operation at Goldfield, Nevada.*

1978 to 1979: *Consultant for major gold-silver heap leach in Nevada; 100,000 tons-per-month under-leach operation.*

1970 to 1978: *General Partner and Chief Executive Officer;*

A). Sierra Mineral Management/Merchant Sterling Joint-Venture with PLACER DEVELOPMENT; McDermitt Mercury Mine[Nevada].

$10,000,000 mine development--Mine went into production in 1975 and is still in production.

Limited Partners are the Harriman Trust [Merchant Sterling], Warren Corning, the BARR BROS[Bond dealers], Flagler Mathews, etc..

B). '71 Minerals; Tombstone, Arizona gold-silver heap-leach operation. Over 4,000,000 tons of dump and open-pit ore was leached in the first four years.

The Limited Partner was the HARRIMAN TRUST.

[2]- RESUME OF R.F. Hewlett

C). GOLD-CREEK JOINT-VENTURE; Cripple, Creek, Colorado gold-silver heap-leach operation. Similar to our Tombstone operation in that 150,000 tons-per-month were put under leach.

Another Limited Partnership with the Harriman Trust.

D). STERLING VENTURE; Nelson, Nevada. Gold-Silver heap-leach from open-pit mining.

Another Limited Partnership with the Harriman Trust.

1968-1970: Mine Finance; Consultant on Aquisition and Mine Valuation- [Zurich, London, Rome, Johannesburg, Paris, Belem(Brazil), and New York].

1963 to 1968: Chairman of the Board; COMPUTEC RESEARCH, GFI COMPUTE INDUSTRIES, COMPUTERIZED NATURAL RESOURCES, and R.F. HEWLETT & ASSOCIATES. Computer Application Companies with world-wide consulting in mining operations and other natural resource based ventures.

1957 to 1963: Bear Creek/Kennecott; Statistical Engineer. Developed computer applications for ore reserve computations, design of underground and open-pit mines utilizing econometric analysis/optimization, exploration drilling patterns, financial evaluation of exploration projects, production scheduling, and process optimization for copper smelting.

Yuba Mining Company; Exploration, mine examination, and design for open-pit gold-silver deposits in Idaho.

Utah Construction; Mine engineer for underground mining of oil shale in Colorado[Union Oil].

Kaiser Aluminum; Bauxite exploration, mine examination, fluorspar deposit evaluation, engineering and plant process in Jamaica, Panama, Brazil, and the United States.

[3]- Resume of R.F. Hewlett

1970 to 1978 JOINT VENTURES :

1970 PLACER AMEX; Mercury at Cordero Mine near McDermitt, Nevada.

1970 NORANDA MINES; Walker Mine, Plumas County, CA [Copper, gold, and silver].

1970 NORANDA MINES; Copper/gold at Hassayampa Mine area, Prescott, Arizona.

1970 U.S. SILVER; Copper, gold, and silver at MacKay, Idaho.

1970 NEW YORK ROSARIO; Silver and gold operation at La Colorada Mine, Sonora, Mex..

1971 TENNECO; Porphyry copper exploration in Arizona, California, and New Mexico.

1971 QUINTANA MININERALS; Porphyry copper exploration at Coyote Wash, Safford, AZ.

1971 AMERCIAN METALS CLIMAX; Porphyry copper explorat near Silver Bell, AZ.

CONSULTING EXPERIENCE BY COMPANY

American Metal Climax
American Potash & Chem Corp.
American Smelting & Refining
Anaconda Company
Banner Mining Company
Behre-Dolbear
Cerro De Pasco
Cities Service
Compania Minera De Cananea
Duval Corporation
El Paso Natural Gas
Ethyl Corporation
Granby Mining Company
GranIsle Mines
G.T. Bator & Associates
Hanna Mining Company
Irish Base Metals
Kaiser Aluminum & Chemical Corporation
Kennecott Copper Corporation
Kingdom of Saudi Arabia
Miami Copper
Minera Bayovar
Molybdenum Corporation of America
Noranda Mines
Occidental Petroleum(Minerals)
Placer Development
P.C.A.N.T.(Portugal)
Quintana Minerals
Revere Copper and Brass
Superior Oil Company
Swindell-Dressler
Texas Gulf Sulphur
Transvaal Consolidated Goldfields
Trans World Airlines

[4]- RESUME OF R.F. Hewlett

CONSULTING EXPERIENCE BY COMMODITY

GOLD

Virginia Mine[Transvaal, So. Africa]
Loraine[Transvaal, So. Africa]
Cripple Creek[Colorado]
Goldfield[Nevada]
Thunder Mtn.[Idaho]
Nelson[Nevada]
Congress[Arizona]
Elkhorn[Montana]
Mojave[California]
Chloride[Arizona]
Virginia City[Nevada]
Oro del Rey[Utah]
Merker[Utah]
Central City[Colorado]
Wickenburg[Arizona]
Crown King[Arizona]
Oatman[Arizona]

SILVER

Packard Mine[Nevada]
Tonopah[Nevada]
Calico[Barstow, CA]
Candelaria[Nevada]
Ojos Calientes[Mexico]
Zacatecas[Mexico]
Tombstone[Arizona]
Silver Plumb[Colorado]
Eureaka/Ely[Nevada]
Rey Mart[Arizona]
La Colorada[Mexico]
Leadville[Colorado]
Searchlight[Nevada]
Prescott[Arizona]
Yucca[Arizona]
Pierce-Gleason[Arizona]
Reville[Nevada]

COPPER-MOLY-GOLD

Safford, Arizona[KCC]
Tanama, Puerto Rico[KCC]
Lights Creek, CA[AMEX]
Robinson District, NV[KCC]
Diamond H, Miami, AZ
San Xavier North[ASARCO]
Ray Mines, AZ[KCC]
Granisle, B.C.[Granby]
Michiquillay, Peru[ASARCO]
Jabal Sayid, Saudi Arabia
Ruby Creek, Alaska[KCC]
Kalamazoo, AZ[Quintana]
Silver Bell, AZ[ASARCO]
Cananea, Mex.[Anaconda]
Cajo Abaho, P.R.[AMAX]
Lampa, Peru[ASARCO]
Galore Creek, B.C.[KCC]
Lakeshore, AZ[El Paso]
Marcopper, Philippines
Castle Dome, AZ[Cities]
Newman, B.C.[Nornada]
San Manuel, AZ.
Twin Buttes, AZ[Anaconda]

COMMODITY	LOCATION
CU-ZN-PB-AG	Timmins, Ont.[Texas Gulf]
"	Cerro De Pasco, Peru
NICKEL-CU	Ragland Nickel[Quebec]
"	Pikwe, Africa
SULPHUR	Benguet, Philippines
URANIUM	Homestake-Sapin[New Mexico]
BRINES/SOLID	Trona, CA
PB-ZN-AG	Northgate, Ireland
"	Salem, Missouri[KCC]
"	Viburnum, Mo.[ASARCO]
"	Terre Monte, Portugal
COPPER-IRON	Craigmont, B.C.[PLACER]
"	Boss-Bixby, Mo.[Am. Pb-Zn]
PHOSPHATE	Florida
"	Tunesia
"	Sechura Desert, Peru
IRON	Colima, Mexico[HYLSA]
MOLYBDENUM	Climax, Colo.[AM METALS CL]
"	York Hardy, B.C.[ASARCO]
"	Questa, N.Mex.[MOLY CORP]
"	Endako, B.C.[PLACER]
"	Hall, Nevada[Anaconda]
"	Alice Arm, B.C.[KCC]

PUBLICATIONS

1). 1970 - "Comparison of the Triangular, Polygonal, and a Statistical Method of Computing Grade and Tonnage of Ore for the Silver Bell Oxide Porphyry Copper Deposit".

2). 1969 - "Case Histories of Exploration Drilling and Ore Reserve Estimation"; Geological and Mining Society of British Columbia, Vancouver, B.C., Canada.

3). 1968 - "Case Histories of Automated Ore Reserve Estimates"; Annual Meeting of the AIME, New York.[In collaboration with Dr. A. Banfield of BEHRE-DOLBEAR].

4). 1968 - "New Techniques in Prospecting for Coal"; American Mining Congress, Annual Meeting of the Coal Division, Pittsburg, Penn.

5). 1965 - "Open Pit Mine Design Utilizing a Digital Computer"; Annual Meeting of the American Institute of Mining, Metallurgical and Petroleum Engineers, Chicago, Ill.

6). 1965 - "Design of Drill-Hole Grid Spacing for Evaluating Low-Grade Copper Deposits"; Report of Investigation, U.S. Bureau of Mines.

7). 1964 - "Future Computerization in the Mineral Industry"; International Symposium on Application of Statistics, Operations Research, and computers in the Mineral Industry, COLORADO SCHOOL OF MINES, Golden, Colo.

8). 1964 - "Empirical Models of a Copper Reverberatory Furnace"; International Symposium on Applications of Statistics, Operations Research, and Computers in the Mineral Industry, COLORADO SCHOOL OF MINES, Golden, Colo.

9). 1964 - "Dynamic Problems in Mining"; International Symposium in Computers and Computer Applications, UNIVERSITY OF ARIZONA, Tucson, Arizona.

10). 1964 - "Application of Simulation in Evaluating Low Grade Mineral Deposits"; Bureau of Mines, Report of Investigations Number 6501, 63 pages.

11). 1964 - "Polynomial Surface Fitting Using Surface Data from an Underground Copper Deposit"; U.S. Bureau of Mines, Report of Investigations Number 6522, 27 pages.

12). 1964 - "Simulating Mineral Deposits Utilizing Monte Carlo Techniques and Mathematical Methods"; U.S. Bureau of Mines, Report of Investigation Number 6493, 27 pps.

13). 1963 - "Computer Methods in Evaluation, Development, and Operations of an Ore Deposit"; Annual Meeting of the American Institute of Mining, Metallurgical, and Petroleum Engineers, Dallas, Texas.

14). 1963 - "Pit Design Utilizing a Digital Computer"; Symposium of Operations Research, STANFORD UNIVERSITY, CA.

15). 1963 - "A Basic Computer Program for Computing Grade and Tonnag of Ore Using Statistical and Polygonal Methods"; U.S. Bureau Of Mines, Report of Investigation Number 62

16). 1963 - "Computing Ore Reserves by the Triangular Method Using a Medium Sized Digital Computer"; U.S. Bureau of Mines Report of Investigation Number 6176, 30 pages.

17). 1962 - "A Survey of the Techniques and Applications of Computer in Resolving Operating Problems"; American Mining Congr San Francisco Annual Meeting.

18). 1962 - "Computing Ore Reserves by the Polygonal Method Using a Medium Sized Digital Computer"; U.S. Bureau of Mines Report of Investigation Number 5952, 31 pages.

19). 1962 - "Mineral Deposit Evaluation Using Mathematical Models and a Digital Computer"; University of Arizona Symposium on Computer Applications, vol. 1, section K, pages 1-55.

20). 1962 - "Use of High-Speed Data Reduction and Processing in the Mineral Industry"; U.S. Bureau of Mines, Information Circular Number 8099, 82 pages.

21). 1962 - "Formulating Computer Problems"; University of Arizona Symposium on Computer Applications, vol. 1, D[37 pgs.].

22). 1961 - "Calculating Ore Reserves Using a Digital Computer"; MINING ENGINEERING, v. 13, pages 37-42.

23). 1961 - "Small Mines Can Make Wide Use of Computers"; MINING WORLD, vol. 23, no. 7, pp. 38-40.

24). 1961 - "Computer Calculates Best Method to Develop High-Angle Escalante Vein"; MINING WORLD, vol. 23, no. 10, pp. 31-35.

25). 1960 - "A Comparison of Various Methods of Calculating Ore Reserves Using a Digital Computer, University of Arizona Computer Symposium.

The Hewlett Reaction System (HRS) - An Outline

1. Introduction

Many precious and base metals present very complex recovery problems that result in extremely high capital and operating costs. This may restrict new operations to the development of only large reserves and relatively high grade properties. In this way the latter costs can be justified when they may run into the hundreds of million dollars to just oxidize a refractory ore. These expensive methods which include roasting, calcining and pressure oxidation in autoclaves are usually followed by equally expensive leaching, electrowinning or vacuum liquation to extract the values.

The previously described costs, which reduce profits for existing operations of all sizes, severely restricts the development of new mines. It certainly reduces the viability of many mining ventures when the only recourse is to deliver ore to a custom processor. Other potential operations with small reserves and/or lower grades are never developed.

The Hewlett Reaction System (HRS) solves many of these problems. The system permits the recovery of precious and base metals and minerals from complex ores and includes the cleansing of the environment of deleterious elements. The sub-system to the HRS produces pure metals eliminating the delivery of doré, for instance, to a refinery.

2. Application of the HRS

The HRS can be applied to most forms of mining including alluvial (placer) and hardrock of all kinds. Unlike the cyanidation process, which is extremely toxic, the primary solvent used in relatively low concentration is a bromine/chlorine complex. The toxicology of both bromine and chlorine are well known and are universally used to purify domestic water supplies. Significantly, the HRS permits far more rapid and high recovery of metals and minerals than conventional processes.

3. Scope of the HRS

I. Oxidation &/or chlorination of waste water, mine effluent and sewage.
ii. Reduction of metal ions in toxic waste water.
iii. Solid toxic water clean-up; detoxification.

Note: Refractory gold and other precious metals that cannot be treated by conventional cyanidation but can be treated with the HRS are as follows:

iv. Precious metals in a sulphide matrix.
v. Gold ores attached to carbon by a weak organic bond.
vi. Encapsulated gold in silica, silicates, carbonates, etc.
vii. Films on gold: iron oxide/hydroxide, silver chloride from electrum, sulphide coatings and iron silicates (desert varnish).
viii. Manganese (valance +4) and psilomelane and any other compound that requires reduction before leaching.
ix. Interstitial gold in magnetite, haematite and psuedomorphs.
x. Oxygen consuming minerals such as arsenic and antimony sulphides, etc.
xi. Speiss and matte forming elements.
xii. Certain physical properties such as texture.
xiii. Excess iron.

4. The HRS Components

The component parts of the HRS are as follows:

I. Reaction Chamber, vertical or inclined, etc. to suit mill facility.
ii. Circulating/Holding Tank.
iii. Pulp Circulating Pump.
iv. Ion Exchange System containing resin columns and elution facility.
v. Furnace and pure metal ingot pouring facility.

5. The HRS Process

The HRS is a natural oxidizing system where slurried concentrates are pumped from a circulating tank to the head of a downward sloping chamber. Preoxidation is achieved with the addition of calcium hypochlorite in the flow of the pulp and which is made turbulent causing rapid oxidation. Low pressure air and ozone are introduced to the circulating pulp as an accelerant. At a predetermined pH a small quantity of HCL may also be added to accelerate the process which causes a short burst of hydrogen sulphide gas. The latter emission is scrubbed with normal safety practices being observed. The oxidizing process is monitored and when the potential is reached the leaching phase commences with the addition of the solvent and reagent. The leach phase is monitored by atomic absorption and when the metallic maximum potential is reached the solution is decanted, filtered and the pregnant solution reports to ion-exchange. The ion-exchange resins are then stripped in the elution process, the resultant precipitates are fired and pure metals poured into ingots. Meanwhile, the barren solution is returned to storage, the potency adjusted and the solution reused in the processing of further concentrates. The total process time is usually well within the limits of a single eight hour shift.

Significantly, the HRS can be adapted and sized to treat the smallest alluvial concentrate or scaled up to process a high tonnage of complex ore fed to a hardrock mill. The system is environmentally and user friendly requiring only nominal metallurgical qualification.

The HRS is specifically adapted to process concentrates created by a Knelson gravity circuit.

An HRS is designed, fabricated, installed and commissioned on the basis of cost plus fee.

Metallurgical Research and Assay Laboratory

745 Sunset Road Suites 8
Henderson, NV 89015
702-565-0074

ASSAY REPORT

Assay Number: 4852 Date: 9/3/98

Customer : RONALD OWENS

Sample Identification: HEMATITE ORE

Element	Oz/Ton
Au-Gold	0.72
Ag-Silver	0.74
Pt-Platinum	1.09
Rh-Rhodium	0.11
Os-Osmium	2.62
Ru-Ruthenium	0.19
Pd-Palladium	0.32
Ir-Iridium	0.69

REGISTERED ASSAYER CERTIFICATE NO
19127
DONALD E
JORDAN
Date Signed 9/3/98
ARIZONA, U.S.A.

These results are based on well known accepted analytical procedures used solely on the sample submitted by the customer. This report is prepared for the exclusive use of the customer. No warranty as to the reproducibility or extractability of the material other than the sample is given. Donald E. Jordan and/or Metallurgical Research and Assay Laboratory make no representation express or implied on material other than that represented by the sample assayed.

Note, " #VALUE! " MEANS THAT ELEMENT HAS NOT BEEN ANALYZED FOR THIS REPORT.
Unless prior arrangements are made, all samples will be discarded after 30 days.

Metallurgical Research and Assay Laboratory

745 Sunset Road Suites 8
Henderson, NV 89015
702-565-0074

ASSAY REPORT

Assay Number: 4872 **Date:** 9/18/98

Customer : RONALD OWENS

Sample Identification: HEMITITE - CYANIDE TEST C - 227 gms. In 1000 mls. / agitated, temp ambiet, 12 hours
Material was pre leached using a weak HCl solution to enhance cyanide leaching. Soltion was assayed and no values were found. The pre leach did not seem to help recovery.

	Element	PPM	Oz/Ton
pre leach solution	Au-Gold	0.0	0.00
	Ag-Silver	0.0	0.00
	Pt-Platinum	0.0	0.00
6 hours	Au-Gold	0.9	0.11
	Ag-Silver	1.5	0.19
	Pt-Platinum	0.0	0.00
12 hours	Au-Gold	1.2	0.15
	Ag-Silver	4.0	0.51
	Pt-Platinum	0.1	0.01

REGISTERED ASSAYER CERTIFICATE NO. 19127 DONALD E JORDAN Date Signed 9/18/98 ARIZONA, U.S.A.

These results are based on well known accepted analytical procedures used solely on the sample submitted by the customer. This report is prepared for the exclusive use of the customer. No warranty as to the reproducibility or extractability of the material other than the sample is given. Donald E. Jordan and/or Metallurgical Research and Assay Laboratory make no representation express or implied on material other than that represented by the sample assayed.

Note: " #VALUE! " MEANS THAT ELEMENT HAS NOT BEEN ANALYZED FOR THIS REPORT.
Unless prior arrangements are made, all samples will be discarded after 30 days.

Metallurgical Research and Assay Laboratory

745 Sunset Road Suites 8
Henderson, NV 89015
702-565-0074

ASSAY REPORT

Assay Number: 4871

Date: 9/18/98

Customer : RONALD OWENS

Sample Identification: HEMITITE - CYANIDE TEST B - 227 gms. In 1000 mls. / agitated, temp ambient, 12 hrs.

	Element	Oz/Ton
6 hours	Au-Gold	0.12
	Ag-Silver	0.17
	Pt-Platinum	0.00
12 hours	Au-Gold	0.14
	Ag-Silver	0.24
	Pt-Platinum	0.01

REGISTERED ASSAYER
CERTIFICATE NO
19127
DONALD E
JORDAN
Date Signed 9/18/98
ARIZONA, U.S.A.

These results are based on well known accepted analytical procedures used solely on the sample submitted by the customer. This report is prepared for the exclusive use of the customer. No warranty as to the reproducibility or extractability of the material other than the sample is given. Donald E. Jordan and/or Metallurgical Research and Assay Laboratory make no representation express or implied on material other than that represented by the sample assayed

Note: " #VALUE! " MEANS THAT ELEMENT HAS NOT BEEN ANALYZED FOR THIS REPORT.
Unless prior arrangements are made, all samples will be discarded after 30 days.

DONALD E. JORDAN
804 PURPLE SAGE TERRACE
HENDERSON, NV 89015

EDUCATION:

REGISTERED ASSAYER ARIZONA #19127

UNIVERSITY OF COLORADO, BOULDER, CO.
Ph.D., Analytical Chemistry - 1961. Thesis directed by Dr. Harold F. Walton.

IDAHO STATE UNIVERSITY, Pocatello, ID.
B.S. Chemistry - 1948

EXPERIENCE

11/90-Present

METALLURGICAL RESEARCH AND ASSAY LABORATORY, Henderson, NV Owner and Chief Analytical Chemist. Research in Metallurgy, Analysis, and recovery of precious metals found in complex ore, concentrates, etc, utilizing digestion, fusion, furnacing techniques; and, analysis by plasms spectroscopy. Recovery studies by cyanide, bromine, ion exchange, high temperature fire assay and others. Established a sampling program for ore bodys and concentrates, suitable for investor acceptance. Continued analytical service providing high quality analysis to clients. Continued specific procedures using high resolution DCP/ICP and our study of inter-element interferences and elimination of them.

4/87-11/90

ALPHA RESEARCH CORPORATION, Henderson, NV Chief Analytical Chemist and co-founder. Research in analysis and recovery of precious metals found in ores, concentrates, dore, and scrap by utilizing various digestion, fusion, furnacing, wet chemical techniques; and, analysis by plasma spectroscopy. Analytical service providing high quality, complete analysis (up to 71 metals) to clients. Developed specific procedures for the use of high resolution ICP, and a compendium of some inter-element interferences and solutions to eliminate them.

9/86-12/86

UNITED STATES RESOURCE DEVELOPMENT CORP., Henderson, NV Corporate responsibility to set up an analytical laboratory for Fire assay, Chemical methods, and ICP spectroscopy to determine gold, silver, platinum group, along with strategic, and rare earth metals.

6/85-9/86 J.B. LABORATORY, Phoenix, AZ Set up and establish D.C. Plasma Spectroscopy to analyze ores, concentrates, liquid leach products, black sands, cinder cones and Dore bars for gold, silver and platinum group metals.

1/83-5/85 ANALYSTS, INC., Oakland, CA CORPORATE RESEARCH CHEMIST, Research and development. Supervised four chemists in non-routine oil, inorganic and metallurgical analysis, utilizing infrared, A.A. Ferrography, and plasma spectroscopy. Was also responsible for developing new methods for metal analysis.

LABORATORY MANAGER Supervised a 28 person laboratory in analyzing oil samples, performing computer data entry, printing results and forwarding reports to corporate headquarters, maintained supplies and lab inventories. Hired, trained, evaluated, and, where necessary, discharged lab personnel. Was able to change company data base from routine to a higher value status.

1974-1982 PHELPS-DODGE CORPORATION, Morenci, AZ SENIOR RESEARCH CHEMIST - Analytical Research Chemical Techniques to determine trace and higher concentrations of most metals. Developed the use of DCP Spectroscopy for the following:
Wear metals through oil analysis,
Trace impurities in Cathode Copper,
Analysis of refinery slimes for Au, Ag, Pt, Pd, Cu, Se, Te, Sb, As, Pb, Ni, (simultaneously),
Analysis of impurities in Nickel Sulfate prepared from slime and many other trace elements such as Rhenium, Gallium, Germanium, Tantalum and others.

Studied various leaching techniques for the tailings leach program.

1973-1974 UNIVERSITY ANALYTICAL CENTER, UNIVERSITY OF ARIZONA, Tucson, AZ SENIOR CHEMIST - University Analytical Center (UAC). Organized and directed service activities, and participated in sampling and analysis of air pollutants.

1960-1973 CONTINENTAL OIL COMPANY, Ponca City, OK ANALYTICAL RESEARCH SECTION. SENIOR RESEARCH SCIENTIST.
Organic and Physical Properties.
Organic Laboratory.
Inorganic Laboratory.

Methods Research, development and modernization, complete fertilizer technology, including Super Phosphoric Acid, using Automated Technicon and Chromatography techniques.

Separation and determination of Pyrophoric Aluminum organic compounds. General capability in UV, IR, Flame Emission, A.A., Chromatography, Thermal (DTA, TGA), and time of Flight Mass Spectroscopy.

Participated in preparation and analysis of Metallo Organic Sulfonate Standards for used oil analysis. Helped establish a complete Copper Ore methodology (see Phelps Dodge above). Also, established chemical and instrumental methods for water quality evaluation and structural parameters on rigid silicate foams.

1959-1960 NATIONAL BUREAU OF STANDARDS, Boulder, CO CRYOGENIC ENGINEERING LABORATORY. Search of World Wide Literature for Thermal Physical properties of Cryogenic Solids and Liquids. Coded the data for storage and retrieval.

1952-1957 MONSANTO, St. Louis, MO Characterization of the structure of Anhydrous Mono Calcium Phosphate. Developed a method to determine trace quantities (0.2%) Magnesium in the presence of Phosphate and Calcium.

Plant chemist in Method Development, modernization of methods at Trenton, Michigan, and Carondelet in St. Louis, Missouri.

1948-1951 J.R. SIMPLOT, CO., Pocatello, ID ACTING CHIEF CHEMIST - Control Laboratory including complete Ore Assay Facility.

PUBLICATIONS Please see attached list.

Donald E. Jordan

PUBLICATIONS

1. D.E. Jordan and C. Callis, "Determination of Magnesium Content of Calcium Phosphates and Commercial Lime", Anal. Chem., 30, 1991 (1958).

2. C.D. Gutsche, T.D. Smith, M. Van Ufford, D.E. Jordan, "A New Synthesis of Bridged Ring Ketones and Medium Sized Ring Compounds", J. of Amer. Chem. Soc., 80, 4117 (1958).

3. H.F. Walton, D. E Jordan, S. Samaday and N. McKay, "Cation Exchange Equilibria with Divalent Ions", J. Phys. Chem., 65, 1477 (1961).

