



Mr. Kent Morton  
Mill Superintendent  
Giant Yellowknife Mines Ltd.  
Yellowknife, N.W.T.

October 15, 1981

Dear Kent:

Earlier this year we tried to interest Giant in purchasing our calcine residue stockpile. Giant declined siting roaster feed problems.

I am taking the liberty of enclosing a research report on the calcine residues. Possibly you would reconsider accepting the residue with this new information.

I would like to thank you again for the well organized tour of the alkaline-chlorination plant.

Yours truly,

*L.P. Stephenson*  
L.P. Stephenson  
Senior Concentration Engineer

LPS/ad

cc JMW  
J.R.G.  
H.M.G.

14/22 }  
how about heap roasting?  
25' x 500' x 1'

50,000 tons?  
@ 0.40 sh.

20,000 oz.  
@ 450 Grs.  
= 9M

IF WE ADD 36 T.P.D.  
i.e. 30% OF ROASTER FEED  
= ABOUT 4400 FEED.

send to plant  
50-70,000 lbs  
1972-70

COMINCO LTD./TECHNICAL RESEARCH

INTERIM REPORT

CON CALCINE RESIDUE TREATMENT

SILVER & GOLD, GENERAL SECTION 144, REPORT NO. 16

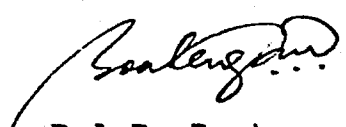
INTRODUCTION

Until 1970 the process in the Con concentrator at Yellowknife involved:

- (i) cyanide leaching of finely ground ore;
- (ii) non-selective sulphide flotation of residue from (i);
- (iii) roasting of the sulphide concentrate, and
- (iv) cyanide leaching of the calcine to produce a pregnant solution containing dissolved gold and a calcine residue.

Because of poor gold recoveries in the last stage there is now an incentive to re-treat the calcine residue, which has undergone more than ten years of weathering, for second-pass extraction.

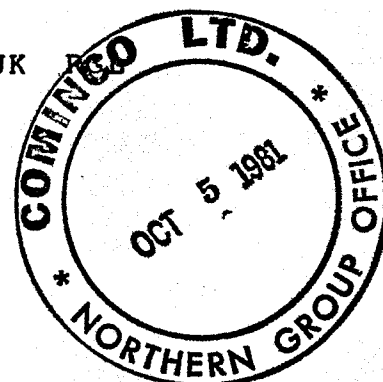
The objectives of this study included determining the properties of the residue material, investigating the possibility of gold concentration by further beneficiation, and assessing the technical feasibility of gold recovery by cyanidation leach methods.



D.A.D. Boateng  
Research Engineer

DADB/kmb  
September 4, 1981

DISTRIBUTION: Technical Research : HEH GMS WJK  
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Con Operations : DRW  
Northern Group : LPS✓



## SUMMARY AND CONCLUSIONS

1. The Con calcine residue material has undergone some weathering, the extent of which depends on depth. Oxidation reactions within the residue material, which should result in the liberation of its gold content and make it accessible to the leaching reagent, are not sufficiently extensive and have been limited to the surface material. The gold content of the material is closely associated with iron-sulphur compounds.
2. Beneficiation techniques including magnetic separation and hydrocyclone classification, were not found to be effective methods for the concentration of gold in the material.
3. Water percolation through beds of the material was slow and found to depend upon the height of the bed and the manner in which beds were formed. Percolation type leaching methods are not feasible.
4. A wash and conditioning of the material in alkaline medium prior to cyanide leaching were found to be necessary to control reagent consumption.
5. For a composite residue material (from all depths) cyanide leaching after pre-conditioning did not result in satisfactory gold recovery (~40%); a somewhat better recovery was obtained from the more weathered surface material (68.6%).
6. While variations exist with depth and location of material, a gold content of 0.4 oz/ton and a calorific value of 500 kcal/kg appear to be representative.

## RECOMMENDATION

A re-roast of the material will likely free the gold for leaching, but since the Con does not have the required facilities, a processing arrangement with Giant Yellowknife Mines Ltd., who are equipped for roasting, should be considered.

