

ASSESSING A TWO-STAGE HEAP LEACHING PROCESS FOR PLATREEF FLOTATION CONCENTRATE

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Abstract

The Platreef is a relatively rich ore body in terms of the presence of platinum group metals (PGMs) and base metals (BMs). It differs from the Merensky and UG2 ore bodies in that a significant portion of the PGMs are associated with the silicates or on base metal sulphide-silicate grain boundaries, it has an unfavourable Pt to Pd ratio and although there is significant Au, there is much less Rh and Ru than in the Merensky or UG2 ores. Additionally, a large portion of PGMs occur as bismuth-tellurides and arsenides, which have been observed to be slow floating, requiring high mass pulls to obtain high concentrator recoveries. The high mass pulls invariably leads to grade dilution. Due to this unique mineralogy the conventional process of milling, flotation, smelting and matte pressure leaching leads to marginal process economics, with high energy and water costs, high skilled labour requirements and large capital and operating costs. Moreover, the Platreef resource lies in an arid area of South Africa, with a low skilled labour pool and poor electrical infrastructure. Additionally, a considerable amount of the PGMs will report to the tailings and will not be recoverable by flotation. A novel flowsheet to process this ore is proposed here, which represents a parallel low-cost route, along the standard method. This revolves around a two-stage heap leaching process consisting of a high temperature bioleach to extract base metals followed by a precious metals heap leach. This is currently being evaluated through preliminary bench scale test work. The results so far, which will be presented, have been promising.

Key words: heap leaching, platinum group metals, base metals

1. Introduction

The mineralisation of the Platreef ore is unique in that the platinum group metals (PGMs), unlike the Merensky and UG2 ores where they occur mainly as sulphides or alloys associated with base metal sulphides, occur largely as Pt and Pd bismuth-tellurides and arsenides while PGM sulphides and alloys make a much smaller contribution, and with the remainder being various alloys, and tin and antimony

minerals (Bushell, 2006; Shamaila and O'Connor, 2008; Vermaak, 2005). At least half the PGMs occur in association with silicate minerals, 20 % occur on the boundaries between silicate and sulphide minerals, another 20 % are associated with the base metal sulphide minerals and the rest with iron oxide, chromite and on the boundary of oxide and silicate minerals (Bryson, 2008; Bushell, 2006; Vermaak, 2005). The 4E PGM concentration in the ore is comparable to typical ores mined from other sites in the Bushveld complex, around 4-5 g/t, but after flotation the final concentrate results in a PGM grade of around 70 g/t (Bryson, 2008; Mogosetsi, 2006), well short of the 100-300 g/t concentrate normally sent to the smelter (Mogosetsi, 2006; Seymour and O'Farrelly, 2001). This is partly due to the deportment of the PGMs and due to the fact that bismuth-tellurides are slow floating in comparison to the other PGM minerals (Shamaila and O'Connor, 2008; Vermaak, 2005). Higher mass pulls can be used to obtain a higher concentrate recovery, but this leads to grade dilution and a poor concentrate material which impacts negatively on smelter operations. Additionally, milling of the ore through conventional techniques (excluding ultrafine grinding in stirred mills and high pressure grinding rolls (HPGR)) liberates only 70 % of the PGMs (Bushell, 2006; Newell, 2008). Moreover, the Platreef resource lies in an arid area of South Africa, with a low skilled labour pool and poor electrical infrastructure. Further still, there is variation of grade and hardness along the length of the reef body. As a result, processing the concentrate through the conventional method of milling, flotation, smelting and acid pressure leaching alone does not realise the full value of the ore and is therefore not an economical choice. However the substantial PGM and base metals (BMs) content, along with the relatively high gold content, justifies the investigation into alternative parallel methods.