4. D. Beichler, D.E. Jordan, and W.D. Leslie, "Spectrophotometric Determination of Vanadium in High Purity Aluminum", Anal. Chem., 35, 1685 (1963).

5. D.E. Jordan and F. Veatch, "Spectrophotometric Determination of Trace Concentrations of Carbonyl Compounds", Anal. Chem., 36, 120 (1964).

6. D.E. Jordan, "Selective Hydroxyl Group Determination by Direct Titration with Lithium Aluminum Amide", Anal. Chimica Acta, 30, 267 (1964).

7. D.E. Jordan, "Hydroxyl Group Determination in High Molecular Weight Alcohols and Complex Organic Mixtures", J. of Am Oil Chem. Soc., 41, 500 (1964).

8. D.E. Jordan, "Quantitative Saponification of Highly Hindered Esters by Perchlorate Anion Catalysis", Anal. Chem., 36, 11, 2134 (1964).

9. D.E. Jordan and D.E. Monn, "Rapid Determination of Magnesium in the Presence of Calcium and Phosphate by Titration with CDTA", Anal. Chimica Acta, 37, 379 (1967).

10. D.E. Jordan, "Simultaneous Determination of Trace Concentrations of Saturated and Alpha, Beta-Unsaturated Carbonyl Compounds in Organic Systems", Anal. Chimica Acta, 37, 379 (1967).

11. D.E. Jordan, "Spectrophotometric Determination of Saturated and Alpha, Beta-Unsaturated Carbonyl Compounds in Complex Systems at a Single Wavelength", J. of Am. Oil Chem. Soc., 44, 400 (1967).

12. D.E. Jordan and A.B. Carel, "Here's a Quick Field Test for Halogenated Hydrocarbons in Crude Oil", Oil and Gas Journal, August 28, 1967.

13. D.E. Jordan, "High Precision Determination of Up to 75% P205 By Automated Technicon Technology. Technicon Symposium Oct. 1967.

14. D.E. Jordan and D.E. Monn, "Spectrophotometric Endpoint Detection for Determination of Magnesium in the presence of Calcium and Phosphate by Titration with CDTA", Anal. Chimica Acta, 39, 401 (1967).

15. D.G. Beichler and D.E. Jordan, "Extraction and determination of Vanadium", U.S. Patent #3,345,126, October 3, 1967.

16. D.F. Hagen, D.G. Beichler, W.D. Leslie and D.E. Jordan, "Controlled Deactivation Hydrolysis and Determination of Aluminum in AlR3", Anal. Chimica Acta, 41, 557 (1968).

17. D.E. Jordan, "Determination Linear 1 - Chloroalkanes in Mixtures of Linear and Non-Linear (C12-C16) chloroalkanes by Reaction Kinetics", Anal. Chem., 40, 1717 (1968).

18. D.E. Jordan, "Visual Titrimetric Determination of Total Reactivity and Differentiation of AlR3 and ADAH in Mixtures", Anal. Chem., 40, 2150 (1968).

19. A.B. Carel and D.E. Jordan, "Determination of Total Phosphate and Identification of Acid Insolubles in Superphosphoric Acid", J.A.O.A.C. 52, May, 1969.

20. D.E. Jordan and J.L. Hoyt, "Application of an Automated Coulometric Titrator to the Determination of Water in Phosphoric Acids", J.A.O.A.C. 52 May 1969.

21. D.E. Jordan, "Automated High Precision Determination of Phosphorus at Concentrations of 0.01 to 7.5 mg P205/ml", J. A.O.A.C. 52, Technicon Corporation, May 1969.

22. J.L. Hoyt and D.E. Jordan, "The Simultaneous Determination of Phosphorus, Iron and Aluminum in Phosphate Rock and Fertilizer Systems by Automated Colorimetric Analysis", J.A.O.A.C., 52, No. 6 (1969).

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I.P.M.I.
Proceedings
(Scottsdale, Az)
Nov. 1989

International
Precious
metals
Institute ?

Note:
1989 Conf
was in
montreal.

1992 (16th CONF)
was in
Scottsdale

THE ANALYSIS OF DESERT FLOOR, BLACK SANDS, VOLCANIC CINDERS, SERPENTINES AND OTHER MATRICES FOR GOLD, SILVER AND PLATINUM GROUP METALS BY PLASMA SPECTROSCOPY

BY

DONALD E. JORDAN

ABSTRACT

A method for the determination of gold, silver, and the platinates by Atomic emission plasma spectroscopy is presented. Atomic emission plasma excitation combined with a high resolution echelle grating (approx. .02A) shows no interference when analyzing the precious metals at the chosen wavelengths. Analysis of the eluant is completed and tabulated in about five minutes after sample preparation. Time required for sample preparation varies widely depending solely on the complexity of the ore matrix. A survey of some platinate literature from 1905 to 1984 is included.

Contribution from:
Alpha Research Corporation, 8390 S. Fourth St., Henderson, NV 89015

INTRODUCTION

ly, precious metals have been determined by chemical fire say procedures(1). Accurate results are obtained for Gold and Silver in simple matrices. However, in complex materials such as volcanic cinders, black sands and similar type ores, fire assay is most difficult without some kind of pretreatment. At best, only limited Gold and Silver is obtained by fire assay procedures, with essentially no Platinum group metals. Davis(2), Beamish and Van Loon(3), and others have presented methods to determine the Platinum group metals in pure mixtures by fire assay, while Ammen(4) and Liddell(5) each describe procedures for the separation of the Platinum group by chemical methods.

Substantial Platinum deposits equal to Gold, or nearly so, occur in the black sands of eight Western states and South Dakota as described by Emmons and Eckel(6). Their study included over 2,000 samples from a total of 34 states, territories, British Columbia, Mexico and South America. Platinum, Iridium and Osmium have been produced in Shasta County as a byproduct of gold dredging on Clear Creek, Cottonwood Creek and Roaring River by placer miners on Beegum Creek(11,12,13,15). Other studies have shown Mariposa County, California, Devil's Gulch, to assay $94/T Platinum and $44/T Gold at 1920 prices(7). The same publication shows Flaky Gold and a ratio of 4.08 Osmiridium to one part Platinum in Mendocino County, California. Two sample lots of 30 tons each averaged $1.81 and $1.88 per cubic yard. Additionally, a shipment from upper Beegum Creek, Shasta County, contained 1.6 oz., assaying 1.25 oz. Osmiridium, .05 oz. Gold and 0.3 oz. Platinum. The values were: Iridium $400-$450/oz., Osmium $75/oz., and Platinum $105/oz. Merti et. al., also discusses the Platinum group metal deposits at length, including the United States(14).

The most recent information by government agencies alleges there are no primary Platinum group metal deposits in the United States except for the Stillwater Complex in Montana. This seems to be a direct anomaly to the work outlined above. It is felt the problem occurs because of the inability of many laboratories to detect the Platinum group; or, if they do detect the Platinates, their reluctance to report the values. Many laboratories approved by the government use only fire assay or fire assay followed by dissolution of the prill and an atomic absorption finish. Some try to determine the precious metals concentration in ore by direct atomic absorption of eluate using an organic (usually MIBK) extraction.

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fire assay followed by
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organic (usually MIBK)

With the development of stable high energy plasma excitation sources for atomic emission spectroscopy, an alternative to fire assay is available. Here, the plasma spectrometer must be coupled with a high resolution grating (approximately .02 angstrom)(8,9,10) to successfully perform multi-element analysis of precious metals at low PPM levels and other metals in all matrices without interference. The use of either DCP or ICP excitation coupled with a high resolution echelle grating provides a substitute for fire assay. Results for all precious metals, including preparation time, can be completed in less than six hours for simple and up to 24 hours for the most complex ores studied to date. The continued development of stable plasma sources should result in additional "state of the art" plasma spectrometers in the near future.

EXPERIMENTAL

APPARATUS

A Spectraspan III-A manufactured by Spectrometrics, Inc. Andover, Mass, with sequential and multi-element capability, and a Leeman Labs Plasma Spec ICP 2.5 manufactured by Leeman Labs, Inc., Lowell, Mass., were used in this study. The precious metals determined and their wavelengths are outlined below:

ELEMENT	**WAVELENGTH, NM**	
	ICP	**DCP**
Gold	267.6	267.6
Silver	328.07	328.0
Platinum	214.42	265.9
Rhodium	369.24	A
Osmium	228.23	A
Ruthenium	267.86	A
Palladium	340.46	340.46
Iridium	215.27	A

A - Not Analyzed by DCP.

Fire assay and classical chemical procedures were done in our chemistry and fire assay section.

REAGENTS

Only C.P. reagents and Class A glassware were used. All reagents were analyzed prior to and during use.

STANDARDS

All standards were purchased as single element plasma standards. The matrix high standard was prepared by analytical dilution of the purchased 1,000 and 10,000 ppm standards. The multi-element standard contained HCl, HNO_3, 10/ug/ml each precious metal, 20ug/ml of several base metals; 500 ug/ml iron and 80 ug/ml each of Manganese, Thallium, and Mercury; all made to 500.0 ml.

SAMPLE PREPARATION

A. DESERT FLOOR, LAKE BEDS AND OTHER SIMPLE MATRIX ORES.

Accurately weigh 2.5 gm of -200 mesh ore into a 250 ml Class A Volumetric flask. Add 20 ml Conc. HNO_3 and swirl to mix. Let stand 20 minutes at room temperature to oxidize any volatile material to a higher oxidation state. Then add 60 ml Conc. HCl and digest at a low boil or simmer for four (4) to five (5) hours. Remove the sample from the heat source, cool, and dilute with DI water to volume and mix well.

B. BLACK SANDS, VOLCANIC CINDERS, AND OTHER COMPLEX ORES.

Accurately weigh from 0.5 to 2.5 grams of -325 mesh sample into a 250 ml Class A Volumetric flask. Digest as in (A.) above, at a low boil or simmer from twenty (20) to twenty-four (24) hours. Remove, cool and dilute to volume with distilled water then mix well.

C. OTHER METALS AND MIXTURES.

1. **Resins.**
   Perform a wet ashing digestion on dried resin using H_2SO_4 and Nitric Acid. Upon completion, dilute to 50 ml, add 60 ml Aqua Regia and continue digesting for 30-60 minutes. Cool, dilute to volume and mix well.

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THER COMPLEX ORES.

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water then mix well.

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ue digesting for 30-60
e and mix well.

2. **OSMIRIDIUM, OSMIRHODIUM, PLATINO RHODATES AND PLATINO IRIDATES, ETC.**
These samples require an oxidative alkaline fusion at 800 degrees C. in a Zirconium or Nickel crucible for at least one-half hour.

CALIBRATION AND ANALYSIS

A. DCP (SPECTRASPAN III-A, also Beckman and ARL)

The DCP is set up as described by Reednick(10). Align the grating to its geometric center (253.2 nm) using the preselected channel in the cassette and a **PURE** single element standard for that channel. Complete the alignment with the same element in the multi-element standard mixture.

The instrument was calibrated using high and low standard mixtures chosen so the sample would fall between the standard values. The channel representing each element in the multi-element cassette is actuated by turning on the channel switches and pressing the channel button. Values for the high and low standards were entered into the computer for each element in their respective channels. The photomultiplier tube (PMT) voltages were set at 5. The high standard was pumped into the plasma and the computer activated. After the "Autorange" button was depressed, the elements were ranged through two (2) eight-second integrations and the results printed. This step was repeated until each channel showed a photomultiplier (PMT) reading near 25 (any value between 12-50 can be used) and a count per integration near 4300. To achieve this, PMT voltage to the affected channels is increased or decreased. When the adjustments were completed, the low standard was pumped into the plasma. Enough time must be allowed to eliminate all residual high standard memory (7-12 seconds). After the "Low Std" button was depressed, the low standard was ranged through two (2) eight-second integrations and the results printed. The low and high standard mixtures are analyzed as samples by pressing the "sample" button. Calibration was complete. Samples were analyzed the same as the standards (i.e., two (2) eight-second integrations). High and low standards were analyzed as every fourth and fifth sample.

Standard additions were used (both single and multipoint) for each sample and the values were corrected for matrix effects.

The DCP instrument parameters were set as follows:

Sleeve Pressure	50 psi Argon
Nebulizer Pressure	30 psi Argon
Anode Spacing	5/32"
Sample Volume	.80ml/min

B. ICP (LEEMAN LABS PLASMA SPEC 2.5, LOWELL, MASS)

Set up the ICP as described in the manufacturer's operating manual. The ICP instrument parameters are set as follows:

Auxiliary	Pressure 5 psi	Flow Rate 0 L/min
Nebulizer	Pressure 40 psi	Flow Rate .4 L/min
Coolant	Pressure 5 psi	Flow Rate 12 L/min
Pump	Slow	
Power	0.54A	

The Source is optimized using sequential mode as outlined. Press Sequential A for the A program. Input Nickel 3 (231.60nm) to the program. Next press Peak 1 to align the optics. Then press Peak 2 to optimize the source. After the alignment is finished, the CRT will say "ready". Press "Scan". In a few seconds the CRT will say "Press Profile". The resulting display will show the Nickel peak symmetrical with the APEX directly over the "P" on the X-axis. If it isn't, repeat. Then do Peak 3 and Peak 4, and make adjustments necessary to obtain the curves outlined in the manual.

Next, align the wavelength of the metals of interest. Press sequential multi and "A" on the keyboard. Prepare the single elements of interest at about 100 ppm each. On sequential multi "A", input all the wavelengths for each element; i.e., Gold, Au1, Au2, Au3, ... These will be entered into program A. Now, input the 100 ppm gold standard into the plasma for 30-45 seconds and press Peak 2. The optics will automatically align, then the wavelength will optimize. When the CRT shows ready, press scan and then profile when scanning is done. All the wavelengths for the element should show the same symmetry as for Nickel; i.e., the peak must be symmetrical and the Apex directly above the P on the X-axis. If they are skewed, repeat Peak 2. Print the aligned profiles for reference. Repeat the sequence for each element of interest. After all the elements are aligned, prepare a program of all the elements of interest as outlined in the Manual using only the wavelengths best suited for the analysis. Complete pages 2-6 of the program as shown in the Manual. Page 6 sets the important

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analytical parameters for the operation of the program. Scan the wavelengths, then press profile which displays page 1.

To display and print the displays, press "ENTER" and "PRINT". Repeat until all the scans are printed. If any element is unaligned, do a Peak Wavelength (Peak 2) on each single element that is unaligned. Then repeat the scan, profile and print sequence above on the mixed standard. Repeat until all the elements in the high standard are aligned. DO NOT ALIGN ANY WAVELENGTH WITH THE MIXED STANDARD.

B.1 STANDARDIZATION

Press "Std" [1] and "Enter" [1] [Enter] when the data is complete, it will print. Then press "Std" [1] Enter [2] Enter and finally "Std" [1] Enter [3] Enter. This means that for standard #1, we collected three (3) sets of raw data. The sequence is repeated for standard #2. The raw data is used to calibrate the instrument. Press calibrate to present the calibration program. Now press Enter, the calibration curve for the first element will be displayed. If reasonably straight, press Y to accept it, press print to print it, then press Calibrate and Enter, etc., until all the elements are calibrated. Be sure to accept each curve or the calibration for that metal will be excluded. After the calibration Press the "Update" key. The CRT will ask which standard? Pump the high standard mixture into the plasma for approx. 45 seconds and press 2 for the High standard, then Enter. When the update is complete, the value for each metal will be printed. Then press sample and enter "High Std". Press enter and the values will be determined. When this sequence is finished, repeat for the low standard. You are now ready to analyze samples. Repeat the high and low standards as samples every three (3) to four (4) samples, (for complex or high salt matrices, every 2 samples). Report results as troy oz/Ton as follows:

$$\text{Oz/T} = (\text{PPM from ICP})(\text{Dilution/wt. sample})(.029167)$$

Standard addition was used and the values corrected for matrix effects.

DISCUSSION AND RESULTS

All samples analyzed were received from outside sources. If necessary, they were crushed, split and pulverized to a minimum of -200 mesh (74u).

Black sands and volcanic cinders were ground to -325 mesh (45u). The author feels and to some extent has shown that a finer grind (-400 to -600 mesh) might release the precious metals more readily and more completely than a 45u grind. Leaches and liquids received were run as is unless they had to be acidified. Any samples exceeding the high standard were analytically diluted with the low standard to bring them into the high standard range. Any samples exhibiting high silver thus insoluble silver chloride that exceeded the limits of our volume to redissolve AgCl to $Ag(Cl)_4^{-1}$ ($AgCl + 3Cl^- = Ag(Cl)_4^{-1}$) were repeated with a smaller sample.

Both the DCP and ICP were set up according to the manufacturer's specifications. Each one was adjusted to optimize the excitation. Both were warmed up (about one half hour) to minimize drift. For DCP, standards and samples were analyzed through two (2) eight-second integrations. For the ICP, the standards and samples were analyzed through three (3) three-second integrations. In each case, the results were printed and reduced to tabular format for easier understanding.

Table I shows results for selected samples. The first four samples are repeated here from an earlier paper presented at the "Developments in Atomic Plasma Spectrochemical Analysis" Conference at San Juan, Puerto Rico, and published in those proceedings(8). All other results are from ICP Analysis. Samples #2516, #2532, and #2540 all required alkaline fusion. Sample #2532 required three separate fusion to solubilize all the Iridium. The other samples were either received as liquids or were readily soluble by the acid treatment suggested. It is interesting to note the samples come from as far away as New York and many are received locally or 300 or 400 miles away. Some are received as pad leach concentrates from Show Low, Arizona.

Table II shows the results on our sample #3946 by five (5) other well known laboratories. For one of these "A", their results are accepted internationally. "B" and "C" are widely accepted throughout the United States and Canada. "E" is a well known government laboratory and "D" is a laboratory in a very large company interested in precious metals.

It has been suggested by some workers that iron interferes in the plasma analysis for precious metals. This simply is not true using high resolution plasma spectrometers. A study conducted in our laboratory recently with iron from 1,000ppm to 10,000ppm (0.25 to 2.5 gm/250ml) showed no interference at any wavelength of interest.

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When these iron concentrations were analyzed as samples the values for the precious metals increased linearly as the iron increased. Further scanning of the solutions showed minor amounts of Rhodium and others in the scans corresponding to the analysis. It should be pointed out that all iron samples analyzed contain small to medium concentrations of precious metals. For example, iron nails, steel railroad rails, iron salts, etc.

Figure 1 shows the correlation between fire assay and plasma spectroscopy(8) in refinery slimes. Not all samples fire assay without a pretreatment. For example, black sands and volcanic cinders. The graphical results are shown in percent. If the results were equivalent the data points would fall on the theoretical 45 degree line.

SUMMARY

The data here and the USGS Bulletin #285(6) and others(5,7) shows the presence of Platinum Group Metals and Gold and Silver in nearly all types of matrices studied. Black sands, volcanic cinders, serpentines, as well as others, are too complex for simple fire assay and require a pretreat before significant values are obtained. Plasma spectroscopy using a high resolution grating is more than adequate for accurate analysis of precious metals. Iron, nor any other trash metal, show interference in the analysis at the wavelengths of choice; although, a poor choice of wavelengths certainly could cause an interference.

ACKNOWLEDGMENT

To Jim Scott and David Graham for their suggested changes and critique of the material; to Benton Wilcoxon for editing; and to Kateri Jordan for the typing.

BAHAMIAN REFINING SERVICES & MINING EQUIPMENT COMPANY
A Division of Bahamian Refining Corporation
9222 N. 14th AVE., PHOENIX, ARIZONA 85021 • TELEPHONE (602)XXXXXXXXX
NEW NUMBER 602-944-6577 FAX 944-1893

DATE: OCTOBER 7, 1997

NAME: RONALD OWENS, YUKALTA RESOURCES INC.
ADDRESS: 201 - 5201 - 52 AVENUE
PONOKA, ALB T4J-1H6

SAMPLE: #1

Dear Mr. Owens:

This is a report on the sample you submitted, showing the actual recovery of all 16 formulas using BIO-D-Leachent, reported in ozs. per ton.

BASIC	Au	Ag
Formula # 1	.38	.02
Formula # 2	1.20	.14
Formula # 3	1.07	.13
Formula # 4	.02	.00
Formula # 5	1.38	.14
Formula # 6	.91	.03
Formula # 7	.10	.00
Formula # 8	.31	.01

ACID		
Formula # 1	.30	.00
Formula # 2	1.20	.12
Formula # 3	1.27	.12
Formula # 4	.14	.00
Formula # 5	1.34	.14
Formula # 6	.78	.02
Formula # 7	.10	.00
Formula # 8	.16	.01

[redacted]

Fred Finell, Jr President

BRC DID NOT SELECT THE SAMPLE(S) TESTED, AND NO REPRESENTATIVE OF BRC HAS VISITED THE PROPERTY AND/OR PLANT AND/OR LABORATORY FROM WHERE THE SAMPLE(S) WERE TAKEN. BRC HAS NO WAY OF KNOWING IF THE SAMPLE(S) TESTED ARE REPRESENTATIVE OF THE PROPERTY, AND/OR THE METHODS OF TESTING AND/OR PROCESSING USED, AND BRC DOES NOT KNOW THE EXTENT THE SAMPLE(S) REPRESENT OF ANY VOLUME OF ORE(S) ON THE PROPERTY OR THE COMMERCIAL FEASIBILITY OF THE PROPERTY. BRC VERIFIES AND ATTESTS THE VALUES REPORTED WERE ACTUALLY RECOVERED FROM THE SAMPLE(S) PROVIDED TO AND TESTED BY BRC. NO WARRANTIES ON THE REPRODUCIBILITY OF THE SAMPLE ARE GIVEN, EXCEPT FOR IDENTICAL SAMPLE AND TEST REPEATED IN HOUSE. BRC MAKES NO WARRANTY, EXPRESS OR IMPLIED, AND ASSUMES NO LEGAL LIABILITY WHATSOEVER AS TO THE USEFULNESS OF ANY INFORMATION CONTAINED IN THIS REPORT.

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BRS
BAHAMIAN REFINING SERVICES & MINING EQUIPMENT COMPANY
A Division of Bahamian Refining Corporation
9222 N. 14th AVE., PHOENIX, ARIZONA 85021 • TELEPHONE (602) ~~279-9702~~ XXX
NEW NUMBER 602-944-6577 FAX 944-1893

DATE: JANUARY 28, 1998

NAME: RONALD OWENS
COMPANY: YUKALTA RESOURCES INC.
ADDRESS: 201 - 5201 - 52 AVENUE
PONOKA, ALB T4J-1H6

SAMPLE: # 1

Dear Mr. Owens:

Following are the results, all in ozs. per ton, of the tests for platinum group metals on the sample you submitted using BIO-D-Leachent.

	Basic Formula # 5	Acid Formula # 5
Platinum	.10	2.15
Palladium	1.89	2.81
Rhodium	.18	.15
Iridium	.63	.84
Osmium	.43	.51
Ruthenium	2.08	1.47

[redacted]

Fred Finell, Jr., President

FF:kt

[illegible]C DID NOT SELECT THE SAMPLE(S) TESTED, AND NO REPRESENTATIVE OF BRC HAS VISITED THE PROPERTY AND/OR PLANT AND/OR LABORATORY FROM WHERE THE SAMPLE(S) WERE TAKEN. BRC HAS NO WAY OF KNOWING IF THE SAMPLE(S) TESTED ARE REPRESENTATIVE OF THE PROPERTY, AND/OR THE METHODS OF TESTING AND/OR PROCESSING USED, AND BRC DOES NOT KNOW THE EXTENT THE SAMPLE(S) REPRESENT OF ANY VOLUME OF ORE(S) ON THE PROPERTY OR THE COMMERCIAL FEASIBILITY OF THE PROPERTY. BRC VERIFIES AND ATTESTS THE VALUES REPORTED WERE ACTUALLY RECOVERED FROM THE SAMPLE(S) PROVIDED TO AND TESTED BY BRC. NO WARRANTIES ON THE REPRODUCIBILITY OF THE SAMPLE ARE GIVEN, EXCEPT FOR IDENTICAL SAMPLE AND TEST REPEATED IN HOUSE. BRC MAKES NO WARRANTY, EXPRESS OR IMPLIED, AND ASSUMES NO LEGAL LIABILITY WHATSOEVER AS TO THE USEFULNESS OF ANY INFORMATION CONTAINED IN THIS REPORT.

As a mutual protection to clients, the public and this corporation, this report is submitted and accepted for the exclusive use of the client to whom it is addressed and upon the condition that it is not to be used, in whole or in part, in any advertising or publicity matter without prior written authorization from this corporation.

BAHAMIAN REFINING SERVICES & MINING EQUIPMENT COMPANY
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9222 N. 14th AVE., PHOENIX, ARIZONA 85021 • TELEPHONE (602) XXXXXXXX
NEW NUMBER 602-944-6577 FAX 944-1893

DATE: DECEMBER 26, 1997

SAMPLE NAME: # 1

Name: RONALD OWENS
YUKALTA RESOURCES, INC
201-5201-52 AVE
PONOKA, ALB T4J-1H6

RE: Complete Basic & Acid Workup - BIO-D-Leachent

This report is to provide you with BIO-D-Leachent amenability data on the sample you submitted; and preparation, head analyses, and leach recovery results.

SAMPLE PREPARATION

The entire sample which we received from you was dried by slow evaporation to eliminate errors that could be caused by moisture content. It was then thoroughly mixed and put through a Jones' splitter to obtain an average sample and then ground up for the test. A control sample was also taken, and will be held in storage for 30 days to enable us to do additional testing should you request it.

