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Investigating the Reprocessing of Carolin Mine Tailings

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Abstract

During the ten year period between Aug 2001 and Aug 2011 the price of gold has experienced an increase of nearly 1500 USD, a jump from 300 USD/oz to close to 1800 USD/oz. This has resulted in a resurgence of interest in gold tailings for what was once below cutoff grade material which is now valuable ore. The tailings at Carolin Mine are the post-processed product from the previous operation which ran from 1982 to 1984 when the average overall gold recovery was 50.2%. This low recovery indicates that a substantial amount of gold-bearing material was sent to the tailings impoundment. This paper describes the history and methodology used in re-evaluating the Carolin mine tailings material as part of a metallurgical feasibility program.

Property History

Module Resources Incorporated is a Canadian mineral exploration company focused on the development of its Ladner Gold Project located approximately 150 km east of Vancouver, BC, Canada, and accessed by road from Coquihalla Highway Corridor (Highway 5). The project area includes several historic small gold producers including the Carolin Mine (a former underground gold producer which produced 45,000 ounces of gold from 1982 to 84), Emancipation Mine (an intermittent small gold producer from 1916 to 41), Pipestem Mine (an intermittent small gold producer from 1935 to 37), and numerous gold prospects situated along the under-explored Coquihalla Gold Belt.

In conjunction with the development of the property, Module is investigating the potential for re-processing gold from the tailings. The tailings are the post-processed product of a previous operation of the Carolin Mine from 1982 to 1984. During this time, approximately 900,000 tons of ore were processed to recover approximately 45,000 oz of gold with the average overall gold recovery at 50.2%. This indicates that a substantial amount of gold-bearing material was sent to the tailings impoundment.

The original process had the ore crushed underground and moved by conveyor belt to the external coarse ore bin. The adjacent mill, with a nominal 1,500 tpd capacity, processed the ground ore by flotation, cyanidation and a Merrill Crowe zinc precipitation circuit. Approximately 10,000 tonnes of mine tailings were reprocessed in the Carolin Mill in October 1984, just prior to shutdown. Mill head grade for this tailings material was reported as 1.99 gm/tonne and gold recovery as 65.2%. The mine ceased operation in 1984 due to low gold recoveries, grade control problems and the rapidly falling gold price. The mill ultimately collapsed due to snow loading in 1996 and little remains today of the mine's surface infrastructure except for access roads and the tailings deposit.

In the mid-1990s, Athabaska Gold Resources Ltd. conducted significant exploration including underground development and drilling and a gold resource study of the tailings deposit. This work had considerable success in demonstrating gold mineral resources in the tailings deposit and in the underground Carolin Mine deposit. However, production restart was not achieved due in part to the low gold price which existed at that time. A recent NI 43-101 compliant resource calculation is presented in Table 1 (Pearson, 2011) The calculation is based on the drilling and sampling conducted by Athabaska (Mellis, Ash). It should also be noted that the original drilling was completed on approximately 60% of the tailings impoundment where the tailings was the thickest.

Interest in gold tailings is resurging due to the current record high gold prices, making what was once below cutoff grade material now valuable ore.

Table 1: Indicated and Inferred Resources

Au Cutoff (g/t)	Tons> Cutoff (tons)	Grade > Cutoff		Contained Ounces Gold
		Au (g/t)	Au (oz/t)	
1	445,378	1.83	0.05	23,746
1.5	334,529	1.99	0.06	19,446
2	124,058	2.45	0.07	8,861

CAROLIN TAILINGS - INFERRED RESOURCE

Au Cutoff (g/t)	Tons> Cutoff (tons)	Grade > Cutoff		Contained Ounces Gold
		Au (g/t)	Au (oz/t)	
1	93,304	1.85	0.05	5,037
1.5	68,252	2.07	0.06	4,113
2	40,596	2.28	0.07	2,695

Given the previous context, this paper describes the history and the methodology used in re-evaluating the Carolin mine tailings material using modern processes under the following objectives:

- Determining, through metallurgical testing, whether the tailings can be processed to recover the contained gold,
- Providing early cash flow to Module Resources assuming the economics are positive,
- Reducing any potential environmental liability associated with the tailings material through reprocessing, and
- Producing a benign tailings residue that could be used as a paste backfill in the Carolin Mine.