## DETAILS

### RESIDUE MATERIAL

Based on recommendations by Technical Research, the residue stocks were sampled by auger at four spaced holes, taking samples from the surface, mid and bottom sections of each hole. These wet samples were received at TRC in a drum.

### WATER WASHING

A 50g portion of each sample (as received) was agitated in 50 mL of water and filtered. Typical moisture contents of the material are given in Table 1. Table 2 describes the result obtained by the water washing, and Table 3 typical iron and acid content in wash liquors. Surface samples appear most weathered and may be easier to leach if sufficient gold liberation has occurred.

Table 1: Typical Moisture Contents of Con Calcine Residue Material

<u>Sample</u>	<u>Moisture Content, %</u>
1 Top	20.8
1 Mid	18.2
1 Bottom	17.2
2 Top	15.4
2 Mid	15.0
2 Bottom	12.8
4 Top	18.5
4 Mid	13.5
4 Bottom	12.0

Table 2: Water Wash of residue material

<u>Sample</u>	<u>pH of Filtrate</u>	<u>Comment</u>
1 Top	4.8	Unusually wet surface sample (decrease of acidity with depth)
1 Mid	5.6	
1 Bottom	8.0	
2 Top	2.1	Yellowish brown filtrate (likely Fe <sup>3+</sup> ) Colourless filtrate Colourless filtrate (decreasing acidity with depth)
2 Mid	4.2	
2 Bottom	4.4	
3 Top	6.3	colourless filtrates - (low acidity)
3 Mid	6.7	
3 Bottom	7.1	
4 Top	1.8	Brown filtrate, most acidic sample Colourless filtrates, low acidity
4 Mid	6.1	
4 Bottom	6.6	

Table 3: Iron and Acid in Wash From Calcine Residue

<u>Sample</u>	<u>Wash</u>		<u>Original Moisture In Material</u>	
	<u>g/L Fe</u>	<u>g/L H<sub>2</sub>SO<sub>4</sub></u>	<u>*g/L Fe</u>	<u>*g/L H<sub>2</sub>SO<sub>4</sub></u>
2 Top	2.8	6.5	21.0	48.7
2 Mid	0.23	0.39	1.76	2.99
2 Bottom	0.41	1.47	3.61	13.0
4 Top	17.6	23.0	112.7	147.3
4 Mid	0.007	0.88	0.06	7.40
4 Bottom	0.008	0.78	0.07	7.28

\* estimated

As shown in Table 4, the gold content of the calcine residue varied only slightly with location.

Table 4: Partial Analysis of Con Calcine Residue Samples

<u>Sample</u>	<u>Oz/Ton Au</u>	<u>% Fe</u>	<u>% S<sub>2</sub></u>
1 Top	0.420	42.8	18.4
1 Mid	0.420	47.1	15.6
1 Bottom	0.455	49.5	15.8
2 Top	0.450	46.3	31.4
2 Mid	0.405	45.8	26.1
2 Bottom	0.440	43.1	28.7

An average gold content of 0.432 oz/ton was obtained in the residue. The distribution of gold with particle size of the composite material from #2 hole is shown in Table 5.

Table 5: Distribution of Gold With Particle Size  
(#2 Hole Composite Sample)

<u>Size Fraction (Mesh)</u>	<u>Mean Wt %</u>	<u>Oz/Ton Au</u>	<u>% S (Total)</u>
-150 + 200	10	0.300	30.5
-200 + 270	17	0.360	35.3
-270 + 325	15	0.360	34.2
-325 + 400	11	0.360	32.6
-400	45	0.580	21.0

A material balance shows that 50-60% of the gold reported in the fines which made up about 45% of the total residue material.

#### BENEFICIATION

##### Magnetic Separation

The feasibility of using magnetic separation for further beneficiation of the residue material was investigated briefly. A separation achieved on dry samples showed that both the proportions and composition of the magnetic and non-magnetic fractions depended on the location and particle size of the sample. Typical results are given in Table 6.