HEAD ANALYSES

The head analyses reported below were derived from combining the results from actual recovery of both the tailing pulp and the pregnant leach solution. (See attached sheet for the individual results of each formula used.) The total values for the best formula(s) are reported as follows in oz./ton.

FORMULA	GOLD Au	SILVER Ag
Basic # 5	1.38	.14
Acid # 5	1.34	.14

LEACH RECOVERY

The samples prepared for the leach tests for each formula were placed in appropriate sized containers, and the test was commenced. The following formulas were determined to be the best for this sample. (See attached sheet for all formulas used.)

BASIC FORMULA # 5		
DI Water	408	ml
Sodium Bromide	25	grams
Sodium Hydroxide	.50	grams
BIO-D-Leachent	.05	grams

ACID FORMULA # 5		
DI Water	408	ml
Sodium Bromide	25	grams
Acid	none	
BIO-D-Leachent	.05	grams

These tests were done as a soak leach amenability test on the samples and the leach time was 24 hours. No agitation was used and the temperature was ambient, varying from 75 degrees to 80 degrees. The oxidizing strength of the solution was maintained by monitoring with Potassium Iodide Starch Test Paper.

After 24 hours of leach time, the entire slurry for each test was filtered through a Whatman ashless No. 42 filter, and washed with 5 times the volume of the pregnant solution with deionized laboratory water to rinse out the dissolved precious metals from the pulp.

The tails pulp was dried. Then a total recovery by both hydrochemical and ferometallurgical methods was done to determine precious metals not yet dissolved by the leachent.

DI water was added to the entire volume of rinse water containing the pregnant leach solution up to a total of 1000 ml using a volumetric Erlenmeyer flask. The pregnant solution was analyzed by atomic absorption spectrograph and represents the actual values recovered from the sample by this particular testing procedure. (See attached sheet for all results.)

The percentages of recoveries are as follows:

FORMULA #	GOLD (Au)	SILVER (Ag)
Basic # 5	100	100
Acid # 5	100	100

COMMENTS

A 24 hour soak leach is a good method to determine the amenability of an ore to release its values using a certain formula, however, in practice this is not an efficient method and is not used. The amounts of precious metals recovered would be greatly increased in a far shorter time span if there were a liquid flow through the ore as is normally done in production.

Other factors that would increase the yield would be agitation, increasing the solution temperature, fine grinding, and adding an oxidizer to the solution. Any production method used would produce far greater results than the 24 hour soak leach method of testing will produce.

RECOMMENDATIONS & CONCLUSIONS

Of the 16 formulas tested, there were 6 that produced over 1 oz. per ton, and 2 of those were over 1.33 oz. per ton. This ore responds very well to BIO-D-Leachent. The indications for platinum group metals show values much greater than the gold values recovered. We have not yet received the results on the platinum metals. I will call you as soon as I have them.

I am available by phone between 2:00 and 8:00 p.m. daily to discuss this report and answer any questions you may have.

[redacted]

Fred Finell, Jr., Pres.

FF:kt

PRECIOUS METALS RECOVERY SYSTEMS

Developers of the
"MC CLOSKEY" PROCESS for
the Recovery of Gold,
Silver & Platinum Group
Metals from Ores

Specializing in the
Economical Recovery of
Precious Metals Values
from Ores

FAX FROM: John P. Mc Closkey - FAX: (909) 369-7430

FAX TO: Ron Owens - FAX: (403) 783-6487

Dear Ron:

Following are the results of a process test performed on one sample of ore for Gold only recovery:

Sample Description	Elements	Ounces/ton recovered
Sample No.1 of 3 Pulverized	Gold	0.15

Remarks:

The above test results should not be considered as an assay but is a process test which can be scaled up to any size operation desired.

John P. Mc Closkey
Chemical Consultant

Box 948, Riverside, CA 92502

PRECIOUS METALS RECOVERY SYSTEMS

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from Ores

FAX FROM: John P. Mc Closkey - FAX: (909) 369-7430

FAX TO: Ron Owens - FAX (403)- 783-6487

Dear Ron:

Following are the results of a process test performed on one sample of ore as one of 3 samples recently forwarded to the laboratory for testing:

Elements:	Ounces/ton recovered
Gold	5.4
Platinum Group	1.2

Remarks:

The sample was not pulverized further but was run AS-IS

That above test results should not be considered as assays but are process tests which can be scaled up to any size operation desired.

[redacted]

John P. Mc Closkey
Chemical Consultant

Box 948, Riverside, CA 92502
Tel & FAX (909) 923-9598

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Elements:	Ounces/ton recovered
Gold	5.4
Platinum Group	1.2

Remarks:

The sample was not pulverized further but was run AS-IS

That above test results should not be considered as assays but are process tests which can be scaled up to any size operation desired.

[redacted]

John P. Mc Closkey
Chemical Consultant

Box 948, Riverside, CA 92502
Tel & FAX (909) 923-9598

PRECIOUS METALS RECOVERY SYSTEMS

Developers of the Non-Cyanide
Non-Toxic "CIANO" Leach,
for the Recovery of Platinum,
Gold & Silver from Ores

Specializing in the
Economical Recovery of
Precious Metals Values
from Ores

BACKGROUND INFORMATION

JOHN P. McCLOSKEY

Classification: Chemical Consultant

EXPERIENCE:

1975 to Present	**Technical Director:** PRECIOUS METALS RECOVERY SYSTEMS INC., Riverside, California. Develop an economical PROCESS SYSTEM for the recovery of Gold, Silver and the Platinum group metals from ores containing these values. Also, develop an economical non-cyanide, non-toxic process for the recovery of Gold and Silver from ores, using ion exchange technology.
1961 to 1975	**Senior Research Chemist:** Rockwell International, Space Technology Research Group, Anaheim, California. Develop ultra-micro analytical chemical procedures, as required, for the analysis of new and exotic materials related to basic space technology research.
1958 to 1961	**Chief Chemist:** U.S. Army Engineers, Mediterranean Division Laboratory, Athens, Greece. Supervision of chemists engaged in the evaluation of all materials used in new construction and maintenance of U.S. Military Bases in Greece, Turkey, Pakistan, Italy, Iran, Iraq and Morocco.
1948 to 1958	**Chief Chemist:** U.S. Navy, Marianas Division Laboratory, Agana, Guam. Supervision of chemists engaged in the evaluation of all materials used in new construction and maintenance of U.S. Military bases on Guam, Saipan, Rota, Tinian, and all other islands located in the Marianas Area.
1946 to 1948	**Research Chemist:** Union Starch & Refining Co., Granite City, Illinois. Conduct research investigations, and perform chemical analyses of corn products, such as corn syrup, candies, corn sugars, such as maltose, dextrose, sucrose, etc.
1943 to 1946	**Chief Chemist:** Atomic Bomb Plant Laboratory, Oak Ridge, Tennessee. Supervision of chemists engaged in the treatment of the water in what was then the largest power house in the world. Also, special problems related to contamination of some of the return condensate being contaminated with Uranium Hexafluoride from secret atomic bomb plant processes for the separation of the isotopes of Uranium.

Box 948, Riverside, CA 92502

LILBRAT MINING

DATE: July 28, 1997

DIRECTED TO: Ronald T. Owen
[redacted]

PROJECT: Spirit River Project

SAMPLE NAME: S.R. 1

SAMPLE WEIGHT: 80 lbs

GOLD RECOVERED: 0.2 grams

EXTRAPOLATE VALUES: $\frac{0.2\text{ gms}}{80\text{ lbs}} = \frac{5.0\text{ gm}}{2000\text{ lbs}} \div 31.104 = .16$ oz/ton
Gold at $323 USD per ounce = $51.92 USD per ton

METHOD EMPLOYED: Crushed, ground to 60 minus,
scrub, table concentration, chemical scrub of concentrate
amalgamation recovery, nitric reduction.

COMMENTS:

INVOICE: Crushing, milling, recovery $300.00 Cdn. Paid.

Trusting this is of service to you.

Yours truly,

[redacted]

John Savege

311 Spruce Street, Cultus Lake, BC, Canada V2R 4Y7 Telephone: Plant (604)858-4289 Res. (604)858-9585

LILBRAT MINING

DATE: Aug. 1, , 1997

DIRECTED TO: Ronald T, Owen
[redacted]

PROJECT: Spirit River

SAMPLE NAME: S.R. 2

SAMPLE WEIGHT: 74 lbs

GOLD RECOVERED: 0.17 grams

EXTRAPOLATE VALUES: $\frac{0.17\text{ gms}}{74\text{ lbs}} = \frac{4.59\text{gm}}{2000\text{ lbs}} \div 31.104 = .148$ oz/ton
Gold at $323 USD per ounce = $47.71 USD per ton

METHOD EMPLOYED: Crushed, ground to 50 minus, Amalgam added chemical scrub, table concentration, chemical scrub of concentrate amalgamation recovery, nitric reduction.

COMMENTS: present.

INVOICE: Crushing, milling, recovery $300.00 Cdn. Paid.

Trusting this is of service to you.

Yours truly,

[redacted]

John Savege

COMPLEX METALS RESEARCH & DEVELOPMENT, INC.
P. O. Box 1137
LaVerkin, Utah 84745

Telephone: 435-635-2088
435-877-1059

June 11, 1998

Ron Owens
[redacted]
CANADA T4J1H6

SUBJECT: 93 lbs. of ground yellow iron oxide ore:

Placed in air tight leaching vessel. 10 gallons Redox solution added to Ph 2.10, Orp. 1125. One hour leach was checked for Ph 5.40, Orp. 430.

2 gallons Redox added to Ph 2.00, Orp 1150, sealed for 3 hours to Ph. 5.70, Orp. 450; Redox added (3 gallons) to Ph 1.90, Orp. 1168.

5 hours Ph 6.70, Orp. 400, 4 gallons Redox added. Ph. 2.12, Orp. 1180.

2-1/2 hours Ph 6.20, Orp. 410. 4 gallons Redox added to Ph 2.80, Orp. 1120. Ran 14 hours overnight. Ph 4.60, Orp. 450. 5 gallons Redox added to Ph 2.60, Orp. 1130. 2 hrs. Ph 4.10, Orp. 688; leach stopped. Vessel cleaned. Leach flocked and filtered. Total solution from leach 28 gallons.

2000 Ml. of pregnant solution: Ph to 9. Hydroxide added with sodium bora-hydride; heat and filtered. Filter paper was burned. Total Precip 11.4 grams. Assay with 11.4 grams of silver added.

#2
Au: 2.62 Milligrams (Had 200 mg. loss of silver)
Pt. Group - 1.4 milligrams

2 lbs. of ore was dried and ground with impact grinder. Sample was put in water with a wetting agent. Run on concentrating table. Results:

#3
Concentrate - 7.26 grms - 11.8 mg Au
Middles 636.64 grams - 0.52 mg. Au
263.18 grams - 16.04 mg Au

Raw ore - Bank run - 0.37 oz./ton Au
Slimes 0.51 oz./ton Au

Subject: Grinding and Leaching: Impact grinding the concentrating on a table will give you a commercial yield.

Slimes for leaching: I am sure the Redox will work. You must first adjust the Ph ?? before Redox.

[redacted]

Jerry C. Henderson, Research Chemist

JCH:hh

CHARGE: $800.00

PAID IN FULL.

ASSAY REPORT

Sample name or I.D.

Date 10/14/96

All analysis has been performed in good faith by using both time honored and of state of the art techniques. Results should not be used as a monetary guarantee.

INVOICE 59822 RONOWNES PHONE 303/783-6187 FAX 303/783-6187

SAMPLE TYPE:

☐ QUARTZ ORE ☐ BLACK SAND ☐ DORE

☐ ANODE MUD ☐ CORE DRILLINGS

☐ CARBON ☐ ASHES ☐ PRECIPITATES

SAMPLE TYPE OTHER ☐

SAMPLE SIZE: Approx.

SAMPLE SIZE ASSAYED IN GRAMS OR 'RATIO' 12

We reserve the right to assay any size of sample based on metal content and difficulty (Complex).

LABORATORY:

☐ WET CHEMICAL (ACID) ☐ WET CHEMICAL BASIC ☐ ACID FUSION ☐ ALKALI FUSION

☐ SELECTIVE DROP ☐ SELECTIVE LEACH ☐ BULK DROP ☐ HYDROXIDE DROP ☐ SPOT TEST

LABORATORY OTHER

FURNACE:

☐ FIRE ASSAY ☐ HIGH TEMPERATURE SMELT ☐ LEAD COLLECTOR ☐ REDUCTIVE LEAD COLLECTOR

☐ SILVER COLLECTOR ☐ COPPER COLLECTOR ☐ GOLD COLLECTOR ☐ REDUCTION SMELT

☐ CUPELED FURNACE OTHER

NOTES:

ASSAYER

ASSAY DATE 10/14/96

GRAMS RATIO 12	LEACH ML RATIO 25	GRAMS TO LEACH RATIO 2.083

COMPLETE UN-PARTED BEAD ⟶ ○ UN-PARTED BEAD WEIGHT ☐ 0 BEAD WAS TO SMALL TO RETURN ☐

READINGS SHOW TOTAL METAL CONTAINED IN THE SUBMITTED SAMPLE. CHEM-TRON DOES NOT OFFER BACK CALCULATIONS.

	OZ. PT	PPM		OZ. PT	PPM		OZ. PT	PPM		OZ. PT	PPM
PD	0.016	0.5	AG	0.000	0.0	AU	0.042	1.4	PT	0.020	0.7
RH	0.008	0.2	RU	0.031	1.1	OS	0.007	0.2	IR	0.024	0.8

OZ PT READINGS ARE MORE ACCURATE SINCE THEY ARE THREE DIGIT. PPM FIGURES ARE GIVEN IN ONE DIGIT SO SOME OF THE PPM FIGURES WILL BE ROUNDED TO THE NEXT PROBABLE DIGIT.

PD $1.92 AG $0.00 AU $16.17 PT $8.42 RH $3.30 RU $2.48 OS $0.46 IR $4.32

(34.286 PPM = 1 OZ. PT) FIGURES ABOVE ARE DOLLARS PER TON OF SAMPLE SUBMITTED TOTAL VALUE PER TON $37.07

- SAMPLE PREPARATION. $65.00 For micro wave AR digestion or $25.00 for standard type AR or Halox. (For PT group, microwave recommended).
- SPECTRAL ANALYSIS ON D2 CORRECTING ELECTRIC INSTRUMENT. Metal detection - •Gold $19.00 •Palladium $15.00 •Platinum $25.00
- Rhodium $28.00 •Ruthenium $30.00 •Osmium $45.00 •Iridium $32.00 •All 7 metals for only $99.00 *(For Silver assays call for special quote)*
- 5 ASSAY TON METAL IN HAND ASSAYS INCLUDING RETURN OF DORE' BEADS - •LEACH ASSAYS $175.00 •SMELT ASSAYS $175.00

CHEMTRON LABS 14300 DAVENPORT RD. #3B SAUGUS, CA 91350 PH. 805-298-6432 FAX 305-298-7160

ASSAY REPORT

Sample name or I.D.

Date 10/28/96
PRINT

All analysis has been performed in good faith by using both time honored and of state of the art techniques. Results should not be used as a monetary guarantee.

INVOICE 59927 RON OWNES PHONE 403,793-8487 FAX 403,793-8487

SAMPLE TYPE:

☐ QUARTZ ORE ☐ BLACK SAND ☐ DORE
☐ ANODE MUD ☐ CORE DRILLINGS
☐ CARBON ☐ ASHES ☐ PRECIPITATES

SAMPLE TYPE OTHER ☐

SAMPLE SIZE: Approx.

SAMPLE SIZE ASSAYED IN GRAMS OR 'RATIO'
10

We reserve the right to assay any size of sample based on metal content and difficulty (Complex).

LABORATORY:

☐ WET CHEMICAL (ACID) ☐ WET CHEMICAL BASIC ☐ ACID FUSION ☐ ALKALI FUSION
☐ SELECTIVE DROP ☐ SELECTIVE LEACH ☐ BULK DROP ☐ HYDROXIDE DROP ☐ SPOT TEST

LABORATORY OTHER

FURNACE:

☐ FIRE ASSAY ☐ HIGH TEMPERATURE SMELT ☐ LEAD COLLECTOR ☐ REDUCTIVE LEAD COLLECTOR
☐ SILVER COLLECTOR ☐ COPPER COLLECTOR ☐ GOLD COLLECTOR ☐ REDUCTION SMELT
☐ CUPELED FURNACE OTHER

NOTES:

ASSAYER

ASSAY DATE 10/28/96

GRAMS RATIO 10	LEACH ML RATIO 22	GRAMS TO LEACH RATIO 2.200

COMPLETE UN-PARTED BEAD ⟶ ○ UN-PARTED BEAD WEIGHT ☐ 0 BEAD WAS TO SMALL TO RETURN ☐

READINGS SHOW TOTAL METAL CONTAINED IN THE SUBMITTED SAMPLE. CHEM-TRON DOES NOT OFFER BACK CALCULATIONS.

	OZ. PT	PPM		OZ. PT	PPM		OZ. PT	PPM		OZ. PT	PPM
PD	0.023	0.8	AG	0.000	0.0	AU	0.047	1.6	PT	0.018	0.6
RH	0.007	.0.2	RU	0.029	1.0	OS	0.010	0.4	IR	0.048	1.7

OZ.PT READINGS ARE MORE ACCURATE SINCE THEY ARE THREE DIGIT. PPM FIGURES ARE GIVEN IN ONE DIGIT SO SOME OF THE PPM FIGURES WILL BE ROUNDED TO THE NEXT PROBABLE DIGIT

PD	$2.76	AG	$0.00	AU	$18.10	PT	$7.58	RH	$3.85	RU	$2.32	OS	$0.65	IR	$3.84

(34.286 PPM = 1 OZ. PT) FIGURES ABOVE ARE DOLLARS PER TON OF SAMPLE SUBMITTED TOTAL VALUE PER TON $39.09

- •SAMPLE PREPARATION. $65.00 For micro wave AR digestion or $25.00 for standard type AR or Halox. (For PT group, microwave recommended).
- •SPECTRAL ANALYSIS ON D_2 CORRECTING ELECTRIC INSTRUMENT. Metal detection - •Gold $19.00 •Palladium $15.00 •Platinum $25.00
- •Rhodium $28.00 •Ruthenium $30.00 •Osmium $45.00 •Iridium $32.00 •All 7 metals for only $99.00 *(For Silver assays call for special quote)*
- •5 ASSAY TON METAL IN HAND ASSAYS. INCLUDING RETURN OF DORE BEADS - •LEACH ASSAYS $175.00 •SMELT ASSAYS $175.00

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(ADDITION TO - DONALD E. JORDON PAPER-PAGE 131)

MAIN

add. pag.

TABLE I

PRECIOUS METALS

Troy Oz/Short Ton

SAMPLE NUMBER	DATE	IDENTIFICATION	Au	Ag	Pt	Rh	Os	Ru	Pd	Ir
EP-156 (1)	9/22/78	Anode Slimes	90.31	6594.1	0.45					
EP-1-8 (1)	10/6/78	Anode Slimes	39.48	5263.5	0.37				1.69	
LH-1-2 (1)	10/30/78	Anode Slimes	75.87	5476.9	0.31				1.94	
LH-7-8 (1)	1/29/79	Anode Slimes	104.85	6079.7	0.62				2.04	
#2224	1/28/88	Desert Ore	0.34	0.34	0.22	5.48	0.3	1.78	0.34	0.01
#2234	1/28/88	Volcanic Cinders	2.4	1.32	1.56	6.65	2.86	16.63	1.61	1.86
#2516	3/23/88	AT&T #16558	510.43	1.97	4.73	0.99	2.27	0.9	172.09	3.68
#2521	3/23/88	Magma Geothermal	6.47	0.71	2.39	34.69	21.74	4.3	32	20.86
#2532	4/1/88	Conc	829.71	28.81	86.51	94.34	204.49	471.02	663.06	3403.14
#2540	3/29/88	Unroasted Fines	27.12	2.62	4.14	55.13	5.62	16.09	19.96	73.9
#2554	3/30/88	Head Ore	0.99	0.64	1.86	26.34	0.87	1.11	1.73	0
#2537RC	5/5/88	Slag	0.49	1.9	4.13	3.94	1.17	0.67	1.24	1.13
#3174CH	8/26/88	Volcanic Cinders	5.66	0.53	4.11	4.11	5.07	8.37	4.43	16.13
#3530PK	1/3/89	Conc	4.26	0.52	0	3.06	1.17	2.47	1.68	6.36
#3568US	1/17/89	Ore	10.93	0.76	5.13	4.68	8.31	6.09	4.61	23.5
#3713JA	3/25/89	Dore'	0.1	21.8	0.01	0	0.004	0	0.004	0.009
#3755JT	4/7/89	Head Cons	10.63	10.71	8.66	11.09	12.15	22.94	21.57	35.55
#3756JT	4/7/89	Pulp Cons	5.46	81.7	9.67	7.44	21.58	13.84	6.4	37.05
#3767TN	4/10/89	Mag Cons	1.57	0.11	2.71	0		2.84	1.46	11.08
#3771TN	4/10/89	Mag Cons O2 Roasted	4.38	2.26	3.06	9.77	4.08		9.55	14.51

1-Values By DCP (8)

THESE RESULTS ARE BASED ON WELL KNOWN ACCEPTED ANALYTICAL PROCEDURES AND SOLELY ON THE SAMPLES SUBMITTED BY CUSTOMER. THIS REPORT IS PREPARED FOR THE EXCLUSIVE USE OF THE CUSTOMER ONLY. NO WARRANTIES AS TO REPRODUCIBILITY OR EXTRACTABILITY OF THE ORE IS GIVEN. DONALD E. JORDAN AND/OR ALPHA RESEARCH CORPORATION MAKE NO REPRESENTATION OR WARRANTY, EXPRESS OR IMPLIED, AND ASSUME NO LEGAL LIABILITY WHATSOEVER AS TO THE ACCURACY OR USEFULNESS OF ANY INFORMATION CONTAINED IN THIS REPORT.

TABLE 2

COMPARISON RESULTS IN TROY OZ./S.TON OF PRECIOUS METALS BY DIFFERENT LABORATORIES USING DIFFERENT FRACTIONS OF JOHNSON ASSOCIATES SLIME FROM IRON BAR

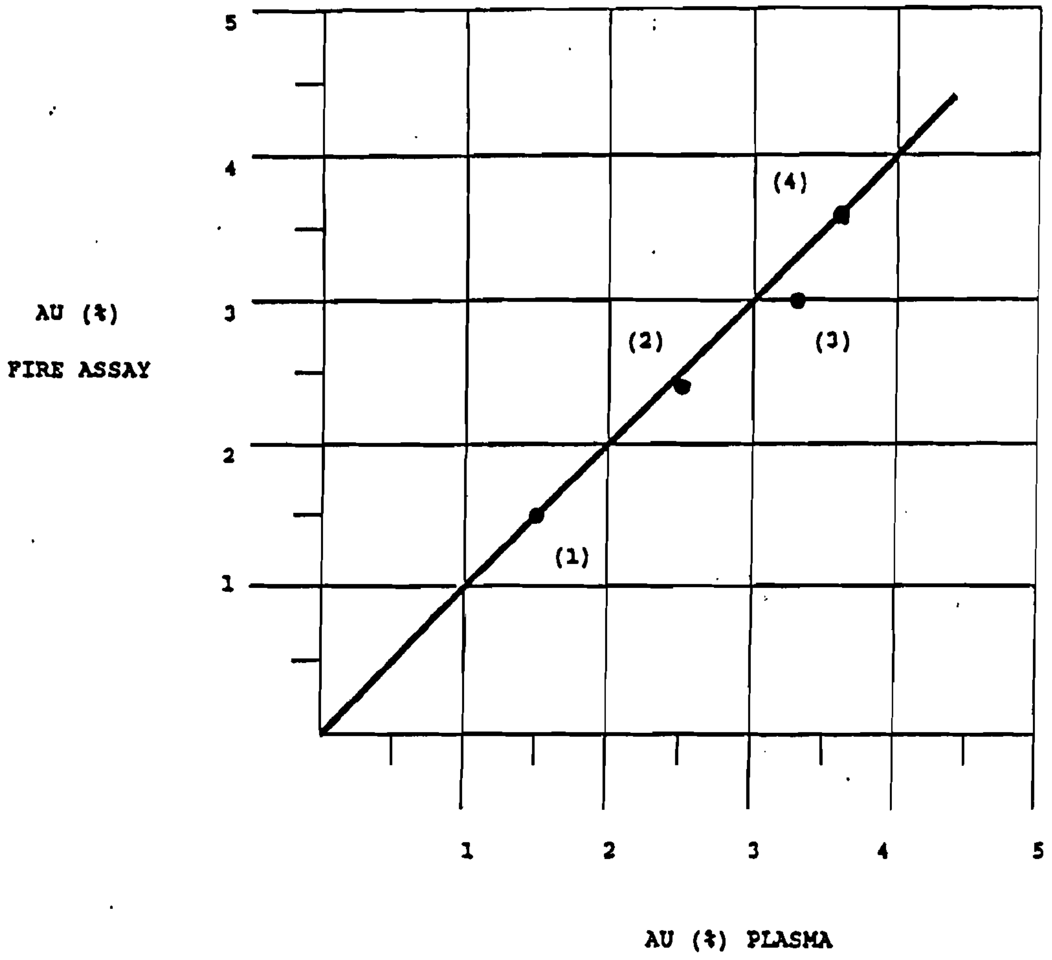
LABORATORY*	SAMPLE	Au	Ag	Pt	Rh	Os	Ru	Pd	Ir	Method
Alpha	#3946	49.3	1143.3	211.5	100	17.2	3325	240.6	82.8	High Resolution ICP
A (1)		<146	2917-5834	2917				437		XRF
B		39.4	1165	513	0.004	Tr	Tr	248	82.3	XRF
C		0.58	0.18	437.6	8.2			166.9		ICAP
D (2)		6	66	75				24		Fire, AA
E		0.16	219	304	14.6	<.2	<.2	123	<.2	XRF, SEM, FA/ICP

*THE UNNAMED LABORATORIES WILL BE IDENTIFIED UPON WRITTEN REQUEST.