In October, 2009, the tailings were sampled using a specially designed amphibious craft. Nine vertical randomly selected holes were drilled to 3 meters depths with recovered core material weighing approximately 75 kg in total. All materials were transported to the mineral and processing laboratory at the University of British Columbia, Vancouver, BC, where they were logged and sampled. Qualitatively, the sampled material showed nearly no indication of oxidation, which was somewhat surprising as some indication of oxidation had been expected.

Metallurgical work was conducted to ascertain the value that could be extracted from the remaining tailings material. The focus of the metallurgical program is to use modern processes and techniques to unlock this retained value. The aim is to of develop preliminary economic assessment under the stated objectives. This paper outlines the results of this investigation and touches on some of the specific highlights discovered through this approach.

A Brief Discussion on Tailings

When gold mining is brought up in conversation, it is the large bonanza-style gold deposits that typically come to mind. With the notion of large economies of scale and massive production comes yielding high returns on multi-billion dollar investments. In the context of such large operations complex engineering and logistical issues, environmental concerns, First Nations rights and public perceptions all factor in to the lengthy process of producing the mine's first gold bar. Given the high price of gold, approximately 1500 USD/oz at the time of writing, there is a resurgence of interest to shortcut the lengthy development of green-field deposits throughout the industry. With this in mind, tailings have been identified as a possible source of quick gold production. At a high level, tailings typically have all or some of the following characteristics:

- Is already crushed and milled,
- Contains material that was below a certain gold content during the former operation: (material below cut-off grade)
- Consists of post-processed ore containing unrecovered value that given an increase in the price of gold may now have greater economic attractiveness,
- Is Often stored on surface in either a dry stack or underwater within a very defined area,
- Processed using historical metallurgical methods and equipment.

Methodology

The project was divided into three main phases, each of which was allocated approximately 23kg of composited tailings material for test work.

Phase 1: Characterization

- Baseline Assays, Compositing, Initial Leaching and Concentration Response

Phase 2: Optimization of Concentration Methods

- Diagnostic Flotation and Grind Sensitivity

Phase 3: Leaching Investigation

- Bulk Concentration Production, Cyanide Bottle Rolls under various leach conditions and Experimental Salt Leaching

A challenge to metallurgical work is ensuring that the representation of the deposit, from collected samples, is accurate. Traditionally, first order metallurgical test materials are a result of the excess retrieval of core samples from geological/resources modeling drill programs. While these materials aid in determining the initial scoping and recovery responses, they do not represent the reality of what the mill may receive during production nor do they take into account the variations inherent in the rock that may affect this recovery. This lack of connection adds variability to processing choices, recovery results and ultimately the economics of the project.

For example, when building metallurgical programs for hardrock greenstone projects, alteration and lithology considerations must be looked at in order to determine overall recovery values which are then used in economic analysis. By collaborating with project Geologists, Mining Engineers and Metallurgists, considerations from a collective of engineering requirements can be built into ongoing development drilling. This ultimately provides a faster and more accurate picture of the project's viability from a metallurgical/recovery standpoint. By determining the responses of the various styles of ore, accurate recovery results can be applied to the resource model. This in turn provides guidance on how to mine and supply ore to the processing centre for specific ore optimized operation conditions. It also provides economic returns based on specifics identified by alteration and lithology.

1st Assumption - Homogeneity:

In comparison to the previously mentioned example, it was decided that (a proper sample collection program would be performed) to avoid characterizing the Carolin mine tailings on the basis of stratified differences due to insufficient resources. This decision was supported by two considerations;

1) From a mining point of view, the Carolin mine tailings pond does not allow for overly selective mining practises to be practically employed. Unless proved otherwise by further study, which at the

time was too costly to conduct random sampling, the assumption of good homogeneity of the pond had to be defaulted to in order to proceed.

2) For simplicity and cost, the material contained in the pond was again assumed to be reasonably homogenized as a result of the processing and deposition that occurred in the pond's creation. Additionally, the historical mill deposition records could not be found, and even if they were it would likely be cumbersome to model or incomplete and unusable data.

