Guidelines for Retreatment of SA gold tailings: MINTEK's learnings

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ABSTRACT

In the West Rand, East Rand and Central Gauteng region it was estimated that there are more than 500 gold mining dumps of which 350 are in the Johannesburg region, covering about 320 000 hectares of surface area. Due to the high cost of primary gold mining in South Africa and the prime real estate occupied by some of these dumps, reprocessing to recover economic value from minerals and land has generated much interest in the last number of years. There is also a growing awareness that there is a potential of realising additional benefits such as minimising problems related to pollution, health and safety such as dust (which is sometimes radioactive) and Acid Mine Drainage (AMD) potential merely by processing the dumps intelligently.

In order to facilitate the exploitation of this secondary mineral resource, MINTEK has undertaken a thorough characterisation study of these dumps to provide guidance for new dump projects over a wide range of scenarios. This study involved the collection of multiple samples from the East Rand, Central Gauteng, West Rand and Free State dumps and evaluating them with respect to metallurgical processing amenability and environmental impact. The outcome was then related to a number of possible scenarios covering not only processing options, but also specific project characteristics to provide tailored decision support to new projects.

Although the overall mineralogy of the whole Witwatersrand gold basin is uniform; the way the dumps originated and were constructed, as well as conditions to which the dumps had been exposed to over time is very variable. For example, results from the sample characterisation indicated that the tailings samples from the East Rand and Central Gauteng regions contained the lowest uranium concentrations (generally not requiring uranium removal prior to re-dumping). Tailings from the West Rand and Free State regions, however, would require both uranium and sulphide removal.

Since the typical flowsheets currently used for dump retreatment do not necessarily result in responsible final processed tailings material with low AMD potential or low radioactivity, such a process advisor may reduce future legacy costs in the country by highlighting to operators the impacts of the different processing options not only on profitability or the environment, but also on their potential liability.

1. INTRODUCTION

As the gold mines in South Africa is increasingly becoming deeper, it is becoming more expensive as well as challenging to supply the surface processing plants with enough feed material for full capacity processing. Consequently, the focus has been on extracting the economic value from the already mined and milled tailings stockpiles that are generally in close proximity to the processing plants. The benefit of reclamation projects would not only be to extract the additional economical value from these tailings, but these projects also provide a second opportunity to process the tailings more responsibly with regard to leaving behind a less toxic, more stable, smaller quantity of tailings whilst making the large areas of land available for use. In 2010 the National Empowerment Fund launched a two billion rand project towards the SA Metals Equity Project for retreatment of iron ore dump materials (Eco Partners, 2011). Following this project, reclamation of gold and uranium dumps are receiving more attention with projects currently taking place in Gauteng (Sibanye Gold, DRDGold), North West Province (Mine Waste Solutions) and Mpumalanga (Pan African Resources, Barberton Tailings Reclamation project) (Eco Partners, 2011; Creamer Media, 2015).

Currently, the government is the largest "owner" of derelict and ownerless (D&O) mines and their associated dumps with a reported number of more than 6000 ownerless mines (including dumps) that would require an estimated 30 billion rand for rehabilitation (Eco Partners, 2011; Yale Environmental, 2015; Engineering News, 1997). In the Johannesburg area alone it has been reported that about 321 square kilometre of land is covered by toxic and radioactive tailings dams (Yale Environmental, 2015).

In 2004 Rösner *et al* did a detailed study on quantifying the environmental impacts of tailings material originating from the Johannesburg region (Rössner *et al*, 1998; Rössner *et al*, 2004). The study concluded

that the tailings from this region contained large quantities of salts and elevated levels of trace elements, radionuclides and other potentially harmful substances such as cyanide. The concentrations of Co, Cr, Cu, Ni and Zn found in the gold mine tailings typically exceeded the threshold concentrations for soils. With this work it was noted that small changes in pH and Eh conditions could lead to remobilisation of contaminants.

Most of the current reclamation project flowsheets is mainly targeting high processing tonnages due to the economies of scale at the lower grade feed materials. These flowsheets also focus mainly on removing the valuable elements such as gold and silver leaving behind a tailings material that still contains uranium, and sulphide (Acid Mine Drainage potential) and other heavy metals.