Table 6: Gold Concentration By Magnetic Separation  
(Sample #2 Hole, -200 +270 mesh, Au:0.360 oz/ton)

	<u>Wt %</u>	<u>oz/ton Au</u>	<u>% Fe</u>	<u>% S</u>
Magnetic Fraction	60	0.460	49.5	26.5
Non-Magnetic Fraction	40	0.200	40.9	45.4

The sulphur to iron ratios of 0.54 and 1.11 obtained for the magnetic and non-magnetic fractions indicate the predominance of pyrrhotite (FeS) and pyrite (FeS<sub>2</sub>) in the respective fractions. For a sample of particle size -100 +150 mesh, x-ray diffraction analysis showed pyrite (FeS<sub>2</sub>) dominant in both fractions. The main gangue material in the residue samples was quartz, and x-ray fluorescence showed the elemental constituents to be Si, Fe, S (high); Zn, Ca, K, As, low Pb and trace Sb.

Magnetic separation could concentrate 80% of the gold in 60% of the residue material but the 40% left had a grade of 0.200 oz/ton.

### Hydrocyclone Classification

Analyses of screen-sized samples did not offer any hope for a reasonable size split of the residue material. It was speculated, however, that the hydrocyclone which offered separation on the basis of particles weight may enable a split of the residue material with favourable gold concentration.

A laboratory glass version (Liquid-Solid Separation Ltd., MK3) hydrocyclone was used in this study. The tests carried out on a composite sample from #2 hole and the results are summarized in Table 7. The results do not appear encouraging at any split.

### WATER PERCOLATION (PERMEABILITY)

Percolation leaching is desirable because it removes the need to re-slurry the residue material, and associated slurry handling and filtration may therefore be avoided. The success of such leaching would depend on solution flow through a fixed residue bed and factors controlling reagent consumption.

### Hydraulic Permeability Experiments

A. Residue material as received was placed in two 54 mm diameter columns to form beds of depths 200 mm and 400 mm respectively. Water was then added from the top and the advancement of the water front observed.

B. Residue material was slurried and transferred into columns forming settled beds of depths 180 mm and 360 mm respectively. A constant water head was maintained above the bed and the water flow measured.

### Observations

In A, the advancement of the water front appeared to be inversely proportional to the square of the bed depth. However, the situation changed after the water fronts had reached the bottom of the beds, and the flow rates appeared equal for a period of 150 min. Then after 220 min. the flow in the deeper bed was 80% that of the shallower bed. In each of the beds there was 30% shrinkage of the depth of the bed.

Because of the difficulty in reproducing these beds and in achieving steady state, behaviour in such beds cannot be accurately predicted. Typical water flux in the first two hours was  $1.8 \times 10^{-3}$  mL/cm<sup>2</sup> min.

**TABLE 7: Hydrocyclone Classification of Con Calcine Residue Material**

Hydrocyclone Size	Vortex Finder	Apex Nozzle	Slurry Density	Pressure Drop	Overflow (10 sec)		Underflow (10 sec)			
					Wt(g)	Wt of Solids(g)	Oz/ton Au	Wt(g)	Wt of Solids(g)	Oz/ton Au
30mm	8.26mm	1.32mm	17.5%	517mm	1378	134.4	0.545	166.5	122.4	0.325
30mm	8.26mm	1.32mm	17.5%	259mm	912.6	91.5	0.535	120.3	88.1	0.305
30mm	8.26mm	1.70mm	0.7%	517mm	1549	14.0	0.744	92.1	65.8	0.380
15mm	4.09mm	1.51mm	0.7%	517mm	453.0	3.3	0.714	25.9	18.2	0.380



In B, the observed flows in the first 5 hours are given in Table 8. The values of  $n$  are as in the relation:

Rate  $\propto h^n$

where  $h$  is the bed height. The flow rate was approximately inversely proportional to the square of the bed height for the first day. After one week the flow in the deeper bed had decreased and a value of  $n$  changed to about -2.5.