While this generic assumption of homogeneity was initially needed to begin the investigation, it was understood to be a temporary view that could change as the program developed.

2nd Assumption - The 'Low Hanging Fruit' Effect

A metallurgical challenge that arises with all tailings ponds, and one that emerged in the outset, is that the proverbial 'low hanging fruit' had already been picked. In this case, for some initially unknown metallurgical reason, the former Carolin process plant failed to extract the remaining value from this material. This is the source of the first open-ended question which sets up the basis of the metallurgical testing methodology for the Carolin mine tailings deposit; Why did the original mill fail to extract the gold which remained contained in the tailings?

In addition to how the material is collected, another main constraint to all metallurgical investigations is the amount of material available to test. It is important to appreciate that sample mass constraints pose challenges in regards to what can be done and how the investigations can proceed. In the specific case of the Carolin tailings, a complete characterization of the tailings was needed due to the age, historical metallurgical information available and the recovery challenges that were experienced by the former mill. This proved to be a difficult pursuit as the type of tests used on the ~75kg of material needed to be chosen carefully in order to assure retention of adequate material for reliable and meaningful test results. eg: a 1kg diagnostic flotation test may well only produce a few dozen grams of concentrate, all of which is required for an accurate gold and element assay. Hence little or no further testing can take place.

As result of this methodology, a strict mass conservation was followed at all steps of the testwork When starting with little background information and suspect historical data, due to 25 years of weathering, results that help develop the flowsheet may not be immediately obvious. With these difficulties in mind, Phase 1 of the Metallurgical program focused on the analysis of bulk assay results from both the individual holes (a check on the homogeneity assumption of the tailings pond) and assays on a mixed master composite as well as other tests to attempt to discover why Carolin mine had difficulties in recovery.

Metallurgical Results

Phase 1 Overview:

It is important to restate that mass conservation was a major factor in governing how the metallurgical test program proceeded. Given the initial assumptions mentioned, a baseline characterization began along with attempts to answer two fundamental parts of the initial open-ended question that was posed: "Why did the original mill fail to extract the value contained in the tailings?"

1. What is the material's response to Cyanide leaching?
   - Are refractory and carbonaceous effects a concern to gold recovery?
2. Can this material be easily concentrated?
   - Producing higher grade material within less mass will have a major impact on downstream processes, capital/operating costs and the overall economics.

To this effect, the following overviews show what testing was conducted.

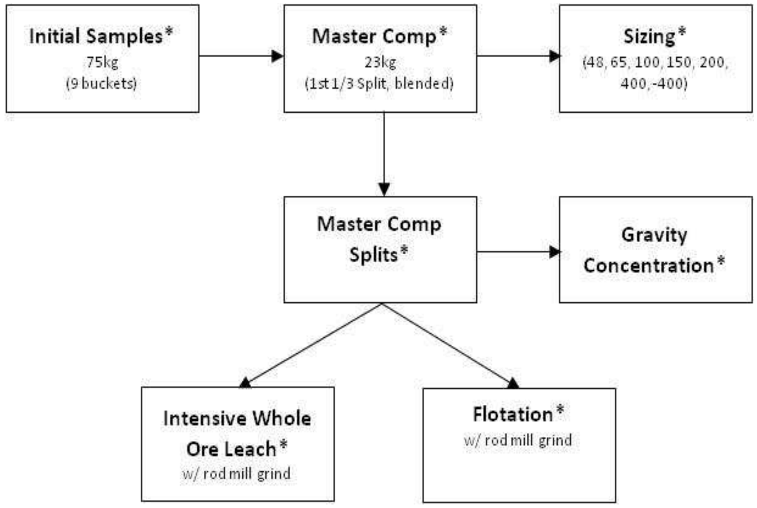

Figure 1: Phase 1

* Assays done at all steps.

Prior to any compositing and testing, the assumption of homogeneity needed to be examined. Figure 1: Phase 1 shows a cross section of assay results and for the most part there was little variability between the sampled holes. Instead the variability was deemed to be too insignificant to warrant special compositing under the context of how much material was available and the goals of the program.