In 2010 SGS published research done where a flowsheet achieved >99% oxidation and removal sulphide, >96% recovery of uranium as well as >95% gold recovery from tailings material collected from Sibanye Driefontein tailings (Flemming *et al*, 2010). This flowsheet included mild-regrind – flotation – mild acid leach – Pressure Oxidation (POX) – Resin-in-Leach (RIL)– Carbon-in-Leach (CIL) – Resin elution – uranium yellowcake precipitation.

Unfortunately such a flowsheet is complex and with the current gold and uranium price as well as the average grades in the tailings dumps, such a flowsheet would not be economically viable.

Thus the need still exists to develop a processing flowsheet that would be suitable for large as well as small scale operations to do successful tailings reclamation while producing a final tailings that does not contain toxins and acid mine drainage potential.

The objective of this study was to characterise, assess the variability of samples collected from different tailings dumps from the East Rand, Central Gauteng, West Rand and Free State dumps, and then evaluate it with respect to metallurgical processing amenability and environmental impact. The outcome was related to a number of possible scenarios covering not only processing options, but also specific project characteristics to provide tailored decision support to new projects.

The long-term objective would be to develop a modular plant design in order to support the government with reclamation of specifically, the ownerless mine dump material.

2. MATERIALS AND METHODS

2.1 Sample preparation

The gold tailings samples for the test work were sourced from various mining locations in South Africa as shown in Table 1. The samples were analysed individually in order to evaluate variability in chemical and metallurgical response.

Table 1. Tailings dump samples.

Site name	Area
Ergo	East Rand
Vlakfontein	East Rand
Aurora	East Rand
City Deep	Central Gauteng
Durban Deep	Central Gauteng
Mooi Road	Central Gauteng
Sibanye	West Rand
Cooke plant	West Rand
South Deep	West Rand
Harmony	Free State
Phoenix	Free State
Virginia	Free State

2.2 Chemical and X-Ray Diffraction (XRD) analysis

The analytical methods and the respective detection limits for the chemical and mineralogical analysis are shown in Table 2.

Table 2. List of chemical and X-Ray Diffraction (XRD) analysis.

Main section	Description	Analyte list	Determination limit
Fire assay	The standard fire assay with no high temperature cupellation followed by the dissolution of the silver prill in aqua-regia and analysis of the Au using AAS	Total Au	0.001 g/t
Inductively Coupled Plasma-Optical Plasma Spectrometry (ICP-OES)	Ores and Slags,Fusion followed by acid dissolution in HCl/HNO_3	Mg, Al, Si, Ca, Ti, V, Cr, Mn, Fe, Co, Ni, Cu, Zn & Pb	Lower limit 0.05%,Upper limit 40%
Combustion	Total S	Combustion (“LECO”)	0.01%
Combustion	Elemental S	Combustion, after extraction	0.2%
Combustion	Sulphate S	Combustion, after extraction	0.2%
Combustion	Sulphide S	Total S – Sulphate S	0.1%
Combustion	Total C	Combustion	0.01%
Combustion	Organic C	Combustion, after extraction	0.1%
Combustion	Carbonate C	Total C – Organic C	0.5%
X-Ray Diffraction (XRD)	Bulk Mineral Analysis	Crystalline substances/minerals	3%

2.3 Diagnostic leaching

Approximately 2 kg of each of the samples were subjected to the sequential diagnostic leach method employing test conditions as shown in Table 3.

Table 3. Standard diagnostic leach procedure.

Step	**Leach conditions**	**Targeted minerals**
Step 1: Direct cyanidation on Feed	30% solids, 10 kg/t NaCN, pH 11 adjusted with NaOH, 24 hour leach	Free milling gold
Step 2: Carbon-in-Leach (CIL) on Feed	30% solids, 10 kg/t NaCN, 25 g/L virgin activated carbon, pH 11 adjusted with NaOH, 24 hour leach	Preg-robbed gold identified by difference comparing direct cyanidation with CIL results
Step 3: Hydrochloric acid leach (HCl) followed by CIL on Step 2 residue	30% solids, 2M HCl for 1 hour at 70°C followed by CIL at conditions used for step 2	Gold associated with HCl digestible minerals (Pyrrhotite, calcite, dolomite, galena, goethite, calcium carbonate, could include calcine, hematite and ferrites)
Step 4: Nitric acid leach (HNO_3) followed by CIL on Step 3 residue	30% solids, 50%(v/v) HNO_3 for 1 hour at 70°C followed by CIL at conditions used for step 2	Gold associated with HNO_3 digestible minerals (Pyrite, arsenopyrite and marcasite)
Step 5: Roasting on Step 4 residue	1 hour at 900°C followed by CIL at conditions as used for step 2	Gold associated with carbonaceous material
Step 6: Calculation assuming the rest of the sample is locked in silica gangue material	Calculation	Locked in silicates