Table 8: Flow of Water Through Fixed Residue Beds

<u>Time (min)</u>	<u>Volume Passed (mL)</u>		<u>Rate (mL/min)</u>		<u><math>n</math></u>
	<u><math>h = 180</math> mm</u>	<u><math>h = 360</math> mm</u>	<u><math>h = 180</math> mm</u>	<u><math>h = 360</math> mm</u>	
36	30	7	0.83	0.23	-1.85
51	42	10	0.80	0.20	-2.00
114	92	21	0.79	0.17	-2.22
175	142	33	0.82	0.20	-2.04
230	189	43	0.85	0.18	-2.24
260	212	48	0.77	0.17	-2.18
290	236	53	0.80	0.17	-2.23
		Overall	0.81	0.18	-2.17

Apparent steady fluxes of 0.035 mL/cm<sup>2</sup> min and 0.0065 mL/cm<sup>2</sup> min were obtained for bed height 180 mm and 360 mm respectively.

#### PRETREATMENT

This consisted of washing of the material to be treated and setting it aside for several hours under alkaline conditions to allow the reaction:

$\text{FeS} + 2\text{OH}^- \rightarrow \text{Fe}(\text{OH})_2 + \text{S}^{=}$   
to occur. The material was further washed to remove as much of the sulphide ion formed as possible. (See details in Appendix A.)

#### LEACHING

Except in one case the leaching experiments were conducted in cylindrical leaching vessels in which agitation was achieved through rolling. Details of the various tests are given in Appendix A. The results obtained are summarized in Table 9.

Gold recovery from the leaching of the composite samples did not far exceed 40%. However a recovery of 68.6% was obtained from the leaching of a surface sample. Obviously the gold recovery appears to be dependent on the extent of weathering in the material.

Because of the need for pre-conditioning of the residue material, the alkaline consumptions would be higher than conventional. The consumptions shown in Table 9 would be somewhat reduced through the recycle of alkaline liquors.

TABLE 9: Leaching Tests

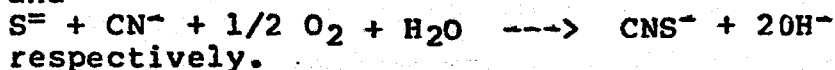
Test No.	Calcine: Residue	Duration hrs.	Alkaline Consumption lb/ton	NaCN Consumption lb/ton	Gold Recovery %	Silver Recovery %
1	#2 Composite 0.420 oz/t Au	27	8.3 (CaO)	1.1	35.7	-
2	#2 Composite 0.420 oz/t Au	24	-	-	38.1	-
3	#2 Composite 0.420 oz/t Au	48	-	-	38.1	-
4	#2 Composite 0.420 oz/t Au	72	11.9 (CaO)	3.8	41.0	55.0
5	#2 Composite 0.420 oz/t Au	72	22.5 (NaOH)	1.05	41.3	-
6 *	#2 Composite 0.420 oz/t Au	21	16.2 (NaOH)	1.1	39.6	-
7	#2 Top (surface sample) 0.465 oz/t Au	72	13.3 (NaOH)	0.67	68.6	21.0

\* Carried out in an open vessel agitated by a 4-blade stirrer.

Both cyanide and alkaline consumptions obtained are still much lower than those obtained in a previous study<sup>1</sup> on the same material. The difference may be largely attributed to the pretreatment scheme. The initial wash removes much of the acid and soluble iron (which may be as high as 147.3 g/L and 112.7 g/L respectively in the moisture associated with the residue material). The alkaline conditioning and second wash helps to remove metal sulphide and sulphide ion, both of which would consume cyanide by:



and



#### ELECTRON MICROSCOPIC STUDY

Samples of the residue material were examined by means of the Scanning Electron Microscope. Micrographs of particles were taken. Point to point semi-quantitative analyses showed that gold occurred in some association with iron and sulphur, the relative amounts varying widely from particle to particle. Similar variations existed for both the original material and leached samples, the only difference being perhaps in the relative amounts of gold in the solid matrix.

The semi-quantitative proportions of iron and sulphur appear to indicate the presence of both pyrite ( $\text{FeS}_2$ ) and pyrrhotite ( $\text{FeS}$ ). It is not clear how the gold is held. However since no "free" gold particle was found, one may infer that the gold is not sufficiently liberated. Points on particles examined had dimensions of the order of a single micron; the gold would therefore be very finely desminated in a matrix containing predominantly Fe-S compounds.