(1) Preliminary results by XRF. Quantitation by DCP and others will follow.

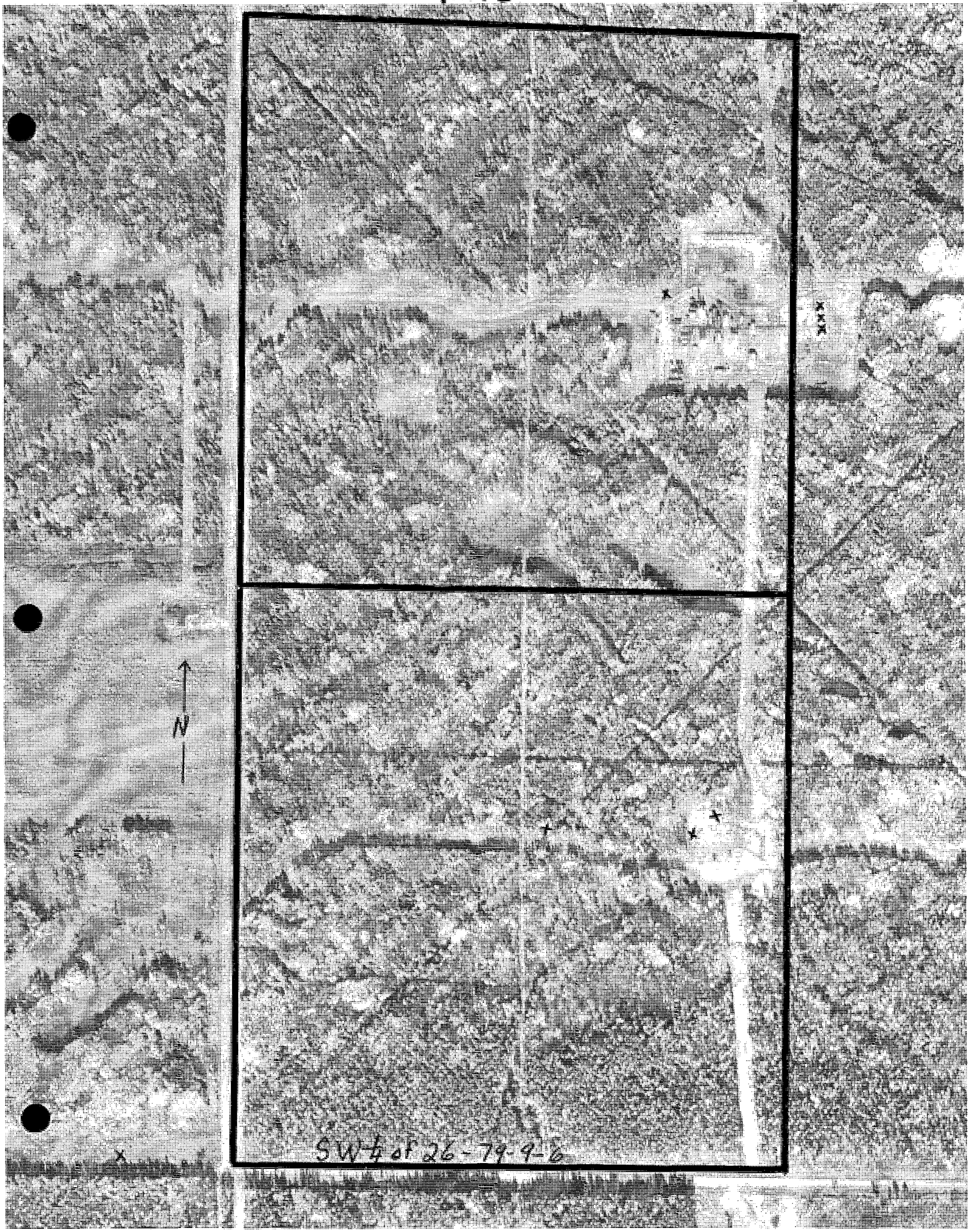
(2) Slime Fire Assayed and the prill dissolved.

Tr = Trace



	FIRE	PLASMA
1)	1.5%	1.5%
2)	2.65%	2.5%
3)	3.05%	3.43%
4)	3.6%	3.6%

FIGURE #2 19980001



FACSIMILE

PROCESS RESEARCH ASSOCIATES LTD.

9145 Shaughnessy Street
Vancouver, B.C.
Canada, V6P 6R9

TEL. (604) 322-0118
FAX. (604) 322-0181

Company:
Attention: Ron Owens
FAX Number: (403) 783-6487
From: Bryan Tatterson
No. of pages (including this page): 3

Date: May 28, 1996
Project No.: 96-041

Message : re : Metallurgical Test Work

The results to date on the Hewlett type leach on your sample are attached. The test procedure is included on the data sheets.

The calculated results show that the minus 200 mesh fraction of the Knelson concentrate assayed at least 1 g/tonne since the gold dissolved amounts to 0.945 g/tonne.

The final leach solution has been stored. Please advise if you want us to proceed through the Ion exchange and elution stages.

If you have any questions, please contact me.

Sincerely,

Process Research Associates Ltd.

[redacted]

Bryan S. Tatterson, P.Eng.,
Senior Metallurgical Engineer.

HEWLETT LEACH TEST DATA

Sample Knelson Concentrate

Test Date: April 29, 1996
Project No: 96-041

Procedure :

The cone crushed sample, at 65% solids, was milled to 80% minus 100 mesh.
The milled slurry was passed through the 3" Knelson centrifugal Concentrator.
The concentrate was collected 10 times to accumulate sufficient sample for leaching.
The accumulated concentrate was screened at 100 and 200 mesh.
The minus 200 mesh concentrate was diluted to 20% solids for leaching with overhead agitation.
An ozone/air mixture was bubbled through the slurry throughout both stages of the leach.
Concentrated HCl and salt (NaCl) were added to each give a solution strength of 10%.
65% $Ca(OCl)_2$ was added stagewise to increase the ORP to above 1000 mV.
The oxidation stage was continued overnight with continuous agitation.
GeoBrom was then added and the leach continued for a further two hours.
Solution samples taken during the leach were analysed for Au & Ag by direct AAS.

Sample Wt. 174 g

Residue Wt. 137.9 g

Time Hrs	Slurry weight (g)	Solution Volume (mL)	10% HCl Added (g)	NaCl Added (g)	65% $Ca(OCl)_2$ Added (g)	GeoBrom Added (g)	ORP (mV)
0.00	869	695	695				403
0.25							526
0.25				69.6			536
0.58							554
0.58					5.0		1102
17.82							520
24.25					37.2		1060
24.25						10.6	1076
24.75							1066
25.75							1062
26.25	918	687					1062

Notes
Addition of the solid $Ca(OCl)_2$ was carried out in stages. The ORP increased and then dropped.
GeoBrom was only added when the ORP stabilized above 1000 mV.
The first solution sample was taken 15 minutes after adding the GeoBrom
The leach was ended when the AAS results showed a levelling off.
The direct AAS Au readings on the samples are subject to interference from Fe.

	Time min	Assay Method	Solution Assays (mg/L) Au	Ag	Pd	Pt	Rh
Sample 1	60	AAS	8.1	1.0			
Sample 2	90	AAS	8.2	0.8			
Sample 2	120	FA/AAS	0.239	2.1	< 0.005	< 0.015	< 0.025

	Calculated g/tonne Au	Ag	Pd	Pt	Rh
Calculated Head (- 200# Knelson Conc) based on Leach solution assay	0.945	8.31	< 0.020	< 0.060	< 0.1
Calculated Head (Knelson Feed) based on Leach solution assay	0.015	0.13	< 0.0003	< 0.0009	< 0.002

SIZE ANALYSIS REPORT

Project no : 96-041 **Date :** May 9, 1996

Sample description : Iron Ore

Grind time : 8 minutes per 2.5 kg charge

Procedure : The cone crushed sample, at 65% solids, was milled to 80% minus 100 mesh.
The milled slurry was fed to a 3" Knelson Centrifugal Concentrator.
Jacket water pressure was maintained at 1.5 psig
Concentrates were removed ten times during the process.
The accumulated concentrates were screened at 100 & 200 mesh
The minus 200 mesh fraction was leached for precious metals.

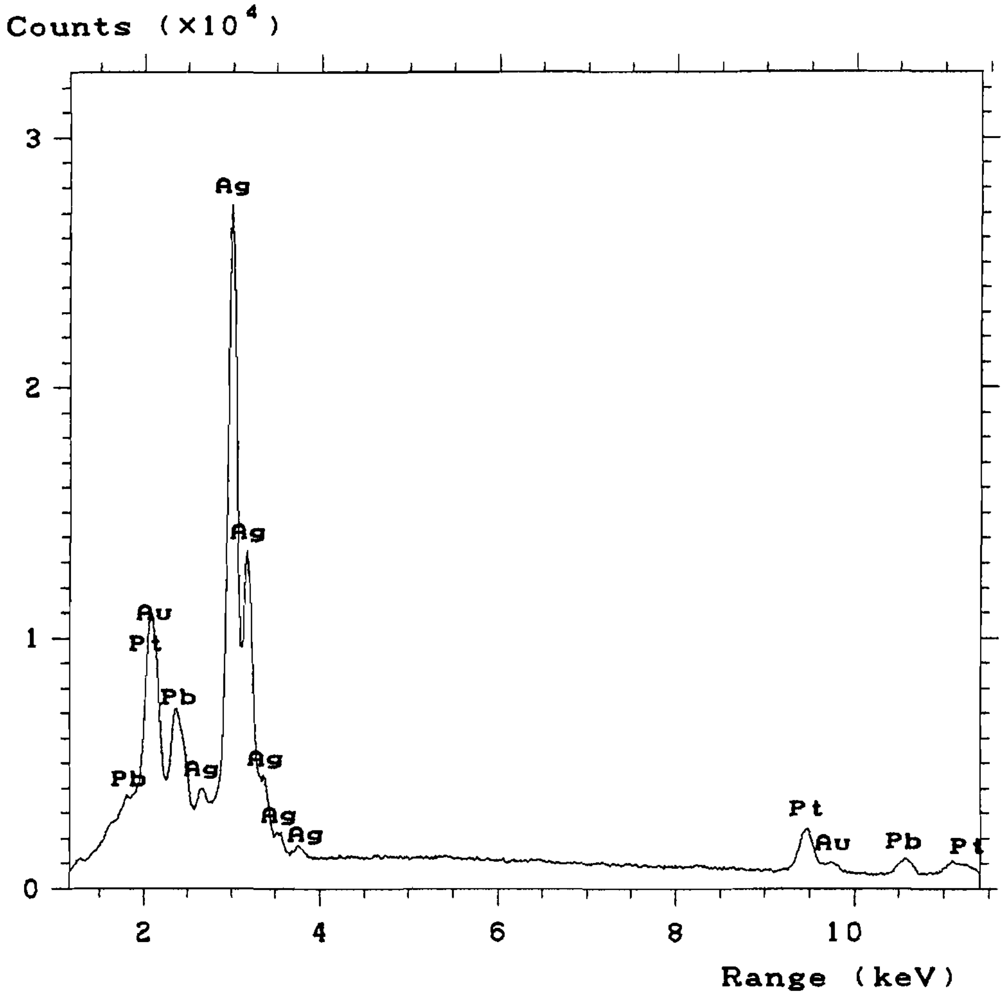
Sample	Size Fraction (Tyler mesh)	Knelson Concentrates		Total Sample	
		Individual Percentage Retained	Cumulative Percentage Passing	Individual Percentage Retained	Cumulative Percentage Passing
Knelson Conc.	100	29.9	70.1	2.1	97.9
	200	47.9	77.8	3.4	94.5
	- 200	22.2		1.6	93.0
Knelson Tail				93.0	
TOTAL		100.0		100.0	

DRAFT

```
#2                                                 LIVETIME(spec.)=    300
 ENERGY     RES      AREA
-    7.4   71.93   152860
TOTAL AREA=  1261500
...
Peak at     .38 keV omitted?
Peak at     .64 keV omitted?
Peak at    1.80 keV omitted?
Peak at    2.10 keV omitted?
Peak at    5.42 keV omitted?
Peak at    6.40 keV omitted?
FIT INDEX=**.**

ELMT         APP.CONC    ERROR(WT%)
AgL  :  1    23.308        .138
PbL  :  1    31.819        .607
AuL  :  1     3.623        .293

ZAF CALCULATIONS


..[ 2 iterations]

 0.00 kV   TILT =    .00 ELEV = 35.00 AZIM =    .00 COSINE = 1.000

Spectrum: #2                                         CHEM.&MATERIALS ENG.

All elmts analysed,NORMALISED

ELMT    ZAF Ratio %ELMT        Error         ATOM.%
AgL  :  1  .800    42.954 +-    .253          58.994
PbL  :  1  .914    51.309 +-    .979          36.689
AuL  :  1  .930     5.740 +-    .464           4.317
TOTAL             100.002                    100.000
----------------------------------------------------------------------------
```

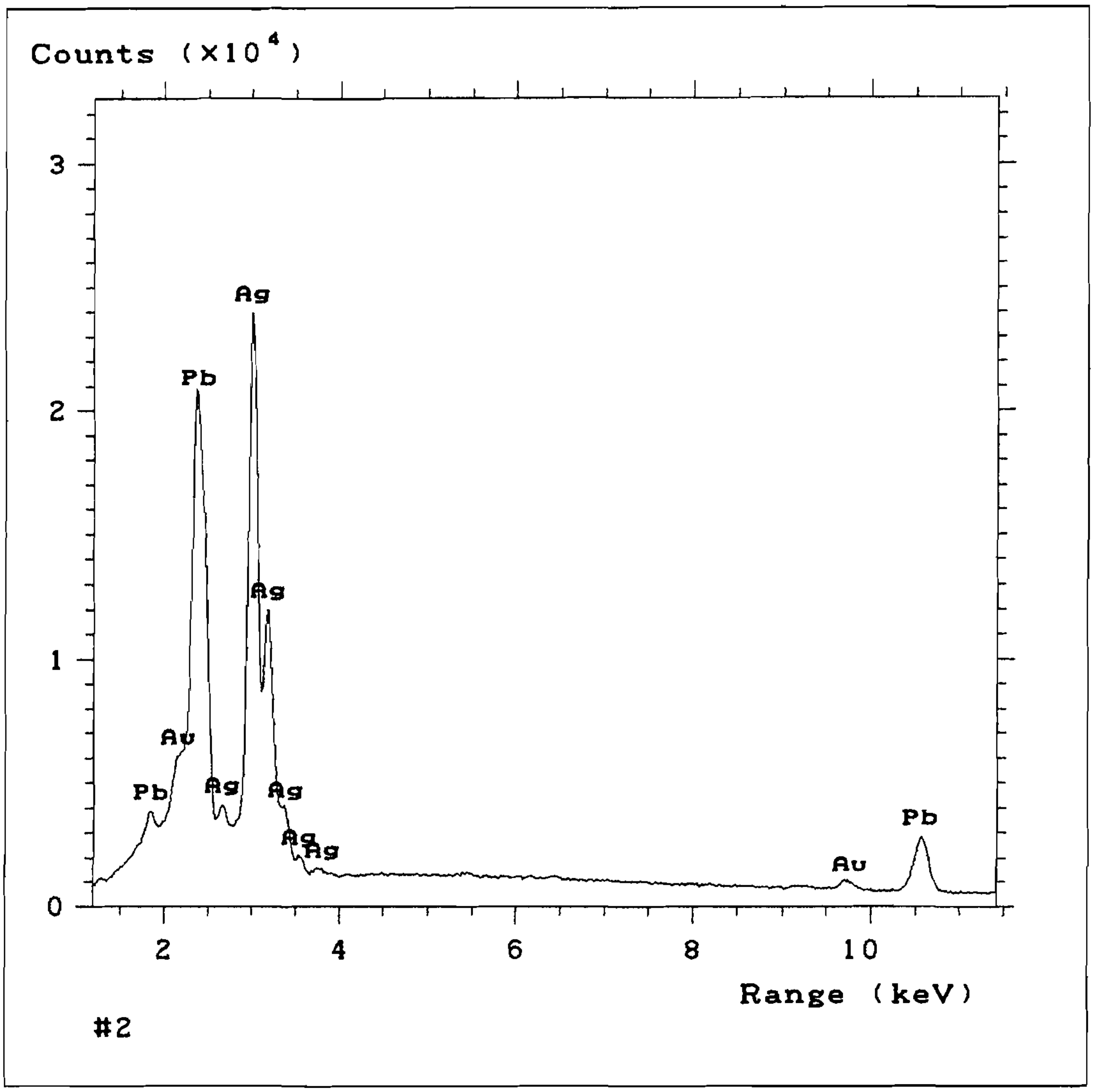
```
#2 (Cut Surface)                           LIVETIME(spec.)=    300
 ENERGY     RES      AREA
-   7.4    72.25    158195
TOTAL AREA=   1303169
...
Peak at     .40 keV omitted?
Peak at    1.24 keV omitted?
Peak at    1.78 keV omitted?
Peak at    2.14 keV omitted?
Peak at    5.40 keV omitted?
Peak at   19.54 keV omitted?
FIT INDEX= 9.11

ELMT         APP.CONC    ERROR(WT%)
AgL  : 1     44.553       .182
PbL  : 1      1.656       .466
AuL  : 1      9.869       .374
BiL  : 1      1.079       .495

ZAF CALCULATIONS


..[ 2 iterations]

20.00 kV  TILT =    .00 ELEV = 35.00 AZIM =    .00 COSINE = 1.000

Spectrum: #2 (Cut Surface)                          CHEM.&MATERIALS ENG.

All elmts analysed,NORMALISED

ELMT    ZAF Ratio %ELMT          Error           ATOM.%
AgL  : 1   .891    77.544 +-     .317            86.450
PbL  : 1   .857     2.995 +-     .843             1.739
AuL  : 1   .874    17.506 +-     .664            10.688
BiL  : 1   .857       952 +-     .896 <3 sd       1.123
TOTAL                 997                       100.000
- - - - - - - - - - - - -     - - - - - - - - - - - - - - - - - - - - - - - - - - - - - - - -
```

```
#3                                     LIVETIME(spec.)=    300
 ENERGY     RES      AREA
-   7.5   72.46   146023
TOTAL AREA=  1198843
...
Peak at    .38 keV omitted?
Peak at   1.58 keV omitted?
Peak at   1.76 keV omitted?
Peak at   2.06 keV omitted?
Peak at   6.40 keV omitted?
Peak at   7.10 keV omitted?
Peak at  19.54 keV omitted?
FIT INDEX=40.16

ELMT        APP.CONC    ERROR(WT%)
AgL : 1    26.511        .145
PbL : 1     9.372        .462
AuL : 1     3.081        .317
PtL : 1    16.372        .415

ZAF CALCULATIONS


..[ 2 iterations]

20.00 kV  TILT =    .00 ELEV = 35.00 AZIM =    .00 COSINE = 1.000

Spectrum: #3                                        CHEM.&MATERIALS ENG.

All elmts analysed,NORMALISED

ELMT    ZAF Ratio %ELMT        Error         ATOM.%
AgL : 1  .811    50.742 +-     .277          65.530
PbL : 1  .898    16.200 +-     .799          10.892
AuL : 1  .914     5.227 +-     .538           3.697
PtL : 1  .912    27.842 +-     .707          19.881
TOTAL           100.011                     100.000
---------------------- -------------------------------------------------------
```

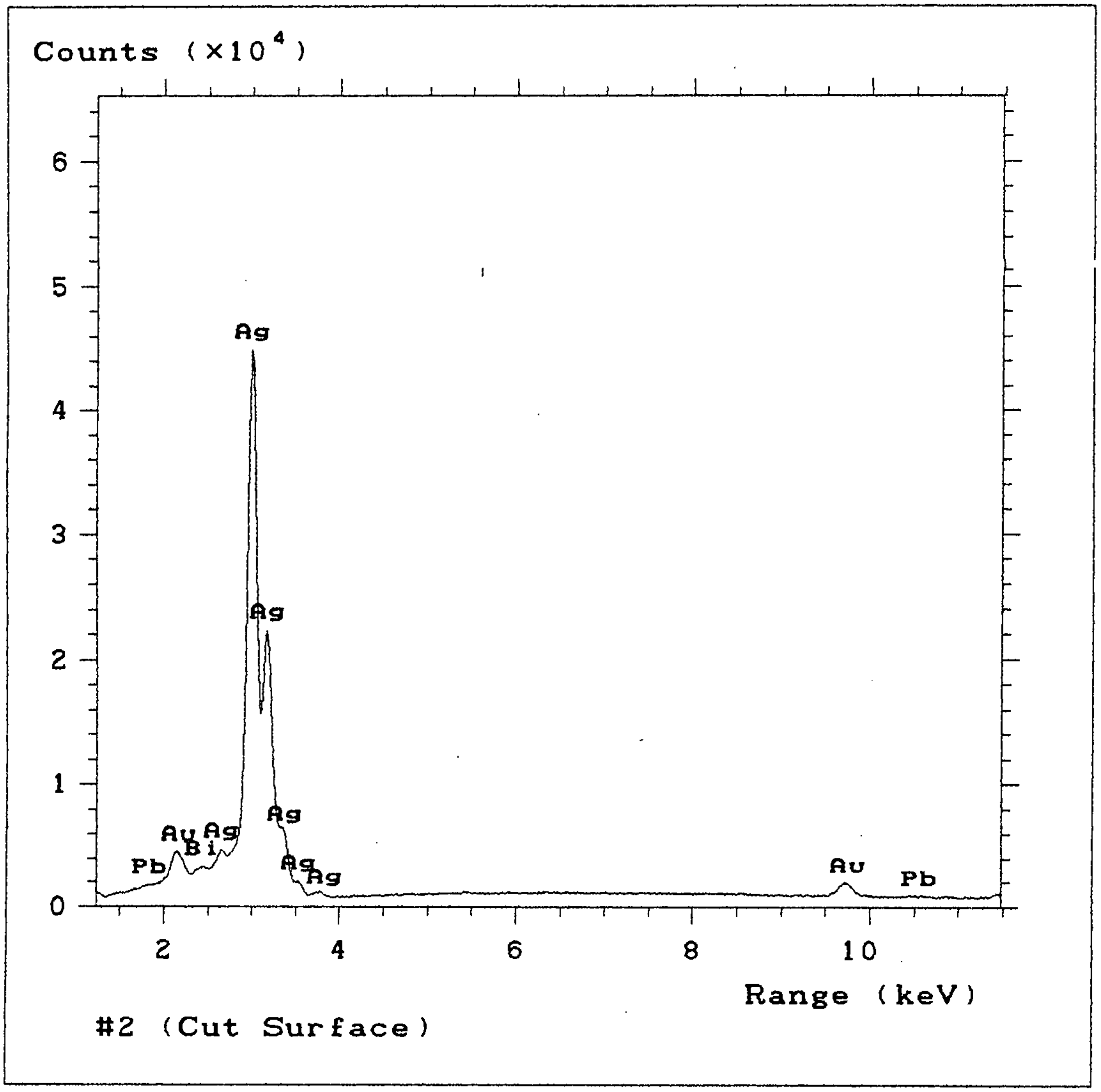

Counts (×10⁴)
3
2
1
0
Ag
Ag
Au
Pt
Pb
Pb
Ag
Ag
Ag
Ag
Pt
Au
Pb
Pt
2
4
6
8
10
Range (keV)