The intensive cyanidation leach tests were carried out in Schott Duran® laboratory grade borosilicate glass bottles. All chemicals used for these tests were AR grade. The bottles were placed on a custom designed (manufactured by Labotec) bottle roller which was operated at a speed of 125 rpm. The pH of the slurry was measured periodically and hydrated lime (as dry powder) was added if necessary to maintain the pH at 10.5 – 11. After 24 hours the slurry was filtered using a hardened cotton fibre, 2 µm filter (supplied by Filtech). The filter cake was washed by re-pulping with Rand Water Board water at a wet solid to water mass ratio of 1:1.5. The washed filter cake was dried in an oven at 60°C for 18 hours. The dried filter cake was analysed for total gold using gold collecting Fire Assay with a gravimetric finish. The filtrate was analysed for gold using Atomic Absorption Spectrometry (AAS) Varian 220 FS and base metals via Induced Coupled Plasma-Optical Electron Spectrometry (ICP-OES) using SPECTRO CIRO® VISION analysis. Sulphur and carbon were analysed using a LECO CS230 analyser.

For the CIL tests pre-abraded and washed, activated HayCarb carbon was added. After 24 hours, the carbon was screened out using a 500-micron hand screen and the slurry was filtered. The acid digestion steps were carried out in a 5 L 316S stainless steel reactor placed on a hotplate fitted with a Heidolph RZR 2041 overhead stirrer. The amount of acid required was calculated based on the analysis of Total C, Total S and sulphide S (1.5 times of stoichiometric requirement). After the acid digestion steps, the slurry was left to cool down and filtered. The filtrate was analysed for gold (AAS) and base metals (ICP-OES).

2.4 Gravity scouting tests

A 20 kg composite sample was passed through a 3.5-inch Knelson Concentrator, at 1.5 psi water pressure. The unit utilizes the difference in particle density between gold and gangue to facilitate separation. A slurry stream was directed into a rotor turning at sufficient speed to impart up to 300G to the material being processed. This centrifugal force magnifies the difference in particle density and the rotor geometry facilitates retention of the heavy particles, with the lower density particles rejected with the process water. Water backpressure, between the riffle rings in the top part of the rotor; allow the heavy particles to move to the concentrate bowl (retention zone). The concentrate bowl generally contains about 100 g of material (depending on density of mineral concentrated). This will thus result in a mass pull of approximately 0.5% into the Knelson concentrate if a feed mass of 20 kg is used. The Knelson concentrates and Knelson tailings material were submitted for analysis of gold via Fire Assay as explained in Table 1.

2.5 Flotation scouting tests

The flotation tests were conducted on a 4 kg sample in a 10 L flotation cell at a slurry density of approximately 33% solids (with Rand Water board water) using a standard D12 Denver® flotation cell at an impeller speed of 1500 rpm. The concentrates were collected by scraping off the froth at 15 second intervals. The flotation tests were conducted over a 20 minute period with sub-samples being removed after 0-1, 1-3, 3-7 and 7-20 minutes of flotation. All the samples from these tests were analysed for Au, total sulphur and sulphide.