#### CALORIFIC VALUE OF RESIDUE MATERIAL

The calorific value or the heat of combustion of the residue material was determined. The combustion products were sulphur dioxide and iron oxide ( $\text{Fe}_2\text{O}_3$ ), the latter verified by x-ray diffraction. The results obtained are given in Table 10. While a simple explanation of the variations obtained is not apparent, for the entire material an average value of 500 kcal/kg may be taken as representative. The corresponding values for pure pyrrhotite ( $\text{FeS}$ ) and pyrite ( $\text{FeS}_2$ ) are 1669 kcal/kg and 1688 kcal/kg respectively.



Table 10: Carific Value of Residue Material

<u>Sample</u>	<u>Kcal/kg</u>
Hole #2, Top	751
Hole #2, Mid	617
Hole #2, Bottom	659
Hole #2, Blend	602
Hole #3, Blend	390
* Composite	497

\* The composite sample represents a blend of 12 samples from four different sites at three different depths.

PARTIAL ANALYSES OF RESIDUE MATERIAL

Various partial analyses of samples of the residue material have been earlier presented. The assays obtained for a composite sample, given in Table 11, may be taken as representative of the average stockpile material.

Table 11: Representative Partial Analyses of Con Calcine Residue Material

<u>Element</u>	<u>Concentration</u>
Au	0.400 oz/ton
Ag	1.20 oz/ton
Fe	43.3 %
S	21.2 %

## DISCUSSION

The water-washing tests revealed that weathering had occurred to different degrees with site and position. Contrarily to expectation, the weathering has been limited to the surface material.

Magnetic separation, while concentrating 80% of the gold content in 60% of the residue material could not be suitable, since the remaining 40% contained 0.200 oz/ton gold and could not be rejected.

Hydrocyclone classification is similarly unsuitable since a reject fraction of low gold content could not be produced at a suitable solids split.

The leaching tests reveal that only about 40% of the gold content of a composite sample could be leached. The leachable gold is virtually completely solubilized within 24 hours. The washing and pre-conditioning were useful in minimizing reagent consumption during leaching.

The somewhat better (68.6%) gold recovery obtained from leaching surface material supports the conclusion that the lower recoveries obtained with the composite samples are due to the gold being less liberated in the bulk of the material.

It would appear that liberation of the gold by a process such as roasting would be necessary to ensure high recovery. An average calorific value of 500 kcal/kg may be taken as representative for the whole material.

The residue material is now adequately known, and since a roast cannot be carried out by the present facilities at the Con, the alternative option of a processing arrangement with Giant appears attractive.

## APPENDIX A

### Details of Leaching Studies

#### Test 1

600 g of #2 composite residue material was washed with 2 L water, the pH of last portion of water separated off was 5.1.

20 mL of 15%  $\text{Ca(OH)}_2$  slurry and 580 mL of water were added to filtered cake from above the pH of the resulting slurry being 10, and the whole was left overnight.

The slurry was filtered and the solid washed with two 150 mL portions of water.

The solid was reslurried (55% solids) and sodium cyanide added to make aqueous phase concentration of 0.8g/L.

The pH was adjusted to 10.5 by adding a small amount of  $\text{Ca(OH)}_2$  solution.

Oxygen was bubbled through slurry (supersaturation occurs - 39.8 mg/L  $\text{O}_2$ ). The reaction vessel was then placed in position and rolled to effect agitation.

The pH of slurry was controlled automatically and maintained at 10.5.

The cyanide and oxygen concentrations in the aqueous phase were measured from time to time. The average oxygen concentration was 7.4 mg/L, and total handling time was 27 hours.

#### Test 2

Similar to Test 1 except leaching time was 24 hours and a 1% solution of  $\text{Ca(OH)}_2$  was used to maintain a pH of 10.5.

#### Tests 3 & 4

Were carried out in the same manner as Test 2 but the leaching periods were extended to 48 hours and 72 hours respectively.

#### Tests 5 & 6

In these a 10% NaOH solution was used for pH control. In test 6 the leaching was carried out in an agitated open vessel for 21 hours.

#### Test 7

A surface sample was leached in the same manner as Test 5 which treated a composite sample.

## References

- 1 Al Gagnon, "Optimization of the Dissolution of Gold From Calcine Residue", December 1973.