#3

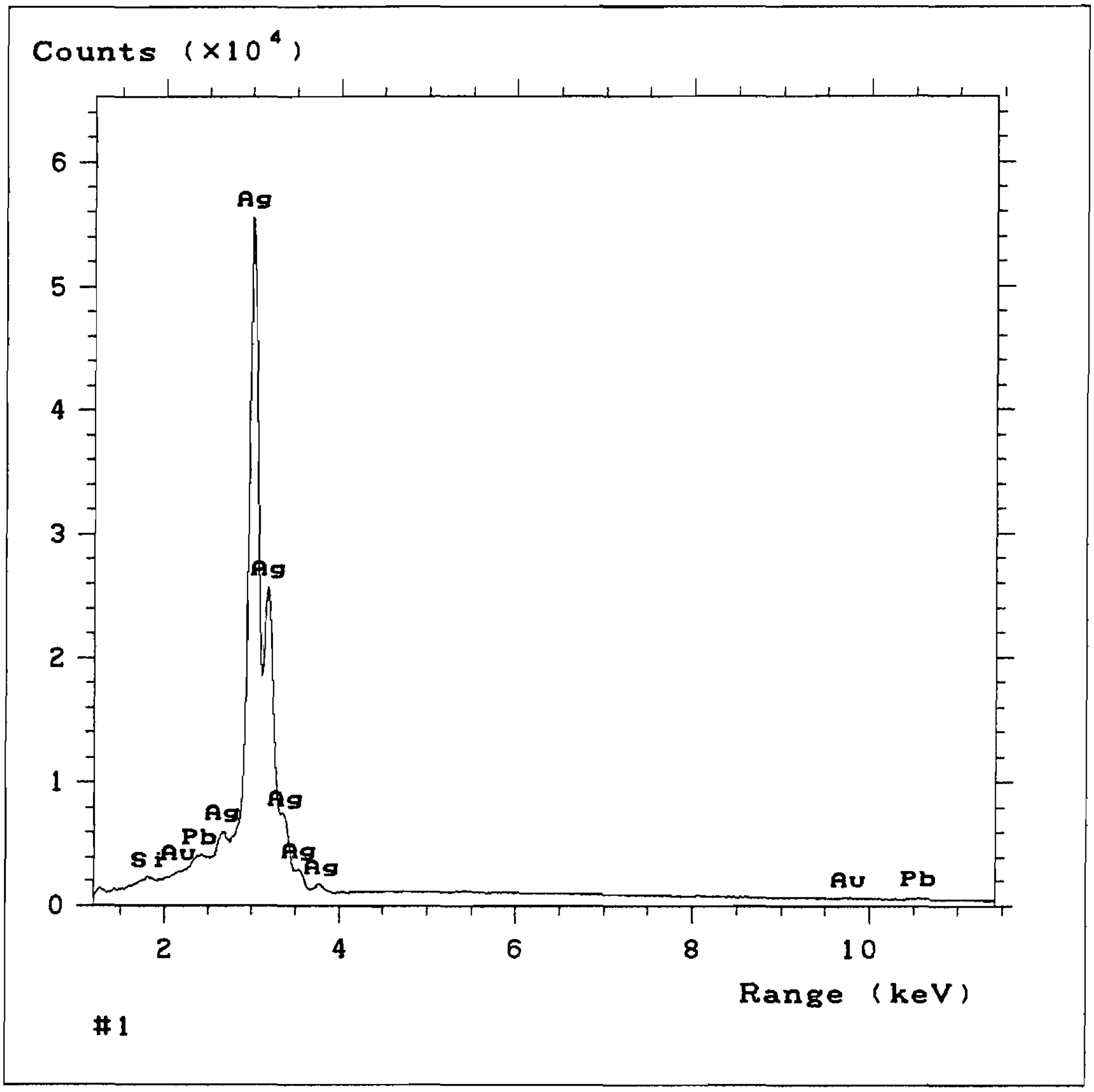
```
 LIVETIME=    100      I/P=   3601 cps

 ENERGY     RES      AREA
-    7.5   72.32     44045
 8040.9  160.48    127702

 TOTAL AREA=    360140 GF=   50.045

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

Spectrum file : IAL1
#1                                          LIVETIME(spec.)=    300
 ENERGY     RES      AREA
-    7.6   72.16    158310
TOTAL AREA=   1303005
...
Peak at     .40 keV omitted?
Peak at    1.26 keV omitted?
FIT INDEX= 5.36

ELMT        APP.CONC    ERROR(WT%)
AgL  :  1   55.013       .201
PbL  :  1    1.586       .375
AuL  :  1     .715       .245
SiK  :  1     .114       .018

ZAF CALCULATIONS


..[ 2 iterations]

20.00 kV  TILT =    .00 ELEV = 35.00 AZIM =    .00 COSINE = 1.000

Spectrum: #1                                     CHEM.&MATERIALS ENG.

All elmts analysed,NORMALISED

ELMT   ZAF Ratio %ELMT          Error          ATOM.%
AgL  :  1  .975   95.117  +-    .347           96.529
PbL  :  1  .833    3.210  +-    .758            1.696
AuL  :  1  .850    1.418  +-    .486  <3 sd      .788
SiK  :  1  .760     .253  +-    .039             .987
TOTAL             99.998                      100.000
-----------------------------------------------------------------------------
```


Counts (×10⁴)
3
2
1
0
Ag
Pb
Ag
Au
Pb
Ag
Ag
Ag
Ag
Au
Pb
2
4
6
8
10
Range (keV)
#2

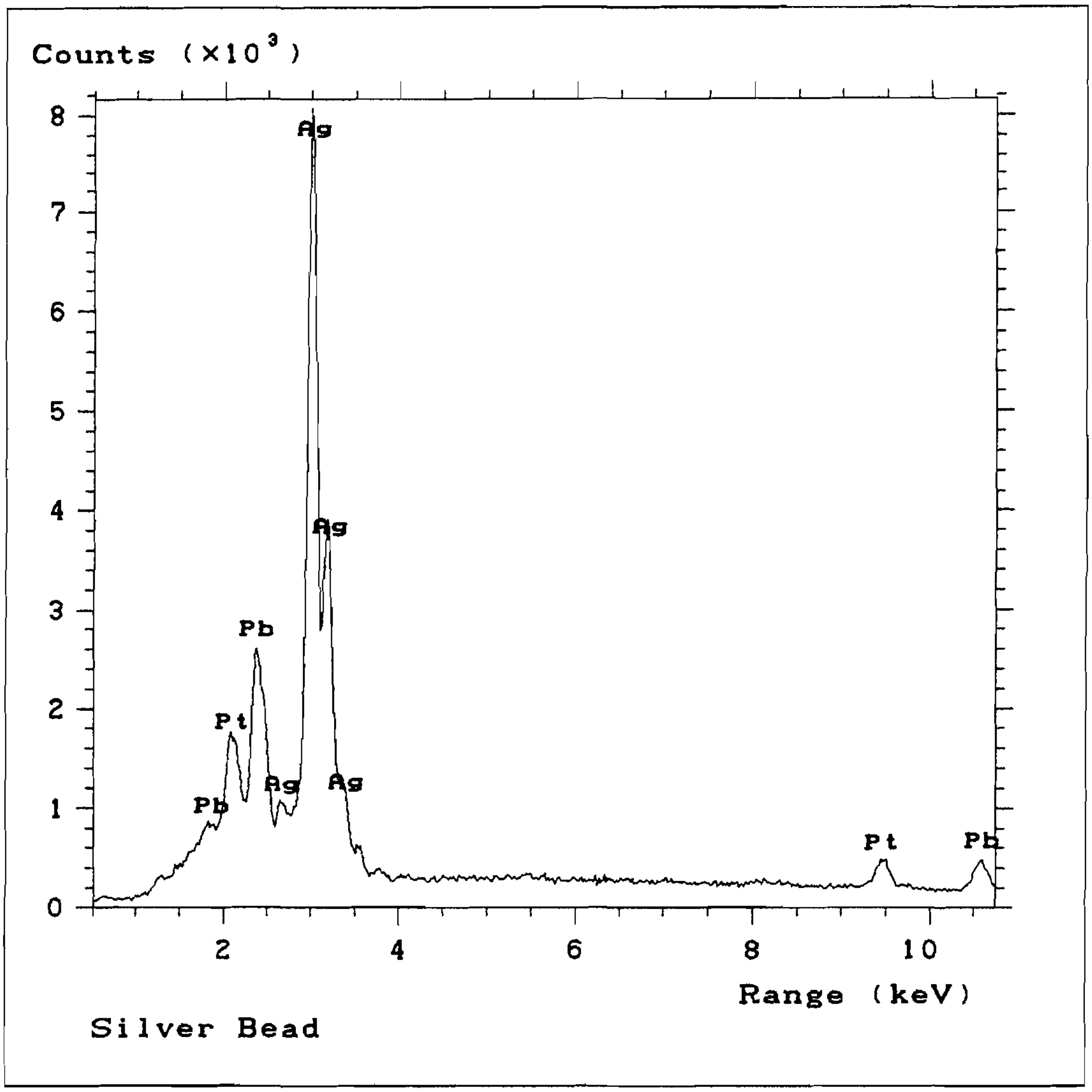
```
Spectrum file : IAL2
#2                                     LIVETIME(spec.)=   300
 ENERGY     RES      AREA
-   7.4   71.93   152860
TOTAL AREA=  1261500
...
Peak at    .38 keV omitted?
Peak at    .64 keV omitted?
Peak at   1.80 keV omitted?
Peak at   2.10 keV omitted?
Peak at   5.42 keV omitted?
Peak at   6.40 keV omitted?
FIT INDEX=**.**

ELMT         APP.CONC   ERROR(WT%)
AgL  :  1    23.308       .138
PbL  :  1    31.819       .607
AuL  :  1     3.623       .293

ZAF CALCULATIONS


..[ 2 iterations]

20.00 kV  TILT =    .00 ELEV = 35.00 AZIM =    .00 COSINE = 1.000

Spectrum: #2                                    CHEM.&MATERIALS ENG.

All elmts analysed,NORMALISED

ELMT    ZAF Ratio %ELMT        Error         ATOM.%
AgL  :  1  .800   42.954  +-    .253          58.994
PbL  :  1  .914   51.309  +-    .979          36.689
AuL  :  1  .930    5.740  +-    .464           4.317
TOTAL            100.002                     100.000
-----------------------------------------------------------------------------
```



#2 (Cut Surface)


Counts (×10⁴)
6
5
4
3
2
1
0
Ag
Ag
Ag
Ag
Ag
Ag
Si
Au
Pb
Au
Pb
2
4
6
8
10
Range (keV)
#1

```
 Spectrum file : GC1     Spectrum : Gain Calibration (Cu)
 LIVETIME=   100    I/P=  3601 cps

 ENERGY     RES      AREA
-   7.5   72.32     44045
 8040.9  160.48    127702

 TOTAL AREA=   360140 GF=  50.045

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

Spectrum file : OCO1
Silver Bead                          LIVETIME(spec.)=   100
 ENERGY     RES      AREA
-   7.5   70.72     43895
TOTAL AREA=   321410
...
Peak at    .40 keV omitted?
Peak at   1.78 keV omitted?
Peak at   2.06 keV omitted?
Peak at   5.40 keV omitted?
FIT INDEX=20.69

ELMT       APP.CONC   ERROR(WT%)
PbL : 1   13.278       .823
PtL : 1    7.922       .583
AgL : 1   23.505       .235

ZAF CALCULATIONS


..[ 2 iterations]

20.00 kV  TILT =   .00 ELEV = 35.00 AZIM =   .00 COSINE = 1.000

Spectrum: Silver Bead                          CHEM.&MATERIALS ENG.

All elmts analysed,NORMALISED

ELMT   ZAF Ratio %ELMT       Error        ATOM.%
PbL : 1  .893   28.594 +-   1.772         18.893
PtL : 1  .908   16.785 +-   1.236         11.778
AgL : 1  .828   54.629 +-    .547         69.329
TOTAL          100.007                   100.000
-----------------------------------------------------------------------------
```


Counts (×10³)
8
7
6
5
4
3
2
1
0
Ag
Ag
Pb
Pt
Ag
Ag
Pb
Pt
Pb
2
4
6
8
10
Range (keV)

Silver Bead

Assaying in Resource Evaluation: The Need for a Clear and Open Mind

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Abstract

This article is concerned mainly with assaying of precious and noble metals, gold, silver, and platinum group metals.

The determination of the amount of saleable metals (or other mineral products), in an ore is the most important step in evaluation of a mineral deposit. Most assays, even if well done, will not tell you how the material occurs in the rock or how much of it can be recovered, but if the amount that is present in the first place is unknown, it is impossible to make any comment as to the efficiency of a recovery system. This may not be important because if a recovery can be made which will provide a large profit, that which is not recovered will not seriously affect the economics of the operation. However, where the grade, and thus the operation is marginal, this is a different matter.

Fire assaying, with its origins in antiquity, is widely regarded as the best, if not the only, way to determine precious metals in naturally occurring ores. There are a lot of misconceptions about the universal applicability of fire assay methods and the more recent combination of traditional fire assay collection techniques with laboratory instrumentation has given rise to a serious, if not ludicrous, situation in the case of some materials.

There are some naturally occurring materials which cannot be successfully determined by fire assay. Some of those same materials can only be determined by instrumental techniques if their chemistry is understood.

Even the most sophisticated instruments in the world will not yield the correct answers if the chemistry of the ores being tested is not taken into consideration in designing the sample preparation and instrumental procedures to be used. Such considerations can only be made if the chemistry is understood, and such proper understanding requires the availability of a mind open to the fact that there may indeed be something new (to man) under the sun. It is important to appreciate the implications of the fact that naturally occurring materials may be in an unusual form; hence the need for a clear as well as an open mind.

The evaluation of precious metal deposits of a certain type involves an appreciation of their setting in terms of plate tectonics models.

Case histories of assaying problems with ores from the southwestern United States are presented which illustrate all of the above points. Attention is drawn to the possible implications of the phenomena described for mineral exploration in the UK as well as worldwide.

Assaying is probably the most critical step in evaluating a mineral deposit but it is one which appears to get less than its fair share of serious attention in general discussion and review. One has only to scan the calendars of the many technical and professional meetings to see the vast numbers of meetings, symposia, short courses, etc., oted to mineral exploration and mining projects worldwide and if one takes the time to browse through the abstracts, it becomes clear that the papers are broadly of three main types:

1) descriptive case histories either of exploration projects or new mine developments;

2) theoretical or academic papers which, all too frequently, present a variety of facts to fit a chosen model or concept for a particular type of deposit;

3) papers on geostatistical ore reserve estimation and/or financial investment analysis.

Occasionally, mineral evaluation is the main topic of a particular short course or symposium and in these, most often, less than 10%, probably closer to 5%, of the meeting is devoted to basic geological or other scientific information.

As an example, a short course given in London in 1990 on Mineral Evaluation contained a total of 26.5 hours of presentations including a one-hour workshop on the use of spread sheets. Of that, only 1.5 hours, less than 6% of the total, was devoted to "Geological and Processing Variables Determining Economic Capacity" which included such things as average grade and reserves; cut-off grades; average grades and reserves above cut-off; dilution and mill recovery; and the economic capacity decision, whatever that is.

One would not wish to give the impression that any, or all, of these exercises are unimportant or unnecessary but it would appear that the acquisition of the data, the actual numbers which go to make the meat of all these sophisticated decision making processes are considered to be either: (a) unimportant, or (b) so easily and readily obtained as not to pose any problem in their determination.

It is, however, pertinent to inquire whether either of these two suppositions is correct and it is important to recognize a distinction between the two kinds of information with which we are dealing.

It is doubtful that anyone would dispute the fact that there are a number of different ways of crunching the numbers in order to reach the all important decision of whether to invest or not to invest; or whether a particular project will "fly" or not. The fact that some companies will "pass" on a particular project, while others will jump in with both feet, testifies to this. The particular circumstances or objectives of one group are often quite different from those of another and so the numbers on which decision making is based are to some degree subjective. This is not the case when it comes to assaying, or at least it should not be the case. The amount of a particular commodity in a given body of rock is not variable, it is finite, although one would acknowledge that this is not always apparent from the assay certificates produced by different laboratories, and it is recognized all too well that the only time one really knows how much metal is, or rather was, in an orebody is after it has been mined out. To return to our two suppositions then:

(a) Obviously they cannot be unimportant, the content of metal or other valuable commodity and the amount that can be recovered are the critical factors in determining whether a naturally occurring material is ore. There are, of course, other factors to be taken into consideration; the cost of extraction, which involves the cost of power, of labor, of capital equipment, etc.; the costs of transportation and marketing; the cost of money; and the cost of safeguarding the environment. It is essential that the grade of the material is known, i.e., how much copper, zinc, gold, or platinum is in the rock.

(b) The second supposition, that this vital information is so easily and readily obtained as not to pose any problem in its determination, is the real crux of the problem. In many situations this is, in fact, the case; generally speaking, there is no problem in determining the amount of zinc or lead in a Mississippi Valley type ore, or of copper in most of the mines in the Zambian Copperbelt and in porphyry copper ore, or of uranium at Key Lake, Blind River and Rossing, or of gold in many of the world's great mines. There are, however, some precious metal deposits which do present considerable problems in their accurate determination, d it is on these that this article is used.

The term "some precious metal deposits" refers to gold, or gold plus PGM (platinum group metals) occurrences, which do not respond in the normal way to standard assay procedures. In fact, in our present state of knowledge, they cannot be satisfactorily determined by standard assay laboratory procedures; specifically fire assay will not produce the correct result. Now, in many jurisdictions, and indeed in many corporate offices "them's fighting words," and if one is rash enough to utter them in audible tones one runs the serious risk of being branded a knave or a fool, or both. However, there is plenty of evidence to demonstrate that they are true and will find acceptance by those willing and able to approach the matter with an open mind. The author's position with respect to such materials is borne of personal experience over many years as a practical, hands on, exploration geologist.

The Nature of the Problem

The sub-title of this article indicates the problems are only partly to do with geology, chemistry and physics. Woolly thinking and closed minds, some of them very tightly closed, are also significant players.

If we run through the practical, scientific aspects of these problems, the other less tangible aspects will become apparent. Fire assaying is not the only problem area; indeed, it may be the least important in terms of the real problem, but it is the one which attracts the most attention, and causes the biggest controversy.

Two important things happened in the years immediately prior to the 1970s and in the 20 years or so since then. First, we lost a whole generation, or perhaps two generations, of fire assayers; with the price of gold fixed for so long at $35 an ounce, it did not matter very much whether the amount of gold in a sample was 0.01 or 0.1 ounce per ton and, thus, fire assaying became something of a lost skill. Many of the old time assayers were more artists than scientists and took a great deal of trouble to get the flux right; they could tell very often just by sight, or by the application of some simple preparatory procedure, whether the ore was acidic or basic and would adapt the flux accordingly, and if they did not get it right the first time they would repeat the assay. More importantly, they knew that fire assaying is not a universally applicable technique as indicated by some of the old texts on assaying. How many laboratories these days have people like that, and if they do, how many of them could afford to repeat an assay?

The modern assay laboratory is designed to handle several hundred, or several thousand samples a day and their price structure is based on volume. One has only to review the current laboratory price lists to see the problem. In Vancouver, as an example, most laboratories will charge about $11 for a gold-plus-silver fire assay with an extra $3.75 or so for sample preparation. When the poverty-line salary level in Canada is $26,000 pa, what kind of an assay are you going to get for $15? Certainly not one that

Continued pg. 64

Assaying-
continued from pg. 10

requires more than a cursory inspection of the ore, or the slag, or of the lead button. One sample is going to be treated exactly the same as the next sample even though the character of the two samples may be quite different. Anyone who thinks that a $15 precious metal assay on a complex ore is going to be accurate on any but an accidental basis is not living in the real world. Even worse these days, some laboratories are offering instrumental precious metal assays, acid digestion/ICP or similar procedure for $3.75, and they would have you believe that they have a foolproof computer program that will correct for all the matrix effects and interferences! For a straightforward simple gold in quartz ore this may be accurate enough, but who is going to decide whether you go for the $3.75 or the $15 assay, and if you have a complex ore is it really going to matter? You are probably not going to get the right answer in either case. This is not to say that it cannot be done; perhaps it can if you have the right equipment and have taken the trouble to prepare proper standards, and have an ICP instrument with a 0.02 Angstrom echelle grating to stretch the line spectrum so that you can be sure that the lines you are reading are the ones you want and are free of interferences, but there are not many commercial laboratories that have this equipment, or who can afford within their basic price structure to take enough time to go through all the desirable precautionary stages. It is important to have a clear mind about what you are paying for and to be able to make the distinction between what you think you are getting for a specific price and what you are really getting. Such decisions need to be made by geologists, not by accountants and budget managers.

The second important factor was the replacement of fire assayers with instrument technicians who are trained to know which buttons to push and which digital read-outs to record but who, for the most part, have no chemical background. They have a solution prepared by dissolving the button from a fire assay, or by digesting an ore sample in acid, or cyanide, and they know that if they aspirate this into the instrument it is supposed to give you the reading of what is in the ore. They are always going to get a reading of something but how much does the technician who pushes the buttons on the ICP instrument, for example, know about how well the fire assay has collected the gold, or the silver, or the platinum, or the rhodium, or whether the element they think they have got in solution is in fact soluble in the form in which they think it has been collected in that particular solvent. Without a solid background in practical bench chemistry, as well as in theoretical chemistry, probably very few of the technicians in the modern commercial assay laboratory are qualified to certify any but the most routine results on straightforward materials. What happens if they get an ore that contains gold plus five or six of the platinum group elements? Do they know enough about the chemistry and the physics of PGE to be sure that they have, in fact, determined them all? Our experience is that very few are sufficiently well-qualified or experienced in handling these materials to be able to give a reliable certified assay result.

One of the biggest problems with regard to fire assaying is the widespread belief that it is possible accurately to fire assay any precious metal bearing ore. People who subscribe to this dogma usually have a second article of faith which says that erratic results on replicate samples determined by fire assay are due to sampling errors, the old standby 'the nugget effect.' Of course, the nugget effect can be a problem and it is essential that sampling procedures be carried out with the utmost care. In the author's experience it is not the universal problem some think it to be and lack of reproducibility can be, and frequently is, the result of much more fundamental facts of physics and chemistry.

Table I lists the various collectors used in fire assaying, and the gravimetric and instrumental procedures which are used to quantify the precious metals collected by the firing part of the assay. Lead has been used as a collector since medieval times and is still, somewhat surprisingly, the most commonly used today. Even more surprising, perhaps, is the fact that in 1991 some assayers still believe that lead is the best collector for precious metals. Some make an enormous concession to modern knowledge and state that nickel sulphide works better for PGMs, as though all six of the platinum group elements behave the same way. Would that they did!

Table 1
Fire assay collectors and analytical methods for precious metals

Fire assay collectors:
Lead, nickel sulphide, tin oxide/tin, copper sulphide

Analytical methods:

F-AAS	Flame atomic absorption spectrometry
GF-AAS	Graphite furnace atomic absorption spectrometry
ICP	Inductively coupled plasma
DCP	Direct current plasma
ES	Emisson spectrometry
S	Mass Spectrometry
FS	Atomic fluorescence spectrometry
NAA	Neutron activation analysis
INAA	Instrumental neutron activation analysis

The facts regarding each of these collectors are summarized below:

Lead

(a) Suitable for gold and silver in the absence of interfering substances;

(b) Less useful for platinum and palladium because their high melting point requires substantial amounts of alloying constituents such as gold or (especially) silver;

(c) Even less useful for rhodium, ruthenium and iridium since these do not alloy with silver.

Iron-copper-nickel alloys

High temperature fusion methods were developed by Beamish in the 1960s to collect precious metals into these alloys in response to demands for better collecting systems but this system is now only used to a very limited extent.

Tin oxide/tin

This method was developed by Faye & Inman (1961) and has been shown to produce more consistent results with some of the more difficult gold/PGE ores but is not widely used.

Nickel sulphide

The use of this collector was developed in South Africa in the early 1970s for platinum group metals but its great drawback is its inability quantitatively to collect gold which usually is lost to the slag. Gold collection into nickel sulphide is improved by the addition of copper to the system.

Copper sulphide

The method of collecting all eight precious metals into copper sulphide, formed by use of copper oxide and sulphur, was developed by Kallman & Blumberg (1986) at Ledoux and Co. and has been shown to have a number of advantages over other methods.

The situation regarding the status of fire assaying in the industry is, perhaps, best summed up by a reply made at a hearing held by one of the Canadian Provincial Professional Engineers Associations in 1989. Counsel put the question as to whether there is any controversy regarding the use and applicability of the fire assay technique. The reply, from the sole advisor to the Regulatory Authority in that particular jurisdiction was, "No, sir, there is no controversy." Well, it is submitted that there is a controversy; witness the following statements:

> *The pyrometallurgical techniques of fire assaying whether used for the gravimetric analyses of total gold and silver values or for preconcentration prior to instrumental analyses remain the most respected procedure for analysis of precious metal values* (Bacon *et al.* 1989).

> *The chemical repertoire available* (in the late 1960s and early 1970s) *consisted chiefly of fire assay techniques involving the lead collecting system. This system, however, failed to be adequate as a support basis for instrumental techniques* (Kallman & Blumberg, 1986).

If these experts cannot agree is it not possible that the industry has a serious problem on its hands?

Time does not permit one to go into the inconsistencies present in the world of precious metal analysis today but maybe it would not be impertinent to enquire why, if these well known and well tried traditional techniques are the best, did Beamish keep publishing revisions of his classic texts, and why did Beamish & Van Loon (1977) develop their well-known wet chemical procedure for the determination of the Noble Metals? Is it possible, perhaps, that there is, in fact, a problem with the universal application of the fire assay technique? If this is the case, then perhaps there is also a professional problem, not [illegible] say a legal problem as well, with all the administrative decisions that have been rendered in a number or jurisdictions based on the inability of assayers appointed by various non-technical regulatory bodies to reproduce, by fire assay, results produced by others through the application of the principles and procedures of classical chemistry and hydrometallurgy?

There are problems also in determining some of these types of ores by instrumental techniques. The problems are not insurmountable but if one is not aware that they exist, how can steps be taken to ensure that they are overcome? Optical emission spectroscopy is a case in point. This technique came into its own about 40 years ago and it is a sign of the rapid advances made in instrumental techniques that today it is becoming quite difficult to find a commercial laboratory that still has optical emission spectrographic equipment in working order and regular use. Often it is even more difficult to find experienced operators for such equipment except, perhaps, in the laboratories of certain precious metal refiners. For those laboratories who do still provide this analytical technique, the standard procedure seems to be a fairly short burn time: 30 seconds commonly but, at most, two minutes. Attempts by the author to get a commercial laboratory to use a five minute burn were met by responses which varied from "we'll have no electrodes left if we burn it for that long" to "we do this all the time and we know how to read platinum group elements."

Table 2 is taken from Twyman (1951) and is based on a method developed by Peters (1933), for the detection and estimation of platinum metals in ores.