2.6 Uranium with subsequent gold leaching tests

Each sample was subjected to atmospheric leaching at 20°C, 30°C, 40°C, 50°C and 60°C over a period of 24 hours. Slurry with 40% (m/m) solid content was prepared and stirred with an overhead stirrer and heated to the required operating temperature on a hotplate connected to a temperature controller. Once the operating temperature had been achieved, sulphuric acid (~800 g/L) was added until the required pH of 1.5 was reached. The redox potential remained above 450 mV throughout the leach tests; thus no MnO_2 was added into the slurries. Intermediate samples were taken at 4, 8, 12, 16 and 20 hours. The samples were filtered and the filtered residues were washed twice with acidified water (pH = 2) by water displacement (wash ratio, wash water to solids, 3 to 1) on the filter cake, and washed the third time with deionised water. The feed samples, filtrates and washed residues were analysed for uranium. The filtrates were also analysed for residual sulphuric acid, ferric and ferrous. The dry head and residue samples were pulverised and analysed via ICP-OES (Table 1). The leach filtrate samples were analysed for U_3O_8, Mg, Cr, Fe, Ca, Ti, Mn, Cu, V, Ni, Co, Si, Zn, Pb, Mo and Al via ICP-OES (Table 1).

Sub-samples of each of the uranium leach residues were subjected to direct cyanidation before and after uranium leaching at different temperatures. The bottle roll method was used at the following leach conditions at 50% solids (m/m ratio), a pH of 10.5 (corrected using hydrated lime), a NaCN addition of 1 kg/t dry solids and a leach time of 24 hours.

At the end of each leach test, the slurry pH was measured and recorded. The slurry was filtered and the residue was washed 3 times with Rand Water Board water via re-pulping. The washed residue was placed in an oven and dried for ~18 hours at 60°C. The dried leach residue was submitted for triplicate Au analysis while the filtrate was submitted for standard AA gold analysis as well as lime and cyanide titrations.

3. RESULTS AND DISCUSSIONS

3.1 Dump characteristics

In Table 4 a summary of the X-ray Diffraction (XRD) results are shown as an average of variable samples from dumps in the different areas. XRD gives an indication of the bulk mineralogy or bulk make up of each of these different areas. These results indicated that the samples from the Witwatersrand gold basin (East Rand, West rand, Central Gauteng and some samples from the Free State) do have similar bulk mineralogical properties suggesting that one kind of robust process could be used for processing of these ores. Inefficiencies in such an approach are generally accepted when the variability and grades of the dumps are considered.

Table 4. Bulk chemistry X-ray Diffraction results.

Mineral	Ideal Chemical Formula	East rand	Central Gauteng	West rand	Free state
		%	%	%	%
Quartz	SiO_2	81	76.4	71.8	70.7
Pyrrophyllite	$Al_2Si_4O_{10}(OH)_2$	12.2	13.7	14.7	16.2
Serpentine	$Fe_{2\text{-}3}Si_2O_5(OH)_4$	0.9	nd	2.2	0.6
Mica	$KAl_2(Si_3Al)O_{10}(OH,F)_2$	4.6	3.8	5.9	9.9
Gypsum	$CaSO_4 \bullet 2H_2O$	1.26	0.9	1.4	2.6
Aluminite	$Al_2(SO_4)(OH)_4 \bullet 7(H_2O)$		5.3	4.1	

Table 5. Average gold, uranium, sulphur and carbon analysis of typical dump samples.

		East rand	Central Gauteng	West rand	Free State
Au	g/t	0.27 (some with spots as high as 8 g/t)	0.32	0.35	0.41
Uranium	g/t	19	nd	59	65
Sulphide	%	0.41	0.25	0.5	0.9
Sulphate	%	0.5	0.01	0.3	0.4
Total S	%	0.9	0.28	0.8	1.05
Total C	%	0.31	0.14	0.26	0.06

The results from the chemical analysis shown in Table 5 and Table 6 correlated with that of the XRD results indicating that the samples contained aluminium-bearing minerals and the major fraction of the samples consisted of silica (quartz). Surprisingly enough very low Co, Ni, Zn and Cr were detected contrary to the previous work reported by Rösner *et al* (2004). The chemical analysis indicated that apart from the aluminium present there were no other heavy metals or elements that could be of environmental concern.

The diagnostic leach results are shown in Figure 1. An average of 58.4% gold was leached via direct cyanidation from the dump samples. Most of the samples did not display a significant amount of preg-robbing characteristics; however, it was found that leaching in the presence of carbon could increase recovery by 1-3%. The majority of the remaining gold was found to be locked within the silica matrix of these samples.

Table 6. Average chemical analysis of typical dump samples.