Table 2: Select Head Sample Results

Sampling Hole #	Au (GM/T)	Ag (PPM)	As (PPM)	Cu (PPM)	Total/S (%)	S/S- (%)	Fe (%)
1a	1.44	<2	2506.2	262.9	1.45	0.87	6.98
2a	1.48	<2	2218.7	202.4	1.49	0.86	6.82
3a	1.41	3	2549.7	256.4	1.81	1.12	7.03
4a	2.38	<2	2628.2	316.5	2.14	1.37	6.55
5a	1.33	<2	2166.9	225.8	1.5	0.88	6.86
6a	1.9	3	2263.2	235.9	1.72	0.97	7.05
7a	1.74	<2	2390.5	271.6	1.93	1.14	7.02
8a	1.67	<2	2578.2	295.3	<0.02	I.S.	7.36
9a	1.38	<2	2452.1	295.3	1.62	0.86	7.3
Duplicate Samples							
1b	1.27	<2	2575.5	258.7	1.38	0.78	7.16
2b	1.63	<2	2229.1	203.2	1.47	0.81	6.9
3b	1.54	<2	2524.9	257.9	1.82	1.03	7.06
4b	2.53	<2	2607.6	312.8	2.19	1.37	6.7
5b	1.49	<2	2158.1	218.3	1.6	0.92	6.92
6b	1.77	2	2286.9	227.6	1.77	1.04	7.08
7b	1.95	3	2439.3	269.3	1.89	1.06	7.26
8b	1.9	2	2613.4	286.2	1.97	1.14	7.46
9b	1.46	4	2434.4	291	1.61	0.84	7.36
Average	**1.68**	**2.83**	**2423.49**	**260.39**	**1.73**	**1.00**	**7.05**
Max	**2.53**	**4**	**2628.2**	**316.5**	**2.19**	**1.37**	**7.46**
Min	**1.27**	**2**	**2158.1**	**202.4**	**1.38**	**0.78**	**6.55**

Much of the assay data was then plotted against gold to provide the first look at what elements may or may not be correlated. From the historical reports on the Carolin mine deposit, it is known that there is a minor refractory component which involved pyrite-arsenopyrite where gold particles are occluded within the host material making them inaccessible to chemical treatments. Additionally a minor carbonaceous component that had preg-robbing characteristics existed; gold in cyanide solution was effectually stolen out of the pregnant leach solution (PLS) by naturally occurring carbon that was unable to be captured and thus exited in the mill's waste stream. However, for post processed tailings material, these points needed to be investigated as the original processing plant may have affected the material, either physically or chemically, with unknown consequences. About a dozen relationships were investigated but one example, Figure 2, visually shows a good correlation between gold and sulphide which confirmed that the tailings still contained a reasonable amount of gold associated with the sulphides.