Peters produced a lead button by blowpipe fusion of the ore with a lead acetate fusion mixture and fluxes and cupelled these to produce a lead prill which was then spectroscopically examined by arcing in a crater of a carbon electrode using a current of 10-13 amps. By taking a series of exposures, each of 20 seconds duration with a time lag of 5 seconds between photographs, he demonstrated that the sequence of volatilization was as shown in Table 2. This work, undertaken nearly sixty years ago appears to have been largely ignored. Why? Perhaps because many assayers are not even aware of it.

If a two-minute burn is used, only part of the platinum metals will be volatilized and any quantification based on spectra so obtained will be inaccurate. If the burn lasts only 30 seconds, none will be volatilized and none will be detected. Could this be why so many assay certificates for spectroscopic analyses for platinum group elements show "non-detected" even when other methods have shown these elements to be present? It is important to have a clear mind about the capabilities of the techniques being used.

Table 2

The order of volatilization of platinum metals

Seconds									
0-20	Pb	Ag							
25-45	Pb	Ag	Au						
50-70		Ag	Au	Pd					
75-95				Pd	Ru	Rh	Pt		
100-120				Pd	Ru	Rh	Pt	Ir	
125-145					Ru	Rh	Pt	Ir	Os
150-170						Rh	Pt	Ir	Os

After Twyman (1951)

Case History: Gold & Platinum Group Elements

Some of the points raised above can be illustrated by a study with which the author has been involved since the early 1980s when asked by a client to sample a certain area in the western United States and to assay the samples for gold. The rocks are phyllites and schists which have been regionally metamorphosed to greenschist facies. Initial scepticism about the likelihood of finding any significant amounts of gold in these particular rocks, brought an assurance that gold was present, a fact proven by amalgamation. This was confirmed by the author who also recovered free gold by amalgamation from the crushed rock. Further sampling was carried out which confirmed that gold values were present in these rocks over a large area. More detailed work involved geological mapping and additional sampling to try to establish the grade of this surface mineralization and it was this which led to recognition of an assaying problem. In an attempt to solve this problem, a composite sample was prepared from over 200 large samples. The control on the preparation of this composite sample was very strict and great care was [illegible]en to ensure thorough mixing so as [illegible] produce an homogeneous sample. It was found that one could always assay very low gold values, in the range up to 0.008 ounces per ton, and that reproducibility in this range was good and within acceptable limits. Occasionally, however, values ten times (or more) greater were obtained. Even using multiple replicates under the most carefully controlled conditions, reproducibility of these higher values could not be achieved within acceptable limits. A research program, initiated to develop a reproducible assay technique and to establish a consistent grade for gold in the rocks under study, was unsuccessful. Several hundred analyses were performed using all the standard fluxes and techniques, as well as many innovative procedures, without being able to achieve satisfactory reproducibility. It was always possible to produce a bead from fire assay procedures but the beads frequently exhibited characteristics which indicated the probable presence of other elements. Amongst [illegible]e experiments carried out were a number of "spiking" procedures in which gold inquarts were added to the sample. Also, varying amounts of a gold ore, which had been assayed successfully with good reproducibility many times in the past, were added to the sample. The results were interesting and informative. They showed that often not all, and sometimes none, of the inquarted gold was recovered and, in the case of the known gold ore, results were inconsistent with calculated chemical balance equations. Strikingly, as the amount of the composite sample was increased, the amount of gold recovered diminished until a point was reached at which no gold was recovered. Clearly the composite sample contained materials which were preventing the recovery of the gold. The mechanism causing this was, at that time, a subject only for speculation.

Table 3 illustrates this phenomenon. It shows that the results, obtained by adding varying amounts of a known gold bearing sulphide ore into the composite sample ("Sawyer Master Blend"), are erratic and that, as more of the composite sample is added, the greater the discrepancy between the expected value of gold in the sample and that actually achieved.

Table 3

The effects of the addition of spiked sulphide ore into the "Sawyer Master Blend"

% Master Blend	% Sulphide	Theoretical gold* (mg)	Measured gold (mg)	Difference
100	0	0.0	0.0	
96.7	3.3	0.007	0.002	-0.005
97.6	6.4	0.001	0.002	+0.001
90.7	9.3	0.002	0.003	+0.001
87.9	12.1	0.003	0.002	-0.001
85.4	14.6	0.004	0.004	0
80.7	19.4	0.005	0.002	-0.003
74.4	25.6	0.007	0.002	-0.005
0	100	0.021	0.021	

*Theoretical gold is that which results from analysis only in the case where the ore is 100%. All other results are calculated.

Further work, using classical wet chemistry and subsequent confirmation of products of chemical separations by glow discharge mass spectrometry and other methods, confirmed that the composite sample carried significant amounts of platinum group elements. There is no doubt that the difficulty in determining the gold in this material is directly related to the presence of the PGEs.

A second composite sample was prepared in 1989 using material from six separate drill holes in another area of similar geology. This composite sample was then analyzed by fire assay, wet chemical separations, and various instrumental techniques. The results were essentially similar to those from the earlier research program. Some of this material was also given to a commercial laboratory in the UK which was asked to perform various procedures to determine its constituents. Fire assay procedures showed very minor amount of gold and essentially no platinum group elements. To convince themselves, and us, that their work was accurate, they also carried out spiking experiments which involved adding known amounts of a previously determined gold ore to the composite sample. The results are reproduced in Table 4.

These results were accompanied by the comment that "it is evident that there is reasonable agreement between the actual assay results and those expected from the dilution ratios used." This is a very significant statement which needs closer examination.

Table 4

Results obtained from spiking a composite sample with known amounts of gold ore

		Gold in g/t	
Sample	Wt % known	Calc.	Reported
A1	100.0	16.6	16.6
A2	75.2	12.5	14.2
A3	48.8	8.10	8.81
A4	24.0	3.98	3.29

Table 5

Discrepancies in gold content of spiked samples listed in Table 4

Sample A2 (14.2 - 12.5)/0.752 = 2.26 g/t MORE
(or (14.2/0.752) - 16.6 = 2.28 g/t MORE]

Sample 3 (8.81 - 8.10)/0.488 = 1.455 g/t MORE
(or (8.81/0.488) - 16.6 = 1.453 g/t MORE]

Sample 4 (3.29 - 3.98)/0.24 = -2.88 g/t LESS
(or (3.29/0.24) - 16.6 = 2.89 g/t LESS)

Table 5 shows calculations using the data from *Table 4*. It is evident that, not only are the two sets of figures (the actual assay results and those expected from the dilution ratios used) not in reasonable agreement but that, if indeed the assayer really thought that they were, one should have serious reservations about using such a laboratory again when their control is poor and they have such low expectations of their accuracy. One suspects that the presence of platinum group elements in these rocks had already been ruled out and the interpretation of results was entirely subjective.

These results confirm those of the earlier work and clearly show that there is some interaction between the gold and the PGE which interferes with their accurate determination. As more of the ore sample is added to the known sample, the amount of gold recovered diminishes. A point, therefore, could be reached at which no gold is recovered at all. Whether this interference is chemical, or physical, or both, is not clear. The scientific approach is to observe and record the facts and then to try to understand what mechanism might be causing them.

Although it is clearly not possible to go into a detailed description of the geology and mineralogy of these deposits in this article it is interesting, and may be important, to look at the geological setting of the rocks in question.

These rocks occur as part of the largest of three klippen in an area which has been described as a superb example of collisional tectonics. Lithologies present include phyllite, micaceous schists, metasedimentary and metavolcanic rocks. There are highly sheared serpentinized peridotite bodies which are at the base of the thrust sheet, probably derived from harzburgite and dunite and which may be dismembered parts of oceanic crust. They clearly have an ophiolite affinity and thus the occurrence of platinum group elements within this assemblage is not surprising.

The general area is characterized by widespread placer mining operations on most of its creeks and rivers over the last 100 years. Numerous old reports on these operations contain references to the occurrence of platinum with the gold in the gravels and black sands worked by the placer miners, and reference to official statistics published by the State Department of Geology and Mineral Industries shows that periods of highest platinum production correspond to periods of greatest placer mining activity.

Field relations indicate fairly conclusively that the source of the gold and other heavy minerals in the drainage system is the package of rocks from which the samples referred to above were collected. What was puzzling is why this source had not been recognized and commercially exploited in the past. The inability to assay these rocks by standard techniques might have obscured the fact that they are the source of the gold and PGM. Also, surface exposures of these rocks do not suggest that they might be lik
candidates. It is only when the ass
ing problem is recognized that the
ology of the area begins to make sen

Two questions remain to be
dressed: I) Why are they so difficul
assay? 2) Are there any geologi
conditions which may be contributi
to the problem?

This article does not intend to
dress these questions in detail,
stresses the point that assaying
mineral resource evaluation is not
ways as straightforward as it may see
However, some generalized commer
may be appropriate.

To address the first point, they
difficult to assay because they do
behave in the manner which, on
basis of normally accepted practi
we expect them to behave. There
good independent evidence which c
firms that there is significant gold a
PGMs in these rocks but normal ass
techniques usually detect only part
the gold, and frequently, none of
PGEs. The spiking experiments,
scribed earlier, clearly demonstrate
there is an interference effect in ope
tion against the successful applicati

of standard fire assay procedures. Also, the apparent variation in composition of the same composite sample reported by different laboratories using other standard techniques such as X-ray fluorescence, INAA, optical emission spectroscopy, and instrumental techniques (such as AA or ICP) on fire assay prills, or by digestion by acid or other chemicals, is evidence enough, at least to the author, that they have unusual properties. Some of these elements may not occur in normal metallic form and may not be detectable by the standard instrumental techniques.

The problem is put succinctly by Duncan & Rouvray (1989) as follows:

> small aggregates of atoms constitute a distinct phase of matter. Their chemistry, at once highly reactive and selective, has possible applications in catalysis, optics, and electronics.

We know that there is a great deal of research being carried out at present into the behavior of elements in small aggregates and at least one learned society, the Faraday Division of the Royal Society of Chemistry, considers it important enough to have devoted a three-day symposium to it in September 1991. Their circular for General Discussion 92, announcing this symposium states:

> There is growing interest, ranging from the theoretical to the preparative, in dispersed and supported particles of minute dimensions consisting of less than ten and up to several million atoms. These possess a wealth of electronic, magnetic, chemical, and electrochemical properties. They also constitute formidable challenges with regard to evolving reliable methods for their characterization and the determination of their internal and surface structure.

That materials having these unusual properties do exist, is an accepted fact among scientists in other disciplines. Is there any reason to believe that they cannot, or do not, exist as naturally occurring material in a geological setting, i.e., in rocks?

The details of the case history cited in broad and generalized terms above, suggests that there are naturally occurring precious metal ores which exhibit some unusual characteristics. Materials which do not initially show detectable levels of an element on standard instruments have been shown to contain significant concentrations after specific treatment.

As to the second question posed earlier, the evidence is far from complete, or from being convincing, but there is a suggestion that the conditions required for the formation of these unusual materials may be more readily found in a specific plate tectonic setting such as that which has evolved as the result of collapse of back-arc and fore-arc basins. Within this setting, it is the presence of thrust faulting that is important for it has led to the juxtaposition of ultrabasic rocks (the source of the PGEs) with host metasediments. If there is a genetic relationship between the type of precious metal mineralization, which exhibits some of the unusual characteristics discussed earlier, and the plate tectonic setting, then this could have a significant impact on exploration for these metals.

Conclusion

It is important to point out that the comments and opinions in this article are intended to be viewed in the perspective of the constantly changing state of our knowledge of the natural universe.

Assaying—Controversy and Reality

By ..vin C. Johnson, Jr., Ph.D.

The purpose of this article is to review some of the features of the fire assay method of analyzing for precious metals and to discuss a few of the more recent misconceptions concerning ores that reportedly "do not fire assay for gold and silver content."

Fire assaying is perhaps the single most important technique used in the mineral industry to demonstrate the visible presence of precious metals in various geologic ores and concentrates. Depending on the quality of application, individual fire assaying results may range from less than qualitative to quantitative. It is unrealistic to give the assayer $15 and a complex refractory ore sample, and expect in return to promptly receive a quantitative extraction and recovery report on the precious metal content. In order to properly assay most ores, and especially to develop a usable fire assay procedure for a particular ore, extensive development work is often necessary. To state that some types of precious metal-bearing ores cannot be fire assayed is as misleading as it is to say that all precious metal-bearing ore can be fire assayed and those that do not respond to certain specified fire assaying procedures do not have any precious metal values. It might be better to state that all ores containing precious metals are capable of being quantitatively analyzed by fire assaying methods if: (1) the sample matrix has been properly prepared prior to analysis, and; (2) the fire assaying technique is properly performed.

In general, the process of fire assaying for precious metal content may be divided into four stages, part of which may not be necessary depending on the desired results. Matrix preparation (Stage 1) is of primary importance in all analytical procedures. Such steps as comminution, oxidation of sulfides and sulfo-salts, distillation of mercury, and the leaching of excessive amounts of base metals from ore samples prior to fire assaying can make the difference between success and failure. The second stage involves the fusion of the prepared ore sample. In this step it is necessary that the flux formulation be correct to insure the proper fusion chemistry. Other variables such as the temperature of fusion, the type of furnace used, and the length of fusion time are of great importance. In fire assaying, litharge is perhaps the most commonly used precious metal collector; however, copper, nickel, and tin, or compounds thereof, are also used for special applications. In certain situations where relatively large amounts of gold and platinum-group elements are suspected to be present, silver is used as an inquarting media, usually in conjunction with litharge. Depending on the reference, from 10 to 20 times the suspected amount of gold plus platinum-group elements is suggested for the amount of silver inquart. This inquart may be added either to the fusion or at the cupellation stage. When the fusion is poured, generally into a cast iron mold, the button of collector-metal is recovered.

Stage 3 in the general procedure of fire assaying is referred to as scorification and this step applies mainly to lead buttons. Scorification is applicable if it is deemed necessary to remove contaminants that were collected by the lead in the fusion step which could seriously interfere with the recovery of the precious metals in the cupellation process. This step is too often omitted in the fire assay procedure, sometimes causing inaccurate results. Scorification of the assay button would tend to prevent much of the interference by many of the transition elements in the recovery by cupellation of precious metals in dore form. At this stage the assay button, which has collected the various metals from the ore sample, may either be analyzed gravimetrically or spectrographically, or the fire assay procedure may be continued through cupellation.

The final stage of fire assaying is cupellation. The prill resulting from cupellation may either be parted with nitric acid and the gold and silver contents of the prill calculated, or the prill may be analyzed by gravimetric or machine methods. In the latter situation the content of platinum-group elements may be calculated.

In reviewing some of the basic problems in fire assaying a particular ore sample for its precious metal content, it seems reasonable that without a prior knowledge of the ore and its contents, on a one or two time trial basis the results of several different assayers will not necessarily coincide. Fire assayers tend to be artisans in their own right. Many of them tend to specialize in ores that are within their own geologic district. In order to replicate their data the procedures that they used to prepare and assay a certain ore must be known.

In comparison, machine analysis by such methods as spectroscopy, X-ray, and neutron activation analysis has problems that are not entirely dissimilar from that of fire assaying. In this situation the analyst is frequently requested to analyze ore samples that contain an abundance of elements that drastically interfere with sample matrix preparation. These elements might also interfere with the detection and quantitizing of many, if not all, of the precious elements. There are operators that have, over a period of years, developed certain procedures for the accurate analysis of various types of ores. Without using the same or similar analytical procedures, it should not be assumed that two different analysts will arrive at the same, or closely similar, analytical results, especially where the precious elements are concerned.

Because of the problems inherent in analyzing certain types of ores for precious metal content, many mine developers have attributed the problem of conflicting results to such possible phenomena as micron-size gold, organic precious metal-bearing complexes, volatization of gold halides, and other such possibilities that would prevent a proper analysis of their ore. In many cases these operators have been able to perform cursory assays and recover enough precious metal content to convince them that the values are present, but for various reasons the results are not easily repeatable by either themselves or by third-party analysts. This could mean that significant values are, in fact, present, but for some reason an analytical procedure that will repeatedly detect these values has not yet been developed for this particular ore body.

A case in point is the Carlin disseminated gold deposit which is located in Nevada. Because of the lack of an appropriate analytical method, this well-known gold deposit was overlooked for many years. It was some time after the deposit was being exploited that a fire assay procedure was developed which actually reflected the gold values present. This procedure was utilized for further exploration and exploitation of this area. Such foresight should be acclaimed, not condemned just because this ore deposit could not be located with "standard" fire assay procedures as they then existed.

The presence of platinum-group elements in an ore sample can present the fire assayer with serious interference problems if they are not considered as the assay is taken through the cupellation stage. If the sample contains considerably larger amounts of platinum-group elements as compared to silver and/or gold content and, in turn, the sample is fire assayed through the cupellation process without the addition of sufficient inquarting media, the collection of total precious metal content from the sample to the prill will be appreciably diminished. If the presence of significant platinum-group elements is suspected in a sample, most professional fire assayers suggest the addition of from 10 to 20 times silver (or perhaps gold) more than the total amount of predicted PGE. If the above phenomena is in doubt, it can only be suggested that a blank sample be spiked with a precious metal content that consists of relatively large amounts of platinum-group elements and relatively small amounts of silver and gold. Perform a standard fire assay analysis (with no addition of inquarting media) through the cupellation stage, and weigh the results and calculate the recovery. The simple fact is, certain combinations of precious metals will not form dore alloys under the normal temperature conditions of cupellation. The myth becomes reality!

It is unrealistic to give the assayer $15 and a complex refractory ore sample, and expect in return to promptly receive a quantitative extraction and recovery report on the precious metal content.

An interesting example of the perils that are frequently encountered when utilizing a "canned" gold extraction flowsheet was recently observed by this writer. The ore being evaluated was a volcanic scoria and the precious metal values were largely present in the form of a metallic-like coating on the vesicle surfaces. Spectrographic analysis of this material indicated the presence of precious element content as well as relatively large amounts of As, Hg, Ti, Se, Te, Bi, Sb, Fe, Zn, Cu, Pb, Cr, Co, Mn, and Ni. Sulfur was determined by other methods to be present as well, but not in significant quantities. Since "standard" fire assay techniques did not reflect the precious metal content in this particular ore, a modified fire assay procedure was developed that enabled the use of fire assaying methods in controlling the extraction study of this ore. The initial extraction procedures that were performed were "conventional" and very similar to those that are proposed elsewhere. The testing procedures consisted of amalgamation, cyanidation, and sulfide flotation. In considering the composition of the precious metal-bearing component of this ore it is not surprising that none of the above three extraction procedures proved even partially successful. If just these conventional "canned" procedures for analysis and evaluation had been used on this ore it is likely that it would have been rejected as being potentially economic.

In conclusion, it should be stated that this article is not a condemnation of any particular method of analysis or ore evaluation procedure. Far from it! This presentation is partially to remind the mineral industry that ore analysts are not machines that automatically provide the same data from the same samples. A great deal of training and experience is required, as well as an open mind, for both mineral analysts and prospective clients. Systems change and new procedures and modifications evolve; otherwise, how can we as an industry manage to survive?

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INTRODUCTION

While reading this bulletin, we hope you will keep in mind that any reference work comprises the best knowledge an author has at the moment and may at some point in the future be considered in error, or in need of revision.

Minerals of Gold contains bits of information drawn from several sources and presented in a unified manner.

In the interest of brevity, we have been forced to use some technical terms with which the reader may not be familiar. However, we have placed the Glossary of Terms at the beginning of the text and repeatedly reinforced the meanings of these terms throughout the text.

Were you to look up any one of these minerals in a standard reference book, you would find them listed as "rare." Please keep in mind that such statements are based upon the entire lithosphere (e.g., the entire surface of the planet).

When auriferous (gold bearing) deposits are taken as the basis of comparison, they are by no means rare. In fact, they are quite common. To some degree or other, they are found in every deposit, including placer.

It is unfortunate that this literal interpretation of the term "rare" has tainted the conceptual understanding of most assayers and metallurgists. As a result, no consideration has been given to the unusual chemical and pyrometallurgical problems encountered when attempting to analyze and/or recover gold from these minerals.

We hope this bulletin will contribute to a better understanding of these problems by the layman and the professional alike.

IN BRIEF

Preceeding the name and chemical composition of each mineral is a series of drawings depicting its crystal configuration (if known). There will be a heavy bar drawn beneath the most common form in which that mineral will be encountered.

If some unusual configuration of the basic forms are involved, it will be included.

Following the name and chemical formula will be a verbal description of the mineral, and where the experts disagree, we have noted the fact and given both versions.

None of these minerals are formed directly from (or within) molten magma. Therefore, they are the product of water borne elements. At great depth and under tremendous pressure, these elements will remain in solution. However, as the temperature and/or pressures diminish, the solution becomes supersaturated and various minerals are formed and precipitate.

This is not a haphazard phenomenon, and depending upon the elements present, it follows a rather orderly procedure.

The proper term for this phenomenon is "paragenetic sequence" which we have kept quite brief and entitled Paragenetics.

Following paragenetics will be Chemical Behavior which is intended to warn the reader of the problems involved when working with these minerals.

We have not attempted to resolve these problems, merely to make you aware of them. Hopefully, at some point in the future, we will be able to publish the method of analysis and recovery for each mineral or set of minerals.

The last page is dedicated to some test procedures which may be of help in identifying these minerals.

GLOSSARY OF TERMS

Antimonical -- containing the element of antimony.

Auriferous -- gold bearing.

Colloidal -- particles so small that they will remain suspended in water literally forever.

Epicenter -- (as used in this article) the point at which the molten magma has stopped and the aqueous solutions and gasses are emanating.

Epigene -- a geological process taking place at or near the earth's surface.

Epithermal (deposit) -- a deposit formed in rock at shallow depth and by low temperature hydrothermal solutions.

Hypogene -- mineralization created by ascending waters, as contrasted to supergene which is mineralization altered or created by descending waters.

Intrusive -- molten rock (magma) which has solidified before reaching the surface.

Paragenetic -- referring to the chronological order of the crystallization of minerals within a vein.

COMMENTS ON CRYSTALLOGRAPHY

For gold to form a crystal structure with other elements, which is repeated and identifiable as a mineral, their atomic radii must be close to that of gold. To simplify the description: one cannot build an orderly pile of basketballs and ping-pong balls.

The elements with which gold can chemically combine are: silver, tellurium, antimony, bismuth, copper and the platinum group metals.

However, the platinoids usually form solid solutions (entrapment of one metal within the matrix of another).

Cleavage is the term used where a crystal breaks easily along a plane, leaving a flat surface.

Fracture is where a crystal has broken across its natural planes rather than with these planes.

Hardness is useful only when a crystal is large enough to manipulate (which is seldom).

The Mohs scale of hardness is: 1) talc; 2) gypsum; 3) calcite; 4) fluorite; 5) apatite; 6) orthoclase; 7) quartz; 9) corundum; and 10) diamond.

Density is a mineral's mass when compared to an equal volume of water.

CHEMICAL SYMBOLS
USED IN THIS BULLETIN

Element	Symbol
Antimony	Sb
Bismuth	Bi
Copper	Cu
Gold	Au
Lead	Pb
Selenium	Se
Silver	Ag
Sulfur	S
Tellurium	Te

Symbolic Terminology (by example)

Au	means one (1) atom of gold
Au_2	means two (2) atoms of gold
AuAg	means one (1) atom of gold combined with one (1) atom of silver
Au,Ag	means gold and/or silver without a fixed ratio
(Au,Ag)	means either gold or silver but in a fixed ratio [e.g., $(Au,Ag)_2$ would mean that either two atoms of gold OR two atoms of silver comprise this portion of the mineral being described].

No identifiable crystal habit (probably isometric)

PETZITE
Ag_3AuTe_2

Description:
Petzite is steel grey to iron black and quite brittle.

Even though this mineral has a hardness of 2.5 (between gypsum and calcite), it is sectile, and this ability to be carved with a knife without crumbling is probably the only clue to its identity in the field.

A density of 9.13 (when pure) allows it to collect well in gravity concentration. Cleavage is described as fair, and the fracture is subchoncoidal.

Paragenetics:
Petzite is found in intrusive and subvolcanic formations where the magma never achieved the surface and hydrothermal activity was absent or nil.

Chemical Behavior:
This mineral is insoluble in alkali cyanides, hydrochloric acid, aqua regia, and potassium hydroxide. However, it is decomposed by nitric acid, both dilute and concentrated. Moreover, the reaction progresses with effervescence.

Ferric chloride will pit the surface, but we do not know whether or not a ferric chloride leach will succeed. (This needs to be explored.)

#

Poorly defined, cleavable, grains

MUTHMANNITE

(Ag,Au)Te

Description:

Muthmannite has never been proven as a legitimate mineral species, but it is often listed as such.

It is described as elongated grains of brassy yellow to grey white metallic substance which has perfect cleavage upon one plane.

The density is 5.59 which is quite light for gold bearing minerals, and the hardness is 2.5 which lies between the hardness of gypsum and calcite.

Paragenetics:

Unknown.

Chemical Behavior:

As the paranthetic (Ag,Au) implies, the gold and/or silver content may vary from molecule to molecule in any sequence, e.g., $(AuTe)_x (AgTe)_y$.

It is decomposed by dilute nitric acid, leaving the gold as a true submicroscopic colloid suspended in solution.