		East rand	Central Gauteng	West rand	Free State
Mg	%	0.5	0.5	0.65	0.25
Al	%	3.56	4.07	3.7	2.53
Si	%	29.5	40.6	35.1	42.2
Ca	%	0.24	<0.05	0.38	0.31
Ti	%	0.25	0.18	0.17	0.13
V	%	<0.05	<0.05	<0.05	<0.05
Cr	%	<0.05	<0.05	<0.05	<0.05
Mn	%	<0.05	<0.05	0.1	<0.05
Fe	%	3	2.32	3.1	1.86
Co	%	<0.05	<0.05	<0.05	<0.05
Ni	%	<0.05	<0.05	<0.05	<0.05
Cu	%	<0.05	<0.05	<0.05	<0.05
Zn	%	<0.05	<0.05	<0.05	<0.05
Pb	%	<0.05	<0.05	<0.05	<0.05
As	%	<0.05	<0.05	<0.05	<0.05

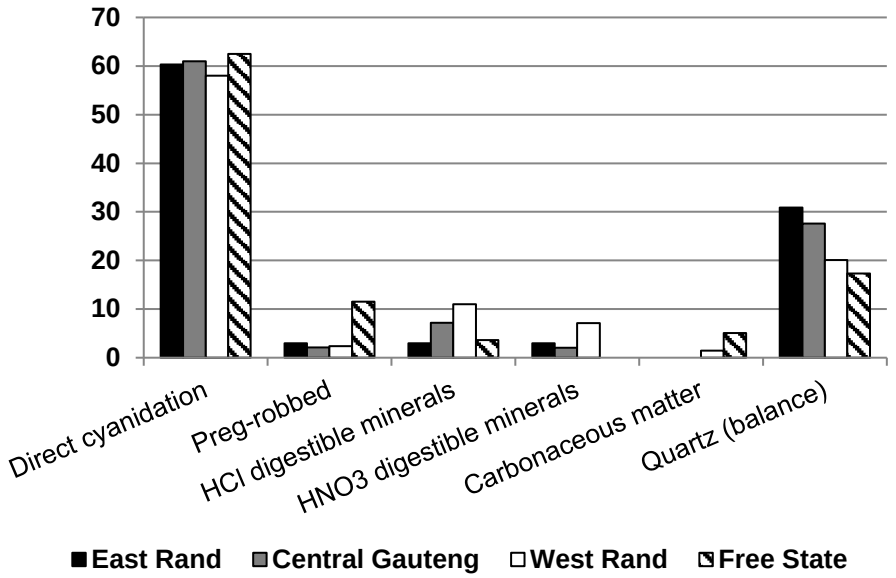


Figure 1: Diagnostic Leaching results

3.2 Typical recoveries

As shown in Table 7 the average particle size differed between the different areas with the samples collected from the East Rand being finer (80% -53 µm) and the samples from the Free State being a bit coarser (80% -106 µm).

From the gravity scouting tests it was found that gold recovery via gravity was not very successful with an average of 10% gold removed to the concentrate stream. The other disadvantage of using gravity for tailings reclamation is that there is no concentration or reduction in the stream containing the environmentally concerning elements such as uranium and sulphide.

Flotation of dump samples is usually considered for most projects as a means of avoiding “whole stream” processing. At these low gold grades the projects would normally require high amounts of sample to be treated in order to be profitable. Also, the advantage of recovering most of the sulphides and uranium into one stream for more intense processing then exist with flotation. Flotation scouting tests utilising a standard

sulphide flotation reagent suite (with no optimisation) indicated that at least ~60% of the gold and ~50% of the sulphides can be removed via flotation for all samples tested. The current accepted criteria for re-dumping is a material containing <0.3% sulphides. From these results, it was shown that via flotation the main stream (flotation tails material) would contain an expected 0.1 g/t sulphides which would meet the requirements for re-dumping.