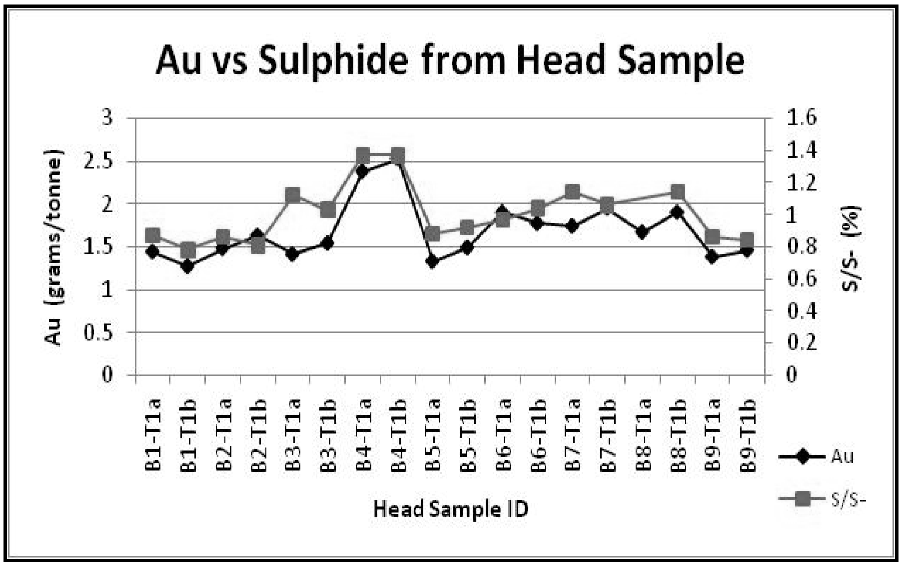

Figure 2: Gold (Au) - Sulphide Comparison by Location w/ Duplicates

Following the correlative investigation, confirmation through optical microscopy of super pan concentrate confirmed again that gold was finely disseminated within arsenopyrite and that occluded gold was on average less than 2 μm in diameter and exposed or locked within various sulphides. Pyrite was the main occurrence with some chalcopyrite and pyrrotite. An example of this is shown in Figure 3 where the arrow points to a gold particle partially locked in arsenopyrite. Weathering did not appear to be significant.

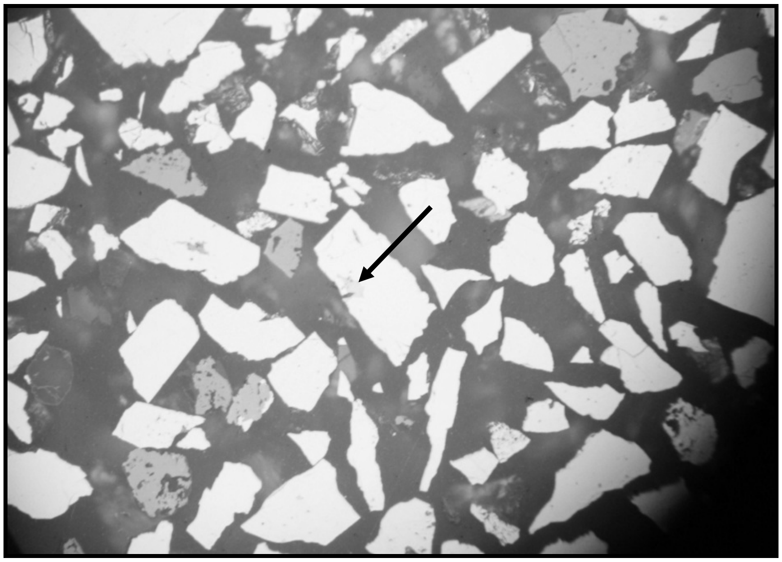

Figure 3: Typical Occurrence of Visual Gold through Visual Microscopy of Super Pan Concentrate

All of these indications many provide a plausible answer to the question of why Carolin could not recover this trapped gold; the processing plant may not have been able to successfully employ their flotation step for sulphide recovery and/or the use of cyanide or grinding was not properly optimized.

Phase 1 - Intensive Whole Ore Leach (iWOL)

Proving whether or not the tailings material can be leached with cyanide, in a general sense, was considered a crucial point in directing future work. As result, Phase 1 consisted of the somewhat unorthodox decision of going straight to an intensive whole ore leach to examine such effects.

The blended master composite (1kg) of the as-received material underwent a Rod Mill grind to achieve a P80 of 93 μm, down from the as-received P80 of 180 μm. Using a cyanide concentration of 20 g/L, the intensive leach obtained a measured overall gold recovery of 87.2% after carbon addition took place and as calculated by the tailings. When charted, the leach results displayed a lower saturated recovery value that averaged around the 75% (+/- 3%) mark. The kinetics were superb in that total gold-cyanide saturation was obtained between the 4 and 8 hr timeframe. The difference in recovery can be attributed to the fact that gold in the PLS was removed by carbon addition at the end of 48 hrs and cyanide then became free again to further leach the remaining gold, thus continuing the cycle. The conditions used in this iWOL are likely not feasible on non-upgraded material, or simply not feasible for any Carolin mine concentrate. They give a good indication of the maximum recovery that may be obtained and show that this material is indeed amenable to Cyanide. Figure 4 shows the results of the iWOL prior to the addition of carbon.