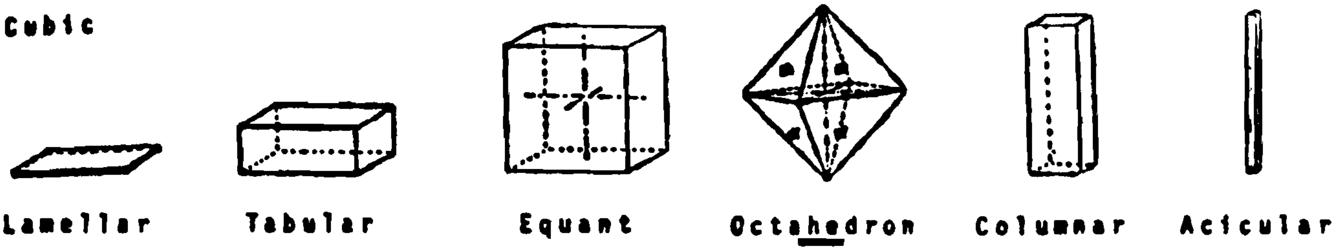
After leaching, muthmannite is soluble in aqua regia if silver predominates. However, as this ratio may change from one grain to the next, there is no assurance that recovery will be complete when using any known leach solvent.

PLATINUM GOLD

(AuPt)

A combination of gold and platinum.

No authoritative published comments.



PORPEZITE -- PROPEZITE

(Au,Pd)

Description:

We are uncertain as to the proper spelling, for one author describes pallaidic-gold (porpezite) as face centered, cubic (octahedrons), often distorted in shape, with a hardness of 2.5 and a density of 19.3, and the second author agrees so far, and both state that the palladium varies from 5% to 10%.

However, the first author states that the color is golden yellow with a metallic luster and sometimes a fine brown powder. While the second author declares that it is metallic and cream-colored with a pinkish tinge.

Personal Note: As an artist who has worked with precious metals in the field of jewelry, might I add: "They are both correct."

Five to eight percent Pd content fits the first description and 8% to 9% the second. By 10%, the color is that of white gold used in jewelry.

Chemical Behavior:

Soluble in aqua regia, alkali cyanide, and sometimes in ferric chloride/acid/salt leaches.

#

RHODITE

(Au,Rh)

A combination of gold and rhodium (one of the platinum group metals).

No authoritative published comments.

#

No known crystal description

FISCHESERITE

Au_3AuSe_2

We have very limited information on this mineral. It is not a true selenide of gold, for the selenium is combined with the silver, and the silver is combined with the gold.

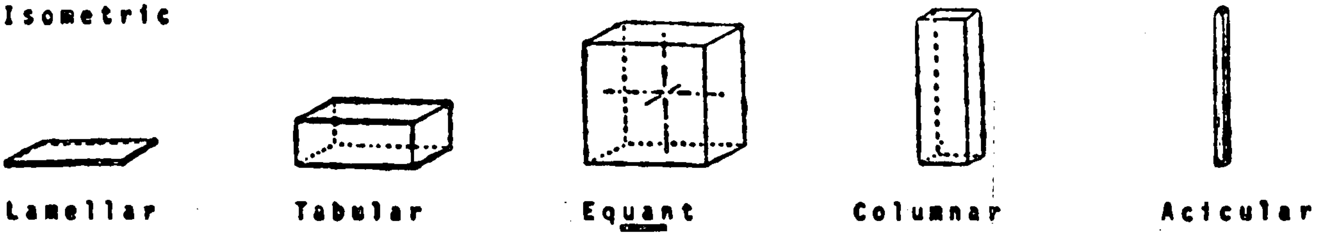
The only comments we found on the deposition of this mineral was the fact that when selenium and gold are found together, tellurium is usually absent.

Generally speaking, the higher the selenium content, the higher the gold content.

Hypothermal deposition occurs earlier than the sulfides, for selenium is less mobile than sulfur.

As the chemistry of selenium with four common valence states is rather complicated, we will not pursue this discussion until more information is available.

Isometric



IRIDIC GOLD--AUROSMIRIDIUM

Iridic gold is iridium (up to 20%) trapped in solid solution in gold.

Aurosmiridium is the reverse, where gold, and/or osmium, is entrapped in a solid solution of iridium.

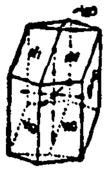
Sorry, we found no other published information on this subject other than the fact that it is soluble in sodium peroxide fusion only.

#

Monoclinic

Lamellar

Tabular



Equant

Columnar

Acicular

NAGYAGITE

$Pb_5Au(Te,Sb)Sb_4S_{5-8}$

Description:

The name is a mouthful, the chemical composition complex, and, to add insult to injury, it is not a true sulfide, but rather an intermetallic compound.

To anyone short of an expert, these monoclinic crystals appear tetragonal. They are usually thin, tabular, and often bent. To find this mineral in a granular form would be unusual.

Its color ranges from greyish white to blackish lead grey with a metallic luster.

Cleavage is perfect, and the platelets are flexible. Its hardness is 1.5 (a little softer than gypsum), and it's density ranges from 7.3 to 7.5.

However, the thin platelets often have a tendency to float by surface tension.

Paragenetics:

Nagyagite is formed in epithermal mineral veins in the company of: native gold, electrum, altaite, native tellurium, pyrite, and alabandite.

As an epigene is, by definition, formed at, or near the surface, one would assume that nagyagite would give way to other minerals with depth.

Chemical Behavior:

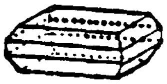
Nagyagite, like most minerals of gold, can be decomposed by dilute nitric acid, which leaves the gold as metal in colloidal solution.

This mineral is insoluble in alkali cyanides, alkali hydroxides, ferric chloride, hydrochloric acid, and aqua regia.

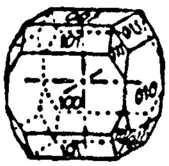
#

Tetragonal

Lamellar



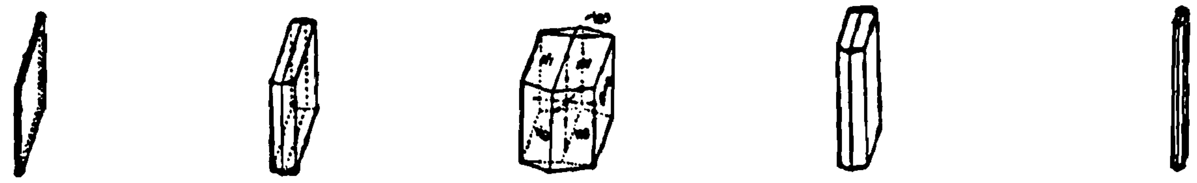
Tabular



Equant

Columnar

Acicular



Monoclinic

Lamellar Tabular Equant Columnar Acicular

NAGYAGITE

$Pb_5Au(Te,Sb)Sb_4S_{5-8}$

Description:
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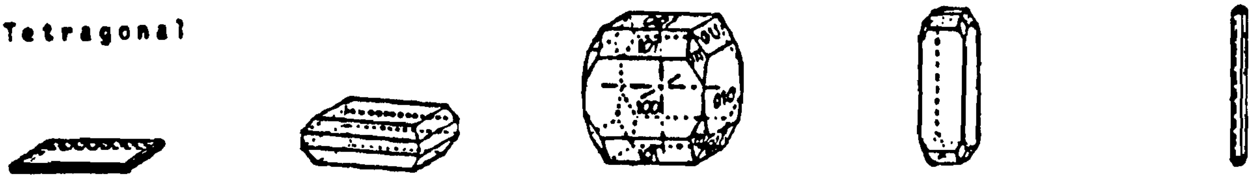
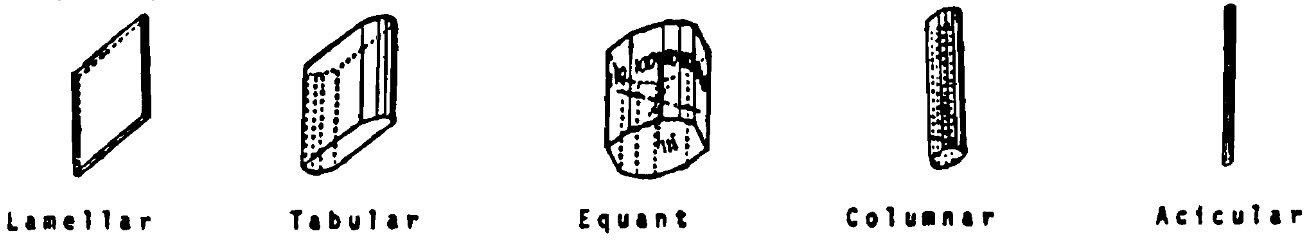
#



Tetragonal

Lamellar Tabular Equant Columnar Acicular

Triclinic



MONTBRAYITE

$(Au,Sb)Te_2$ --?-- Au_2Te_3

Description:
Everyone agrees that montbrayite is yellowish white with a metallic luster, but there are disagreements as to the chemistry of the mineral.

Some reference sources state that an isomorphic replacement of gold and antimony combine with two atoms of tellurium, and others insist that two gold combine with three tellurium, and montbrayite is the resultant mineral.

Personal Note: If the experts cannot agree, is there any wonder that prospectors and assayers often argue over the names and qualities of minerals?

All parties agree that the crystals called montbrayite have good cleavage on all three planes of a triclinic crystal and that the angles at which the axis of these planes meet are 104°30', 97°34', and 107°53' respectively.

Their hardness is 2.5. They are brittle and have a density of 9.94 (when pure).

Personal Note: Someday we will ask an expert how the density of both possible combinations can be so exact. We find this difficult to justify.

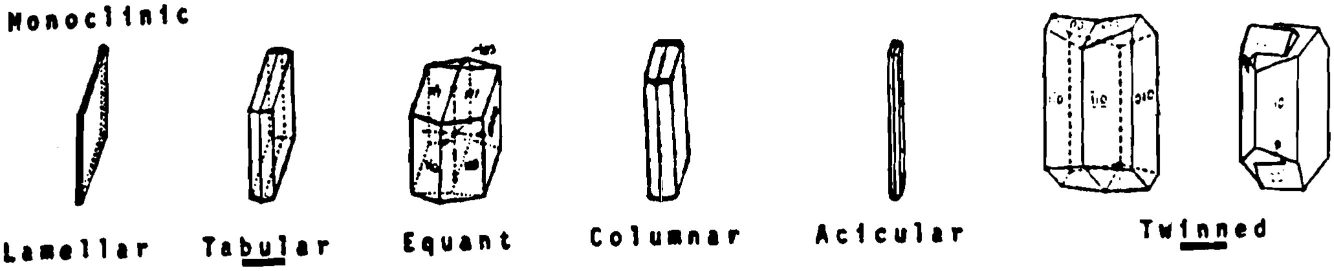
Paragenetics:
We found no specific comments as to how and when montbrayite is deposited. However, we assume the process to be hydrothermal and below 370°C.

This is based upon the fact that gold does not lose its mobility in aqueous solutions until the temperature is below 370°C, and montbrayite is found in the company of: chalcopyrite, frohbergite, "tellurbismuth," melonite and colorodite.

Chemical Behavior:
Montbrayite is attacked only by dilute nitric acid (with effervescence). If the mineral is rinsed shortly after immersion and examined, it will have etched in an alligator pattern.

It appears to be a leach operator's nightmare, for it must be decomposed by dilute nitric acid before any other chemical will attack it.

#



SYLVANITE
$(Au,Ag)Te_4$

Description:

Once again, the experts cannot agree. In writings prior to 1966, the chemical composition was often given as $(Au,Ag)Te_2$. However, more recent works agree that four tellurium to one gold or silver is the correct ratio.

Gold and silver are isomorphic and, as such, one can substitute for the other in any imaginable quantity.

This isomorphic substitution probably explains the difference in color description one encounters when reading various reference works.

There are 3 primary descriptions used. One is: yellowish grey to silver white with a metallic luster. The second is: high silver white metallic with a hint of yellow. The third is: steel grey to silver white metallic.

In any event, the crystal form is a better point of identification, for it is monoclinic and usually tabular or twinned... sometimes granular. When it occurs in the tabular form, it is usually dendritic (like a bush or tree within the host) and/or graphitic.

Sylvanite is always brittle with an uneven fracture but perfect cleavage. The perfect cleavage between the lamellar platelets is reason for the graphitic behavior, and like graphite, it will float upon the surface of water effortlessly.

A hardness of 1.5 on the Mohls scale is difficult to observe due to the brittle quality of this mineral, and its graphitic behavior disguises the density, which is 8.16 times that of water.

Paragenetics:

Sylvanite is the most common telluride gold ore. It occurs in low temperature (less than 200° Centigrade) hydrothermal intrusive veins and subvolcanic hydrothermal deposits, often in considerable quantities.

Chemical Behavior:

Sylvanite is a leach operator's nightmare, for it is insoluble in alkali cyanides, alkali hydroxides, hydrochloric acid, and ferroc chloride.

It has very poor solubility in aqua regia, but the tellurium will dissolve in 12% to 15% (by volume) reagent grade nitric acid, leaving the submicroscopic gold in suspension in the solution.

These colloidal particles are true metallic with a valence of zero. However, they are soluble in any nitro-chloric acid (a combination of nitric acid and hydrochloric acid where the nitric acid is less than 1:3 with the hydrochloric, and water comprises at least 50% of the total solution).

#

(Calaverite continued from Page 8)

When attempting to use cyanide as a solvent, one must take into consideration that the tellurium portion of this compound may be etched only by potassium hydroxide, rather than the less expensive calcium and/or sodium hydroxides which are customarily used. However, the potassium hydroxide must have time to attack the tellurium before the alkali cyanide is added.

When the accompanying minerals do not interfere, calaverite will leach with a combination of sodium chloride (salt) (5%), aqueous ferric chloride (2.5%), and hydrochloric acid (1.2%), the balance (91.3%) being water.

If there are enough requests, ASAT will publish a bulletin on this last method of leaching and the recovery from the leach liquor.

Crystal Habit Unknown

KOSTOVITE

$AuCuTe_4$

Description:

If you have gold in a copper ore along with tellurium and have failed to identify the source of the gold, this mineral may be the culprit.

Kostovite has no distinct metallic luster and varies from brown to reddish grey, to greyish yellow in color.

It is also devoid of any identifiable crystal structure. To date, no one has specified either a hardness or a density for this mineral.

Paragenetics:

Unknown.

Chemical Behavior:

Kostovite resists attack from all standard wet chemical reagents and may be dissolved only by using perchlorates and periodates in combination with acids.

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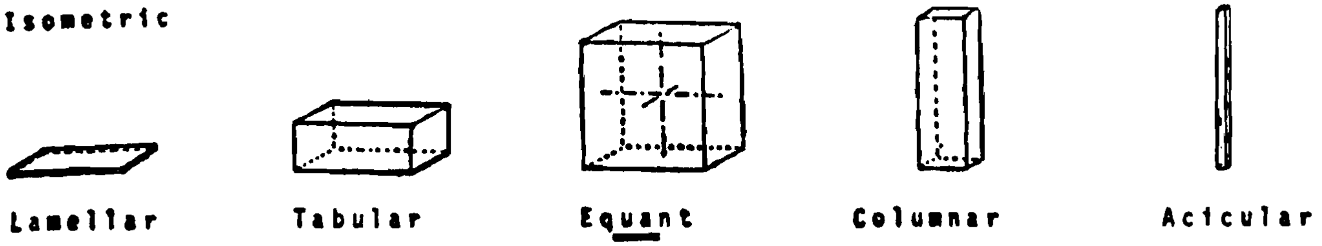


MALDONITE

Au_2Bi

Description:

In its unaltered state, maldonite is isometric (cubic) crystal which is silver white metallic with a pinkish tinge. However, it soon tarnishes to red and/or black.

It is sectile (can be carved with a knife without breaking) and has distinct cleavage.

Usually, it has undergone some degree of alteration and is a homogenous compound of gold and bismuth which is difficult to identify.

The hardness varies from 1.5 to 2, which means that it may be scarred by gypsum but will not scratch gypsum. Before decomposition, maldonite's density is 15.46 (64.50% Au and 35.72% Bi), but the density increases as the bismuth is leached from the crystal.

Using a blowpipe and charcoal block to fuse maldonite will produce a metallic gold bead and a bismuth oxide coating on the coal. Fusibility is very low. In fact, the flame of a candle is sufficient.

Bismuthic Gold is a term often used to describe the skeletal remains of maldonite after the bismuth has been thoroughly leached (about 3% Bi remains).

Paragenetics:

As the ascending mineral laden solutions form the hypogene and their temperatures begin to drop, maldonite is formed at 372°C or slightly less. It is noteworthy that this is the highest temperatures at which gold is deposited in gold quartz veins.

It may be accompanied by pyrite, marcasite, sphallerite, enorgite, goldfieldite, arsenical famatinite, and tennantite.

It is also found in garnetiferous skarns in the company of a host of tellurides and selenides, including selenopyrites.

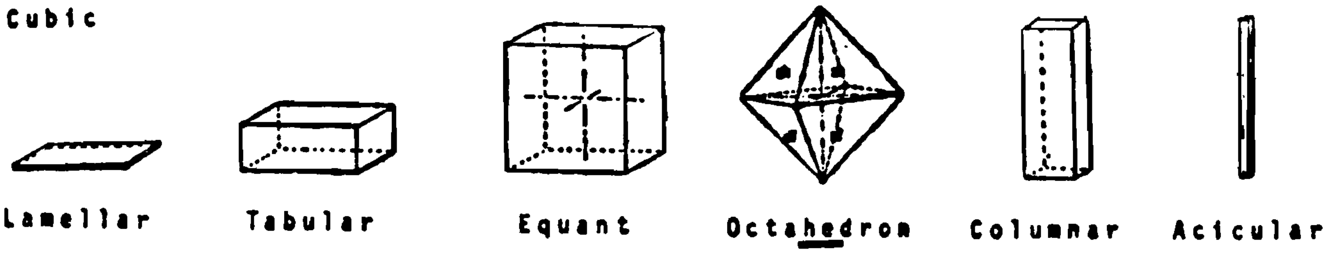
Minerologists have reported its presence in loellingite.

Chemical Behavior:

Maldonite is stable between pH 4 and 8. When these limits are exceeded, the bismuth portion is attacked slowly and with some difficulty. The mineral is insoluble in hydrochloric acid but will dissolve in aqua regia.

We found no confirmed information on the alkaline solubilities nor on the reaction to cyanide. The latter should, however, be negative.

#



CUPROAURIDE ---------- *AURICUPRIDE*

AuCu *$AuCu_3$*

Description:

These minerals are intermetallic compounds of gold and copper which seldom have fixed ratios of one metal to the other.

Cuproauride is the name used when the copper content is below 20%, and auricupride is theoretically 50.8% copper but usually contains at least 60% copper.

When pure and crystalline, they assume the isometric, face centered cube (octahedron); however, they often form globules, leaflets, platelets, and spangles which are mistaken for tarnished native copper.

Color is difficult to describe as it varies from golden yellow with a reddish tint to copper red with a golden tint. Both have a high metallic luster and are malleable.

Hardness varies somewhat with the copper content but averages 3.5 on the Mohls scale (fluorspar will scratch it, and calcite will not).

With a density of approximately 11.5, both minerals concentrate well, unless the particle size is smaller than 120 microns and/or the shape is thin platelets.

Both cuproauride and auricupride have a peculiar ability to resist wetting which interferes with hydrometallurgy and amalgamation.

They are probably the most overlooked minerals of gold.

Paragenetics:

The dispositional possibilities of these minerals are too diverse to describe in this brief bulletin. Let it suffice to say, that if you have copper in any form, they may be present. The only pathfinders are the presence of chromite and niccolite which often accompany these minerals.

Chemical Behavior:

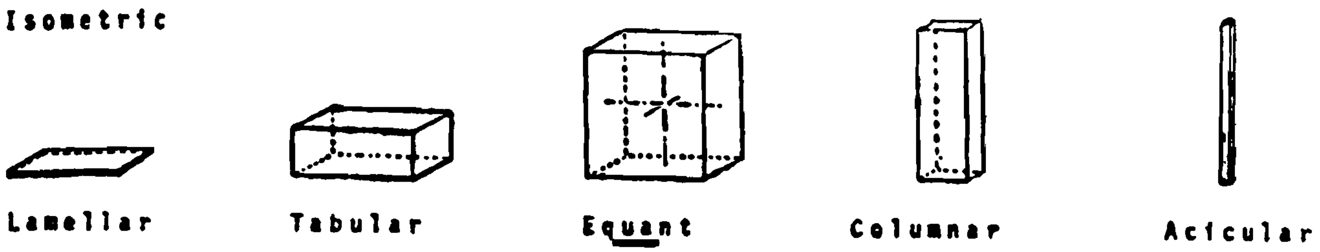
If the total copper content of the ore is fractions of 1 percent, fusion assay is quite successful.

However, the higher the copper content in the ore, the more difficult the fusion becomes, and it soon reaches the point where scorrification is necessary.

Theoretically, both elements, gold and copper, will respond to numerous chemical solvents.

However, the unique ability of cuproauride and auricupride to resist wetting defeats both leaching and analysis.

Moreover, this phenomenon is used to identify these minerals in microchemical techniques, e.g., if the physical description fits and the surface of a polished section is not attacked by nitric acid, hydrochloric acid, aqua regia, ferric chloride, mercury chloride, potassium hydroxide, or potassium cyanide, the mineral is determined to be either cuproauride or auricupride as evaluated by color.



AUROSTIBITE

$AuSb_2$

Description:

This mineral of gold is often mistaken for galena, for its crystal habit is isometric and generally cubic, with the same metallic white luster as lead sulfide.

However, with a ratio of two antimony to one gold, the crystals will soon tarnish like boronite (peacock copper), and this behavior is aurostibite's identifying characteristic.

It is a rather dense mineral with a specific gravity of 9.91 when pure and usually concentrates well. The hardness by Mohls scale is 3 which can be scratched by fluorspar but not by calcite.

Paragenetics:

Aurostibite forms late in the hypogene process as the ascending, mineral laden waters cool. The top end of this series would be in the low temperature quartz stibnite veins where it would associate with stibnite, berthierite, native antimony, and arsenopyrites.

As the hydrothermal waters cool below 150°C, aurostibite is found in the company of stibite, tetrahedrite-tennanite, and hidden in pyrite, arsenopyrite, and marcasite.

Generally speaking, as the sulfide content increases (and the antimony decreases), the gold content decreases as well.

--

During the supergene process where descending waters dissolve and transport mineralization, the antimonical constituents are attacked if the pH drops below 3 or exceeds 8 leaving behind a dull rusty, cellular textured gold which is reluctant to amalgamate.

Aurostibite is usually devoid of silver. However, when it does occur in tertiary, polymetallic veins with ruby silver, it is often of bonanza proportions. It should be noted that this mineral has never been found in skarn.

Chemical Behavior:

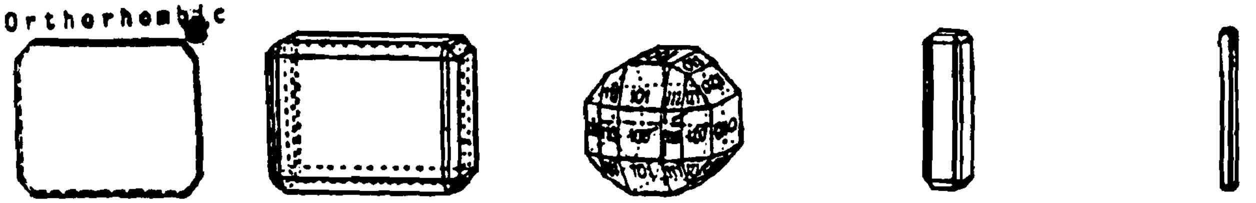
The deceptive appearance of aurostibite mimicking galena often leads to errors in fire assaying, for sulfur is a reducing agent and antimony is not.

This mineral is notorious for defeating cyanide leaching, for it is totally insoluble in alkaline cyanides.

However, the antimony may be removed by leaching with dilute potassium hydroxide or 12% to 15% (reagent grade) nitric acid.

Decomposition may be accomplished with hydrochloric acid and/or ferric chloride. However, in this instance, if (and only if) powerful oxidants are present in the ore, gold will go into solution as well.

Assuming the gold to have remained in the residue, the particle size will average less than 80 microns (will pass through a #200 screen).



KRENNERITE

$AuTe_2$

(with Ag substituting for Au up to 1 part in 4)

Description:

Once again, there is argument as to the color of a mineral with one authority giving silver white to brass yellow and another describing it as silver white with a yellow tinge. However, they all agree that krennerite has a very high metallic luster and good crystal faces.

These crystals are usually coarse grained and orthorhombic. They are very brittle and have good cleavage, with subchoncoidal fracture.

Hardness is 2.5 by the Mohls scale which lies between gypsum and calcite, and their density is 8.6 times that of water.

Paragenetics:

Krennerite is rather unstable in the chemical climate of hydrothermal deposition and is often transitory.

Gold and silica are the most mobile elements in mineralogical chemistry, e.g., they will travel further from the epithermal center (original source of magmatic heat).

In the presence of tellurium, this mobility is reduced for gold, and deposition is closer to the heat source. When silver is involved, as it is in krennerite, the distance from the epicenter is shorter yet, which means that krennerite is the first formed of the late forming minerals in the hydrothermal deposition process.

Among the early forming (less mobile minerals) krennerite may replace nagyagite and/or form as intergrowth in silver rich fahlore and pyrrhotite.

In turn, the hydrothermal solutions migrating through the partially formed rock may redissolve krennerite and replace it with native gold, hessite, petzite, or chalcopyrite.