An average of around 60% of the gold is recoverable via direct cyanidation from the samples. The majority of the gold remaining in these tails was found to be associated with quartz. Test work has shown that with additional milling (thus liberating the gold from the quartz) of the “whole ore”, or flotation concentrates (as well as the flotation tails), the gold recovery from the samples will increase compared to the samples that were not milled. If it is so simple, why not mill it all? The answer generally is related to the fact that it will take a lot of energy to mill the high volume (for example 1.2 million tonnes per month) of sample required to make a tailings project of such low grades feasible. Furthermore, the samples are already fine, and since ultra-fine milling generates a product that becomes difficult to filter and settle, water balances become complicated and cause unnecessary water retention on the subsequent slimes dams. For these reasons, although there could be economic benefit in finer grinding, some projects would not necessarily incorporate ultra-fine milling into the processing flowsheet.

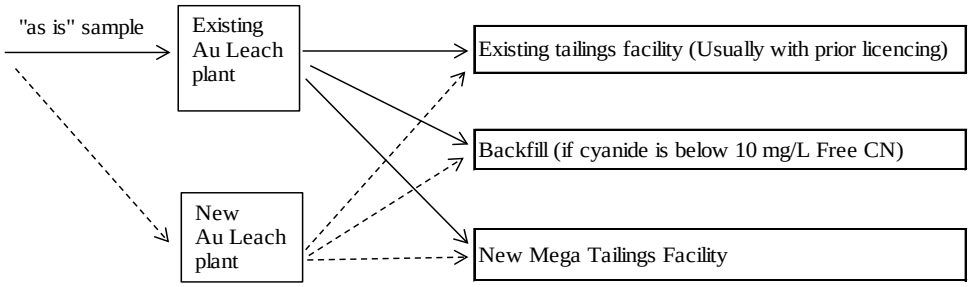
Uranium recoveries were on average 60% at atmospheric conditions at 20°C (targeting the lowest possible temperature in order to avoid high energy requirement). The tests did indicate high acid consumptions and required about 19 kg of concentrated sulphuric acid per dry tonne ore processed. MINTEK is currently in the process of doing a more detailed economical evaluation in order to evaluate alternative ways of uranium recovery. However, at the moment reclamation projects would not be driven by uranium recovery due to low economic benefit. It is advised that projects should consider uranium recovery to meet the environmental requirement of a tailings containing (as low as possible, some projects preferring <20 g/t uranium), but at least below 40 g/t uranium. As shown in Table 7 it was found that an acid leach at atmospheric conditions did achieve this target.

Table 7. Gold, uranium and sulphide recoveries as well as reagent consumptions.

		East rand	Central Gauteng	West rand	Free State
Grind size of dumps as is (average)		80%-53 µm	76%-75 µm	83%-106 µm	80%- 106 µm
Gold recovery via gravity (~1% mass pull to concentrate)	%	7-8	10-12	15	10
Gold recovery via flotation	%	73% in ~20% mass pull	65% in ~20% mass pull	56% in ~20% mass pull	70% in ~20% mass pull
Sulphide recovery via flotation	%	50	83	90	85
Sulphide in flotation tails	%	0.1	0.1	0.1	0.1
Gold recovery via direct cyanidation	%	60	61	58	63
Ave gold residue after direct cyanidation (no grinding or flotation etc)	g/t	0.18	0.19	0.16	0.25
Cyanide consumption	kg/t	0.5	0.9	1.1	0.9
Lime consumption	kg/t	2	2	4	3
Preg-robbing	%	0	3.5	4	5
Gold refractory	%	13	4.5	13	3
Gold locked in silicates	%	25	31	29	29
Uranium recovery (at 20°C atmospheric conditions)	%	75	None detected	63	57
Uranium leach acid consumption	kg/t	12	None detected	24	14
Uranium final tail grade	g/t	10	None detected	19	24

3.3 Flowsheets for larger scale operators

The following flowsheets that are presented has proven to be economically viable based on the value of the gold contained in these dump samples. All three of these flowsheets have been applied successfully in industry. Although these flowsheets were found to be economical, it still did not necessarily comply with the best way of reducing the uranium and sulphides to improve radioactive and acid mine drainage problems, thus ongoing work is required in order to investigate more responsible and more economical processing flowsheets. (Figure 2, 3 and 4)



Direct leaching will recover gold but depending on the sample will not comply with reducing the uranium and sulphides to below environmental concerning levels