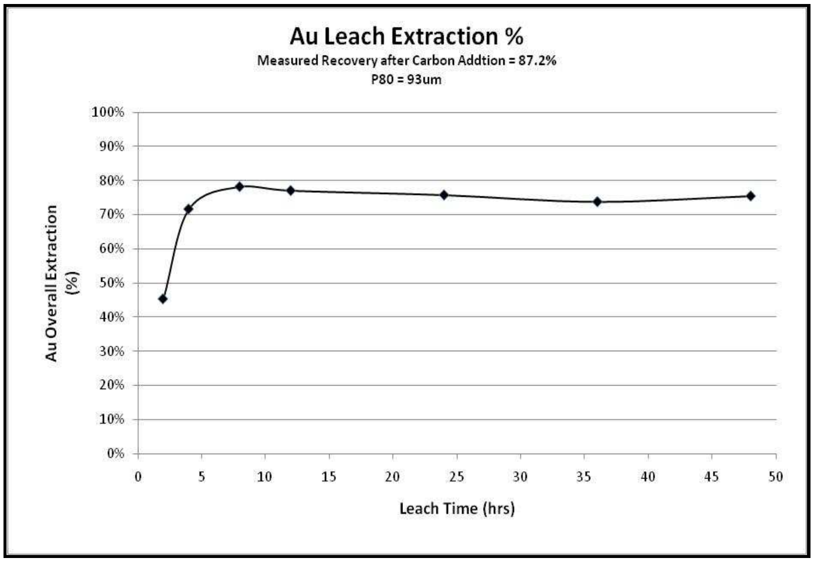

Figure 4: Intensive Whole Ore Leach Results of Gravity Concentrate

Phase 1 - Gravity Concentration

The positive iWOL result was key in directing the work that came afterwards. It showed that the tailings material is not overly refractory and/or preg-robbing. Therefore diagnostic processes in those areas do not need to be pursued. This translates into a large savings of sample material for further work.

Gravity concentration was then investigated due to the known benefits of non-chemical concentration and simple operational considerations when compared to flotation. Using a lab scale Knelson concentrator, two sets of the master composite material were then cycled through a total of 7 times each. Tailings material from each pass was collected and then re-cycled through the concentrator in order to obtain as much gravity recoverable gold as possible. Pass A yielded a result of 27.74 Au g/tonne (which represents 21.6% gold recovery) and .54 Au g/tonne was obtained for Pass B, (which represents a 18.3% gold recovery.)

At the time of these gravity results, it was felt that no further work could be justified to improve on the mediocre performance given that only 20kg from the 73kg was used in generating the results. This was considered a high cost of sample mass to use as flotation and grinding and leaching investigations still needed to occur.

Phase 1 - Flotation Results

Without historical information on the particle sizes used, dosages, chemical types or retention times to help guide the current work, a 'stab in the dark' was made. To determine the initial flotation response, Academics at UBC and experienced lab technicians were consulted. Flotation results this yielded 66% recovery (by tails), with an 8% mass recovery using a three stage flotation involving Potassium Amyl Xanthate (PAX) and Aeroflot 208 (A208) which began with a 10 minute rod mill grind. It was determined that this result was more promising than gravity due to the nature of the material and the initial flotation conditions used.

Phase 2 - Overview

With confirmation that head material can be leached successfully and floated, the 2nd phase of the program took place: determining the diagnostic flotation and grinding size.

Due to work done in Phase 1 and the associated gold-sulphides correlation, flotation was seen to offer the best potential to recover sulphides as well as the now indentified finely disseminated gold. As such, optimizing the flotation reagents scheme and conducting a grind sensitivity became the focus of Phase 2. An overview is shown below.

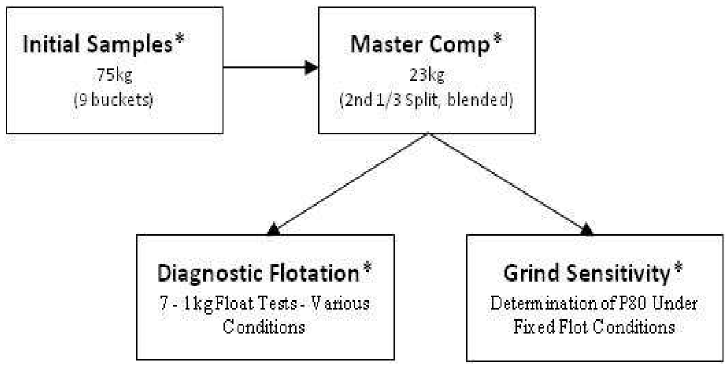

Figure 5: Phase 2

* Assays done at all steps.