Chemical Behavior:

With silver present, this mineral is attacked by nitric acid in any proportion exceeding 12% and decomposes with effervescence. However, the molecular gold is left behind in suspension in the solution, and hydrochloric acid must be added to bring the chemistry to that of aqua regia or, better yet, chloro-nitric acid (approximately 1 nitric, 4 hydrochloric, and 5 water).

Krennerite is etched by ferric chloride and potassium hydroxide which implies that unless other minerals interfere you may be able to use the saltwater leach discussed under calaverite.

This mineral is not directly attacked by cyanide but will often dissolve in this medium if the particle size is small and a potassium hydroxide solution is used to attack the mineral prior to the introduction of alkali cyanides.

Crystal habit unknown

ELECTRUM -- KUSTELITE
(Ag,Au)

There exists a complete interlattice substitution series of gold and silver.

If the silver content is 30% to 45%, it is called electrum. If the silver content is 80%, the name changes to kustelite.

--

In nature, gold and silver are found combined in the above combinations and one other -- 1% gold 99% silver -- which is the normal silver ore combination.

The only other comments are that it is metallic and malleable with a wide range of colors.

Crystal habit unknown

UYTENBOGAARDITE
Ag_2AuS

No published comments save the fact that this is <u>not</u> a gold sulfide, but rather a bonding of sulfur to silver and silver to gold.

#

Crystal habit unknown

PALLADIUM—AUROCUPRIDE
$(Cu,Pd)_2Au_2$

No authoritative published comments.

TORCH TESTS ON CHARCOAL

Dig a small concavity in a charcoal block (or briquet), and fill it with a few grains of the mineral in question.

Using the oxidizing tip of the flame from a propane torch, fuse the mineral to bright red heat. (The oxidizing portion of a gas flame is the near invisible tip where orange flickers sometimes appear -- the deep blue inner cone is the reducing part of the flame).

The minerals which will give metallic beads and/or identifiable residue are:

Calaverite -- leaves a gold button and colors the flame green

Krennerite -- leaves a gold button

Maldonite -- leaves a gold button and coats the charcoal with yellow bismuth oxide

Muthmannite -- leaves a metallic globule and a white sublimate

Nagyagite -- residue of gold and two coatings on the charcoal: one yellow and permanent, the other white and volatile in the oxidizing flame

Petzite -- leaves a metallic globule

Sylvanite -- leaves a metallic globule and a white sublimate

A COMMENT ON SPECTROCHEMICAL ANALYSIS

Spectrographic sensitivity for the element tellurium is 0.5% (by weight) which would amount to 10 pounds per ton of ore and concentrates. Therefore, it only has validity when the tellurides of gold exceed 20% of the ore or concentrate.

If tellurides are suspected, the best determination would be a wet chemical tellurium assay, using sulfurous (anhydrous sulfuric) acid.

GOLD MINERALS NOT SOLUBLE IN AQUA REGIA
THEREFORE
NOT TESTABLE BY STANDARD WET CHEMICAL, AA, ICP, OR DCP.

1. PELZITE Ag3 Au Te2. DESCRIPTION: Steel grey to iron black & very brittle. Hardness 2.5 Density 9.13

2. MUTHMANNITE (Ag, Au) Te. DESCRIPTION: Brassy yellow to grey white. Hardness 2.5 Density 5.59

3. RHODITE (Au, Rh). DESCRIPTION: Dark brown to black. Hardness 3.0 Density 7.70

4. FISCHESSERITE Au3 Au Se2. DESCRIPTION: Grey white to reddish black. Hardness 2.[illegible] Density 8.18

5. AUROSMIRIDUM (Au, Os) Ir2. DESCRIPTION: Rust to black. Hardness 2.7 Density 7.07

6. NAGYAGITE Pb5 Au (Te, Sb) Sb4 S5-8. DESCRIPTION. Metallic grey white to blackish lead. Hardness 1.5 Density 7.3

7. MONTBRAYITE (Au, Sb) Te2. DESCRIPTION: Metallic yellowish white. Hardness 2.5 Density 9.94

8. SYLVANITE (Au, Ag) Te4 (Very poor solubility). DESCRIPTION: Granular metallic steel grey to silver white. Hardness: 1.5 Density [illegible], but floats like graphite.

9. CALAVERITE Au Te2. DESCRIPTION: Metallic silver white to brassy yellow. Hardness 2.5 Density 9.3

10. KASTOVITE Au Cu Te4. DESCRIPTION: Brown to reddish grey to greyish yellow. Hardness 2.5 Density 7.61

11. CUPROAURIDE Au Cu. DESCRIPTION: Golden yellow with reddish tint. Hardness 3.5 Density 11.5

12. AURICUPRIDE Au Cu3. DESCRIPTION: Copper red with golden tint. Hardness 3.5 Density 11.5

13. KRENNERITE Au Te2. DESCRIPTION: Silver white to light brass yellow. Hardness 2.5 Density 8.6

14. AUROSTIBITE Au Sb2. Soluble only with high oxides present in the ore. DESCRIPTION: Metallic white luster mistaken for lead galena. Hardness 3.0 Density 9.91

15. UYTENBOGAARDITE Ag2 Au S. (Barely soluble). DESCRIPTION: Metallic grey to dark brown. Hardness 2.5 Density 6.61

16. AUROCUPRIDE (Cu, Pd)2 Au2. DESCRIPTION: Rusty to metallic black with pink tint. Hardness 2.19 Density 8.91

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NOTE

Determination of trace gold in rock samples by a combination of two-stage solvent extraction and graphite furnace atomic absorption spectrometry: The problem of iron interference and its solution

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(*Received May 29, 1995; Accepted January 16, 1996*)

For the accurate determination of trace gold in rocks and iron minerals by graphite furnace atomic absorption spectrometry, a two-stage solvent extraction method (diethyl ether and MIBK) was designed to prevent strong interference from iron and to effectively concentrate gold. The interference was perfectly condoled by adjusting some operating conditions in the solvent extraction. In the analytical method developed, the recovery of gold was >90%, and the precision (C.V.) was 11.7% (1 μg/l). When 0.5 ml of MIBK was used in the second extraction, the detection limit for gold was 0.13 μg/kg (concentration in rock) at S/N = 2. The gold concentration of geological standard rocks (JB-2 and JB-3) measured by the method proposed in this work was in close agreement with the recommended values.

The gold concentration of sandstone and shale collected from the Shimanto belt, southwestern Japan, was in the range 1.0–2.5 and 1.1–1.8 μg/kg, respectively, while the gold concentration of pyrite contained in the above rocks was about ten times higher than that of the corresponding rocks.

μg = micro gram = 1×10^{-6} g i.e. ≅ $\frac{1}{1,000,000}$ gram or ≅ ppm

INTRODUCTION

In the formation mechanisms of epithermal gold deposits, the origin of gold is one of the most attractive problems in geochemistry. According to one hypothesis (e.g., Richards *et al.*, 1991), its origin is the mother rocks (country rocks) around the gold deposits, that is, it results from continuous dissolution of gold from the mother rocks, transport and selective precipitation. To verify the idea, it is first important to estimate accurately the gold concentration in the mother rocks. Gold occurs in rocks at sub ng/kg-μg/kg levels. To estimate a reliable mean concentration of gold in rock samples, a combination of solvent extraction with MIBK and graphite furnace atomic absorption spectrometry (GFAAS) is convenient. Although Shimizu *et al.* (1984) reported an excellent determination method of gold in ore samples using the combination, the detection limit was 1 ng/g. It is desired to enhance the sensitivity about tenfold in order to determine trace gold in rock samples.

In a series of preliminary experiments, the authors often observed severe interference from iron during measurement using GFAAS with an ordinary one-stage MIBK solvent extraction. To eliminate the interference, Shimizu *et al.* (1984) used the back extraction of iron into a diluted acid solution. However, their method was investigated only for the Fe concentration of 0 to 10 mg/10 ml of MIBK. To determine gold not only in rock samples containing relatively high amounts of iron but also in iron minerals such as iron oxides and sulfides, which are candidates as host minerals for

ng = 10^{-9} g

$\frac{1}{,000,000,000}$ gram

gold, it is essential to examine in detail the iron interference for higher Fe concentration ranges (about several thousands of mg/l at maximum) and to develop a method to eliminate it.

In this study, we developed a two-stage solvent extraction method to eliminate the iron interference and applied it to the determination of trace gold (sub ng/kg to μg/kg) in rock and mineral samples using GFAAS.

EXPERIMENTAL METHODS

Instrumentation

The instrumentation consists of a Nippon Jarrell Ash Model AA-8200 atomic absorption spectrometer equipped with an FLA-1000 thermal atomizer, an HU-21 graphite furnace, an AS-301 autosampler and a Rika Deki R-032 recorder. A pyrolytic graphite-coated tube was used. The background correction was carried out using a D_2 lamp. The peak height was taken for the measurements.

Reagents

The standard solutions were prepared from a 1000 mg/l Au stock solution (Katayama Kagaku) by diluting it with 0.1 mol/l HBr solution. MIBK (methyl isobuthyl ketone) was that for atomic absorption spectrometry and other reagents were of analytical reagent grade. All the solutions were prepared with ultra-pure water.

Analytical method for gold

(1) Complete dissolution of rock samples After complete dissolution of rock sample (1 g) with a mixed acid ($HClO_4$ 1 ml + HF 10 ml) in a PFTE beaker on a hot plate, the solution is evaporated nearly to dryness at around 180°C. After cooling, the residue is dissolved with 4 ml of aqua regia. The solution is evaporated nearly to dryness around at 100°C and the residue is redissolved with 5 ml of a 5 mol/l HBr solution.

(2) Extraction of gold into diethyl ether The solution and the 6 ml of water used to wash the beaker are collected into a separatory funnel. 0.5 ml of H_3PO_4 is added, and the mixture is shaken with 20 ml of diethyl ether for 3 minutes. After the separation, the diethyl ether is removed by evaporation on a water bath.

(3) Extraction of gold into MIBK The resultant residue is moved into a glass tube by dissolving it with 3 ml of 1.5 mol/l HBr solution saturated with MIBK. This operation is repeated three times for the complete recovery and then the solution is shaken with 0.5 to 2 ml of MIBK. In order to collect the small amount of MIBK with a pipet, a slender glass tube was used.

(4) Measurement of gold by GFAAS Taking 0.02 ml of the MIBK layer, the atomic absorption signal for gold at 242.8 nm is measured. The blank and standard solutions for calibration are prepared in the same manner using the two-step solvent extraction described above.

RESULTS AND DISCUSSION

Effect of iron on the atomic absorption signal for gold

When 1 g of rock sample is dissolved, the concentration of iron in the final MIBK solution can reach several tens of thousands of mg/l. Additionally, there is a relatively strong absorption line of iron (242.4 nm) near the analytical line for gold (242.8 nm), suggesting that iron interferes strongly with the determination of gold. First, the effect of iron on the atomic absorption signal was examined using ferric chloride solution (0 to 3000 mg Fe/l) containing trace gold (5 μg/l). Figure 1 represents the atomic absorption signals at 242.8 nm, the analytical wavelength for gold. The operating conditions are shown in Table 1. The symbol (*) shows the peak for the determination of gold. The peak height increased with increasing iron concentration, indicating that the coexisting iron causes a positive error in the determination of gold.

Design of a two-stage solvent extraction for removal of iron and concentration of gold

As shown in Fig. 1, in order to determine trace gold in rock samples, it is necessary to remove iron from the sample solution and to concentrate gold effectively in the final MIBK solution. In

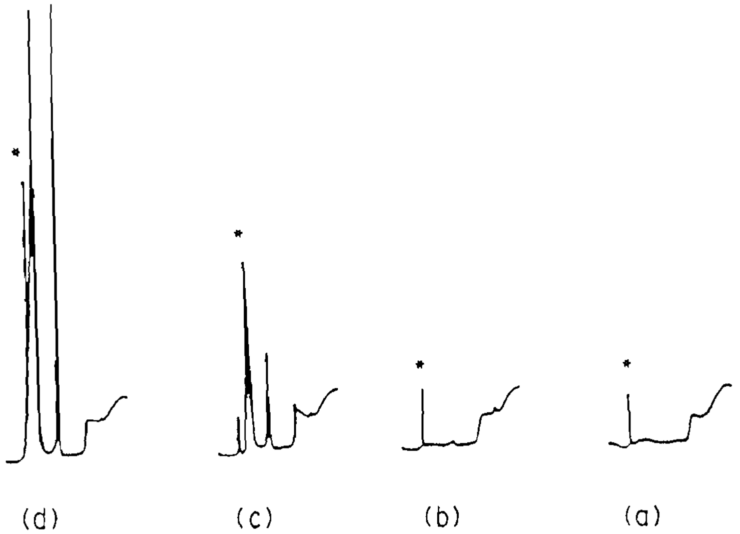


Fig. 1. Atomic absorption signals at 242.8 nm. (: Atomic absorption peak.) Au concentration: 5 μg/l. Fe concentration: (a) 0, (b) 1000, (c) 20000, (d) 30000 mg/l.*

Table 1. Operating conditions

Argon flow rate (dm^3/min)	: 2.0
Wavelength (nm)	: 242.8
Sample injection (cm^3)	: 0.02
Sensitivity	: 0.49
Drying (°C)	: 150
(Ramp, s)	: 30
(Hold, s)	: 40
Ashing I (°C)	: 600
(Ramp, s)	: 10
(Hold, s)	: 10
Ashing II (°C)	: 700
(Ramp, s)	: 3
(Hold, s)	: 13
Atomizing (°C)	: 2300
(Ramp, s)	: 0
(Hold, s)	: 5

the determination of gold by GFAAS, MIBK is one of the best organic solvent because of its high boiling point (136.1°C) and its insolubility in water. As a result, a two-stage solvent extraction using diethyl ether and MIBK was designed. In the first stage, iron remains in an aqueous layer 9 ml) due to complexation with phosphoric acid, while gold is effectively extracted into a large unt of diethyl ether (20 ml). The diethyl ether, because of its low boiling point (34.9°C), can be easily evaporated in a short time without the loss of the tetrabromogold complex that would result from heating to a high temperature. In the second stage, gold is concentrated into a small MIBK layer (0.5 to 2 ml).

Effect of HBr concentration on the extraction of gold into diethyl ether

The extraction of gold ($AuCl_4^-$ or $AuBr_4^-$) into MIBK greatly depends on the acid concentration. Brooks *et al.* (1981) reported that with an acid concentration (HCl or HBr) above 1 mol/l, the proportion of the extraction into MIBK is nearly 100%. Figure 2 shows the variation of the proportion of the extraction into diethyl ether with HBr concentration. Above 1.5 mol/l, the proportion was more than 90%. As a result, the HBr concentration in the solution A was determined to be 1.5 mol/l.

The permissible concentration of iron in aqueous solution before extraction of gold with MIBK

From the results shown in Fig. 1, it is concluded that in the solvent extraction with MIBK, the iron concentration in aqueous solution must

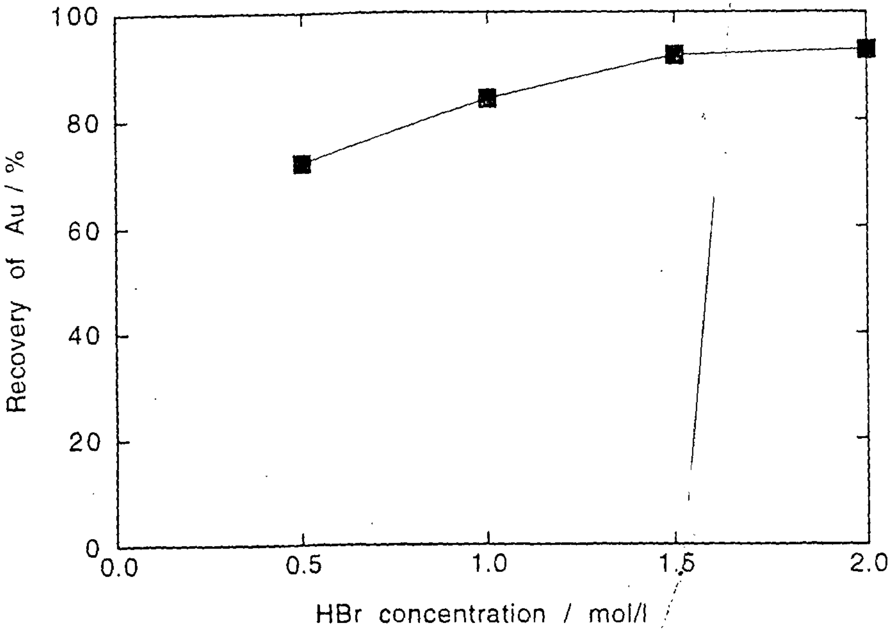


Fig. 2. Effect of HBr concentration on the recovery of gold.

be lowered below 500 mg Fe/l for safety. Accordingly, a large amount of iron must be removed in the stage of extraction with diethyl ether.

Behaviour of iron in the extraction with diethyl ether

The extraction of iron into diethyl ether is prevented by the presence of phosphoric acid as a complexing agent (I. Inamoto, personal communication). Using 9 ml of an iron(III) chloride solution (30000 mg Fe/l) with various HBr concentrations (0 to 2 mol/l), the behaviour of iron during the extraction with diethyl ether (20 ml) was examined in the absence and presence of phosphoric acid. As shown in Fig. 3, in the absence of phosphoric acid, the iron concentration in the aqueous solution before the extraction with MIBK increased with increasing HBr concentration and exceeded the permissible concentration of iron at 2 mol HBr/l. On the other hand, in the presence of phosphoric acid (0.5 ml of H_3PO_4 was added to the iron(III) chloride solution), the iron concentration reached only about 300 mg/l.

In the case of pyrite (FeS_2) sample, the iron concentration in aqueous solution before solvent extraction with diethyl ether is about 52000 mg Fe/l. Even in such a case, a satisfactory result was obtained by repeating the extraction with diethyl ether twice.

Figure 4 shows the effect of HCl and HBr on the extraction of iron with diethyl ether in the presence of phosphoric acid. The extraction of iron from the HBr solution was retarded more greatly than that from the HCl solution. Moreover, since the recovery of gold from the HBr solution was slightly higher than that from the HCl solution when it was extracted with diethyl ether, HBr was chosen for this work.

Precision and sensitivity for determination of gold

To demonstrate the reproducibility, a MIBK solution containing 1 μg Au/l was measured sixteen times by GFAAS. The precision (coefficient of variation, C.V.) was 11.7%. When 2 ml of MIBK was used in the second extraction, the detection limit for gold is considered to be 0.5 μg/l at S/N = 2.

To increase the sensitivity, the volume of

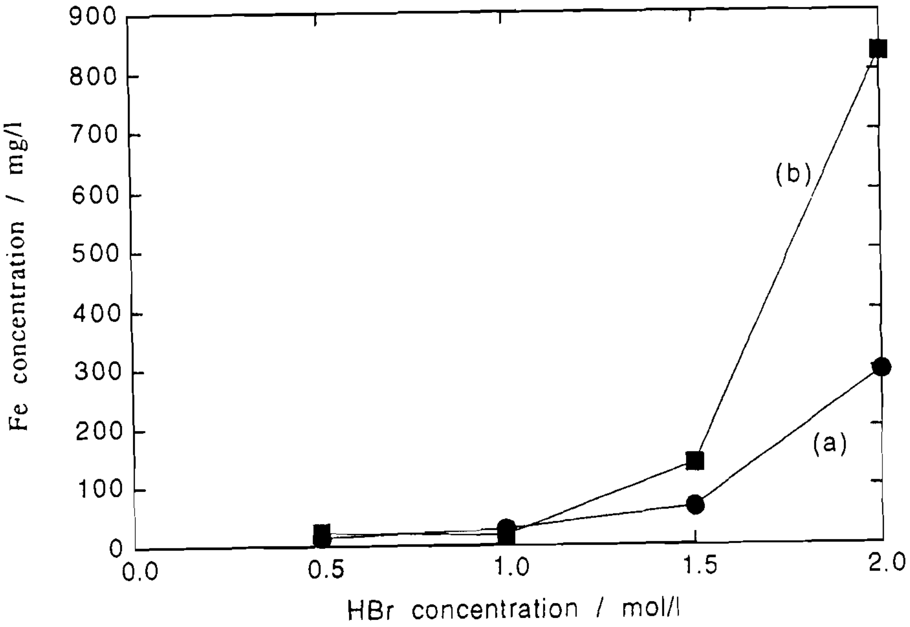
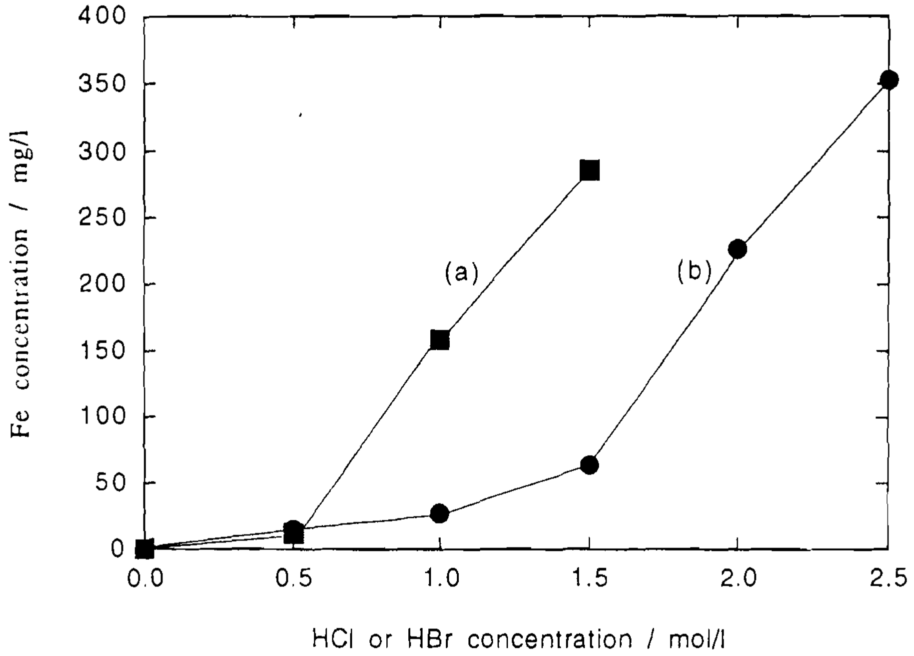


Fig. 3. Effect of phosphoric acid on the extraction of iron into diethyl ether. (a): in the presence of H_3PO_4, (b): in the absence of H_3PO_4. Fe concentration in solution A: 30000 mg/l.



ig. 4. Effect of hydrochloric acid and hydrobromic acid on the extraction of iron with diethyl ether in the resence of phosphoric acid. (a): HCl, (b): HBr.

Table 2. Gold concentration of rock reference samples

	This study	Terashima *et al*.(1992)	Arai *et al*. (1988)	Sakurai *et al*. (1991)
	GFAAS	GFAAS	INAA	INAA
JB-2	6.5	5.36	6.31	7.3
JB-3	3.1	2.06	2.36	—

GFAAS: Graphite furnace atomic absorption spectrophotometry.
INAA: Instrumental neutron activation analysis.

MIBK in the second extraction was reduced from 2 to 0.5 ml. As expected, the sensitivity increased quantitatively by decreasing the volume of MIBK.

Interference from foreign elements except iron

Shimizu *et al.* (1984) examined the effect of foreign elements on the determination of gold by a combination of GFAAS and one-stage solvent extraction with MIBK. According to their report, the interference from foreign elements except iron is negligible.

Determination of gold in standard rocks

In order to check the validity of the analytical method described above, geological standard rocks supplied by the Geological Survey of Japan (JB-2 and 3) were analyzed. The analytical data are given in Table 2 together with the published values obtained by graphite furnace atomic absorption spectrometry (GFAAS) or neutron activation analysis (NAA). The present results are close to those of NAA by Arai *et al.* (1988). Using the present analytical method, the trace of gold in rock and iron mineral samples can be determined accurately in an MIBK medium by GFAAS without interference from iron.

Determination of gold in sedimentary rock samples

To elucidate the source of gold in gold deposits, it is essential to determine the gold contents in various source rocks. In this work, the present analytical method was applied to the determination of gold in some sedimentary rocks consisting of sandstone or shale and pyrite from the Shimanto belt, southwestern Japan.

Table 3. Gold contents of sedimentary rocks and pyrite collected from Shimanto belt

Sample	Location	Fe_2O_3 (%)	Au (ppb)
S	U	1.88	1.0
S	U	3.63	2.3
S	U	2.82	1.9
S	U	4.14	1.1
S	U	0.74	1.2
S	U	5.22	2.0
S	U	3.56	1.3
S	U	3.07	2.1
S	U	3.35	1.2
S	U	2.74	2.5
M	U	7.75	1.6
BS	N	2.29	1.2
BS	N	4.88	1.6
BS	N	6.22	1.1
BS	N	6.31	1.8
QV	H	6.30	2.3
QV	H	0.02	0.9
PY	H		17

Location: U, Uwajima; N, Nobeoka,; H, Hitoyoshi.
Sample: S, Sandstone; M, Mudstone; BS, Black shale; QV, Quartz vein; PY, Pyrite.
The Fe content was measured by X-ray fluorescence method.

As represented in Table 3, since the iron content in the sedimentary rocks was relatively low, the extraction with diethyl ether was performed once, while, in the case of pyrite, it was repeated twice. The gold content in the sandstone and shale was in the range of 1.0 to 2.5 μg/kg and 1.1 to 1.8 μg/kg, respectively. The gold concentration in the rocks collected from the Shimanto belt is low and the concentration variation is relatively small between different areas. On the other hand, the gold concentration in pyrite is about ten times larger than that in the sedimentary rocks.

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