Figure 2. Large scale options 1

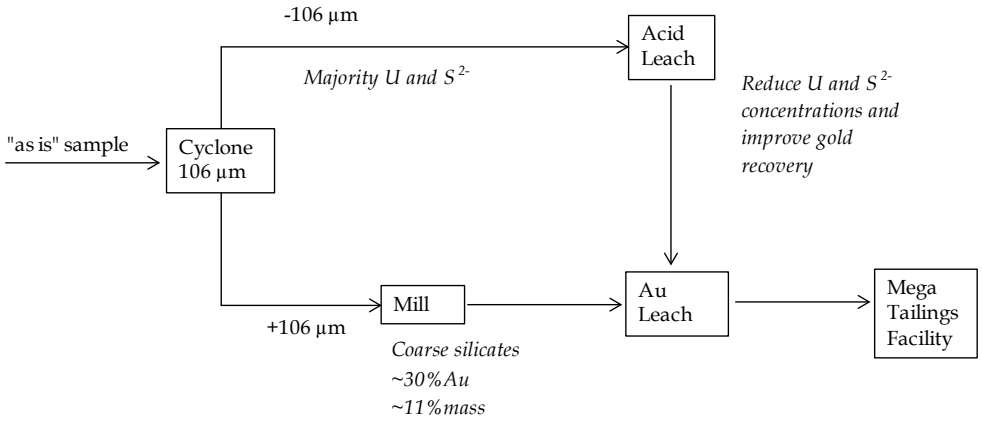


Figure 3. Large scale options 2

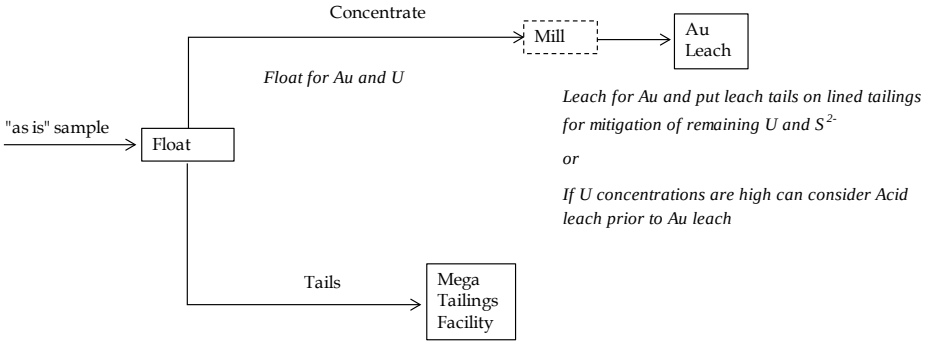


Figure 4. Large scale options 3

3.4 Flowsheets for small scale operators

The following flowsheets that are presented has proven to be economically viable based on the value of the gold contained in these dump samples for small-scale projects. (Figure 5, 6 and 7)