Phase 2 - Flotation Sensitivity

After consulting with UBC academics and experienced flotation technicians, the investigation on the flotation characteristics considered the following as the most important question to guide test-work.

- What effect, if any, did 25 years of weathering and exposure have on the material's ability to be concentrated by flotation?

Although extensive weathering was not seen in optical microscopy, a cautious approach was pursued. The approach investigated the extent of Grinding, Attrition Scrubbing and Sulphuric Acid addition needed to reactivate the material's surface and provide a good flotation response.

In an effort to maintain simplicity and to follow sample conservation, PAX was elected to be the base collector used in all tests due to the sulphide nature of the material and its positive results from the initial flotation test. Cytec A208 and 3418A were then used in various combinations with grinding, scrubbing and acid to get a look into the material's response (Table 3) Detailed P80/Grinding effects are investigated in following section Phase 2 - Grind Sensitivity.

Table 3: Diagnostic Flotation Scheme

Flot #	Recovery (%)	Mass Pull (%)	Grinding	Scrubbing	PAX	Acid (H_2SO_4)	A208 (promoter)	3418A (promoter)
0	65.6	8.0	Yes	-	Yes	-	Rougher	-
1	65.3	15.3	-	-	Yes	Pre-Treat	-	Rougher
2	60.3	9.2	-	Yes	Yes	-	Rougher	Cleaner
3	48.8	6.8	-	Yes	Yes	-	-	Rougher
4	70.5	11.8	Yes	-	Yes	-	-	Rougher
5	74.8	11.7	Yes	-	Yes	-	-	Rougher (x2 Reagents)
6	77.1	20.7	Yes	-	Yes	Pre-Treat	-	Rougher

7	76.6	19.5	Yes	-	Yes	-	Rougher	Cleaner

Phase 2 - Grind Sensitivity

The grind sensitivity proceeded in the manner to be expected for finely disseminated gold. The further it was milled the better the gold flotation response was. For each grind test, flotation conditions were fixed from previous flotation work and considerations were made regarding the costs of the reagents and recovery benefit seen. A208 was determined to be more cost effective given its recovery effect and was shown between Flotation Test #0 and Test #4 and between Test #2 and Test #3. By comparison, 3418A is 3 times the cost of A208 on a per kg basis and it was decided that it would not be pursued further. In order to ensure a reagent agnostic response, consultation with lab technicians suggested doubling the reagent scheme from Test #0. The grind - recovery response is shown in Figure 6.

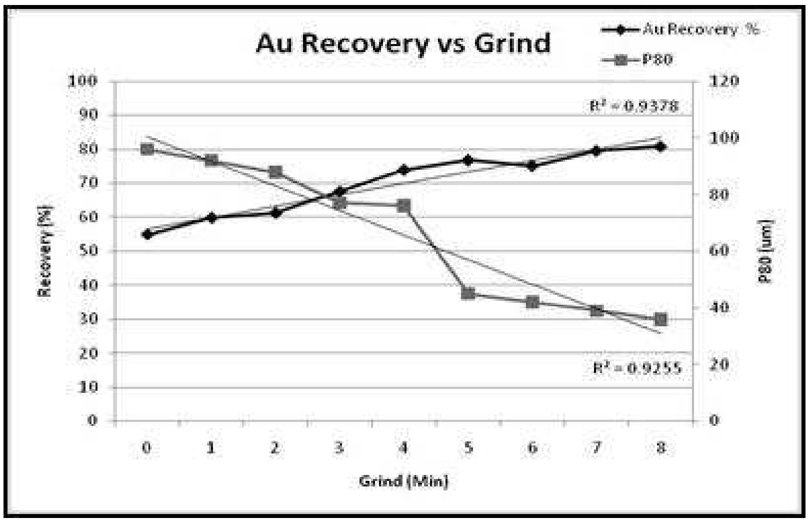

Figure 6: Gold Recovery vs. Grind Time/Size

Phase 3 - Overview

With the rather small incremental recovery increase shown in Figure 6 (from 4 minutes to 8 minutes of grinding) it was decided that the optimum recovery for the bulk flotation was targeted at a P80 of 60 µm. Consequently, five 4 kg flotation tests collected a total of 1.33 kg of concentrate, or 6.6 % mass recovery, and ultimately achieved a P80 of 53 µm.