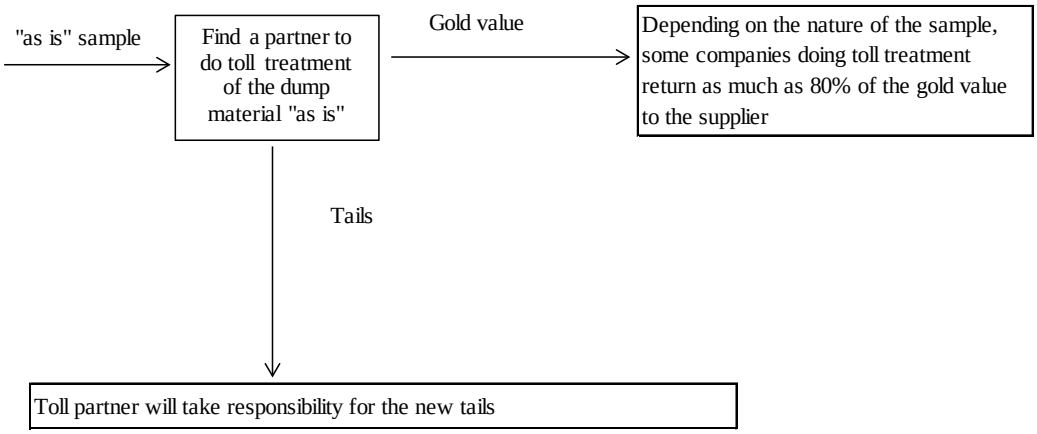


Figure 5. Small scale options 1

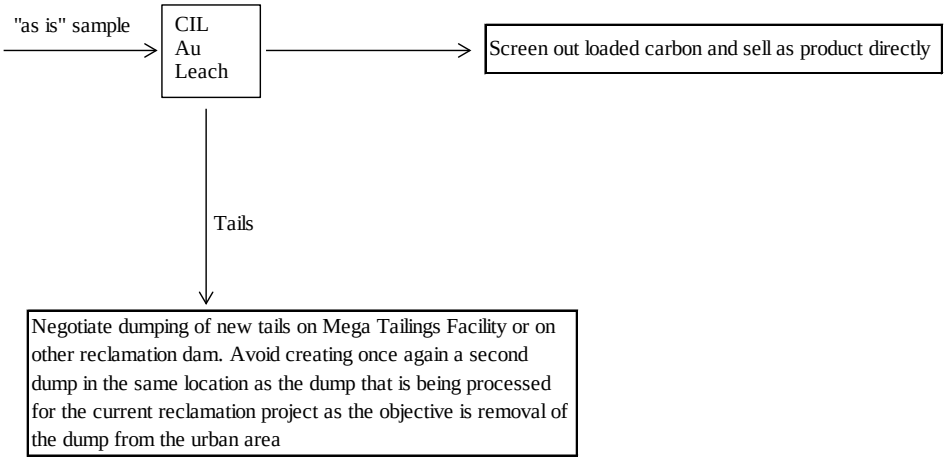


Figure 6. Small scale options 2

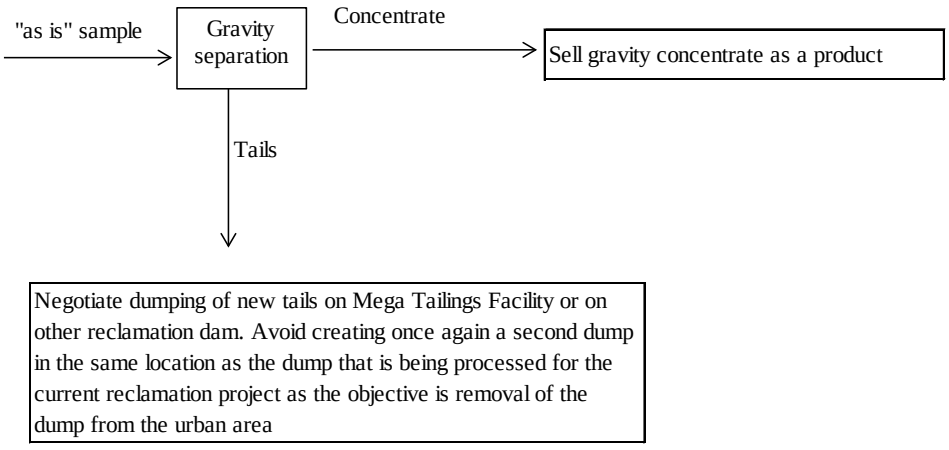


Figure 7. Small scale options 3

4. CONCLUSION

From this testwork it was found that although the tailings dump samples were variable in specific response to metallurgical processes it could be possible to use one processing plant for treating multiple dumps in different areas. Perhaps adjustments to the reagent additions could be made while treating the specific material but overall one flowsheet could be used.

A gold recovery of between 50% and 70% was achieved from the samples tested in a mass pull ranging from 10% to 20%. Sulphide recovery from these samples can be as high as 94% with an average recovery around 88% which will generally ensure a flotation tailings sample containing less than 0.1% sulphides. Thus lowering the risk of acid mine drainage (AMD) as the bulk of the sample will be deposited as flotation tailings sample. The uranium re-dumping criteria of <20 g/t uranium was mainly achieved at mild atmospheric leach conditions. Thus from this study, it was shown that it could be possible to obtain a less environmentally hazardous tailings material.

The current focus for MINTEK is defining a flowsheet that could be applied in modular scale to the ownerless and derelict mine sites specifically. Other applications for the “non-toxic” tailings are currently being evaluated such as brick making, road, and civil works material supply.

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