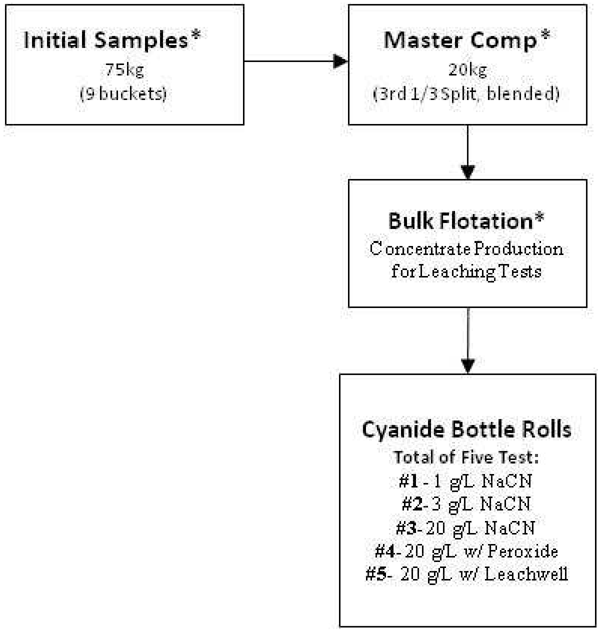

Figure 7: Phase 3

Phase 3 - Cyanide Bottle Rolls

A total of five cyanide bottle rolls were conducted under various conditions and cyanide concentrations. Each bottle roll used 200 grams of flotation concentrate which had an average gold grade of 19.2 grams/tonne and a P80 of 53 m. This represented 10.6x concentration over the original feed grade.

Table 4: Diagnostic Cyanide Bottle Rolls

Test Number	#1	#2	#3	#4	#5
CN Start - Target	1 g/L	3 g/L	20 g/L	20 g/L	20 g/L
Modifiers	- None	- None	- None	- Peroxide - Carbon	- Leachwell - Carbon
Overall Gold Recovery (Tails)	72%	83%	84%	85%	84%
Gold Recovery (Carbon)	-	-	-	59%	80%
Total CN Consumption	0.22	2.39	9.69	14.27	10.87
Total Leach Time	24 Hrs	24 Hrs	24 Hrs	27 Hrs (+3 Hrs w/ Carbon)	27 Hrs (+3 Hrs w/ Carbon)

Four of the five leaches reached their gold saturation within six hours hrs which suggests extremely fast kinetics. Supporting this is the lack of effect that peroxide and Leachwell had on the overall extraction or in the kinetic graphs.

Preg-robbing appears to be minor in the worst circumstance, natural carbon did not appear to effect the overall recoveries during the cyanide leach test. Test #4 returned a marginal enough recovery that a specific Carbon-in-Leach (CIL) test was identified as a gap in the current test program which has been recommend for follow up.

Conclusion

Given the prefeasibility level metallurgical work conducted, the obvious explanation to the recovery problems experienced by the original Carolin Mill is the larger grind size used in the previous plant (P80: 180 μm) compared to the test work mentioned in this paper (P80: 53 μm). However, future work to determine the potential of preg-robbing, if any, has been identified. This may offer more insight into the troubles experienced by the original mill.

Separately, this investigation has shown that the Carolin mine tailings are located in a well defined area. This allowed for the assessment of the project to begin at metallurgical testing. This in turn cut the time and costs required for development of the project to this point.

Additionally a preventative ecological argument can be made as a result of the flotation process employed since it captures both pyrite and arsenopyrite key contributors to acid mine drainage and arsenic toxicity in waterways.

Most importantly, the investigation of the Carolin mine tailings have returned high metallurgical recoveries which clearly warrant further economic and metallurgical pursuit. Carolin mine has shown that the typical assumption of historical tailings containing nothing of value clearly needs to be re-examining in light of the current 1800 USD/oz gold and technological advancements in processing.

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