A recent study on a low-grade PGM concentrate derived from Western Platinum Ltd (Mwase et al., 2011) revealed the effectiveness of using a two-stage heap leaching process to extract PGMs and BMs. The first stage was a bioleach utilising thermophilic microorganisms, operating at a temperature of 65°C, to extract BMs from the concentrate material. Extractions of 52% copper and 95% nickel were achieved within 4 weeks. The benefit of using thermophiles at elevated temperatures of 65-85°C to speed up the reaction kinetics of sulphide mineral bioleaching and enable the leaching of copper from chalcopyrite is well documented (Watling, 2008; Batty and Rorke, 2006; Dew et al., 2000; Rawlings and Johnson 2007; Plump et al., 2002; Stott et al., 2000, Stott et al., 2003; Norris and Owen 1993; Harvey et al., 2002). Critical to commercialising this process in heaps, is that the oxidation of the sulphide minerals is exothermic and generates enough heat to run the heap under thermophilic conditions (Dixon, 2000). The heat can be conserved in the heap by manipulating the rates of solution irrigation onto the heap and air up flow in the heap, as demonstrated by the

patented GEOLEACH™ process (Kelly et al., 2008). This method therefore allows the leaching of ore or concentrate using the low capital and operating costs of heap leaching but with the advantage of operating at a higher temperature for faster reaction kinetics (Petersen and Dixon, 2002). Additionally, this stage is required to reduce the consumption of free cyanide by the copper and nickel. These metals are easily complexed by cyanide and have the effect of taking the place of the precious metals to be leached especially in cases where dilute cyanide solutions are used (Chamberlain and Pojar, 1984). However, the extent to which they consume cyanide depends on the

mineral in which they occur (Drew, 1972). Oxides are more soluble than sulphides; but sulphides still leach slowly and can cause a temporary initial problem if not leached out before cyanidation. Further to this, base metal cyanide complexes can readily be adsorbed to carbon (Marsden and House, 2006), causing further interference in the recovery of PGMs. This interference may also extend to other forms of recovery such as zinc cementation or electrowinning (Aylmore and Muir, 2001).

In theory, the possibility of evaluating a single cyanide leach step to extract both BMs and PGMs exists, but the recovery methods of BMs from cyanide solution are not well established and wide-spread on an industrial level, although they have been explored (Gupta and Mukherjee, 1990; Habashi 1999). In contrast, several technologies, namely solvent extraction, electrowinning, sulphide precipitation and crystallisation, are well established and wide spread in recovering BMs from acidic sulphate based media.

The same study (Mwase, 2011) showed that in a subsequent heap cyanidation process on the residue at ambient temperature, PGMs could be extracted with promising results. Extractions of 20% platinum, 87% palladium and 46% rhodium were achieved, with additional base metal values extracted, within a further 3 week cyanide leach. The role of hydrogen peroxide, or a similar strong oxidant, in aiding the dissolution of PGMs was investigated, but not explored fully in the study. It is postulated that the addition of such oxidants may be beneficial to the process. The study also showed the effectiveness of elevated temperatures during the cyanidation step in significantly increasing the rate of PGMs dissolved from the ore concentrate material. In the case of percolation leaching associated with aeration, materials of construction for irrigation, aeration and solution collection and moisture loss via evaporation are foreseen to limit the temperature for practical industrial application. The authors have recently filed a provisional patent that incorporates solar heated cyanide heap leaching for PGMs, post heap bioleaching for PGMs derived from Platreef deposits (Eksteen et al., 2011).

2. Materials and Methods

2.1 Concentrate preparation

For the test work, flotation concentrate derived from a mill-float-mill-float (MF2) recovery process on Platreef ore was used which has a PGM grade of between 50-60 g/t. A 200 kg sample of this concentrate was thoroughly mixed to homogenise it and split down to 4 kg samples using a 2-way riffle splitter, then a 10-way rotary splitter. Samples for the leaching tests, size analysis, and PGM and BM solid assays were obtained from the 4 kg samples by using a Fritsch Rotary 10-way sample divider and a Quantachrome Instruments 8-way Rotary Micro Riffler. The ore concentrate was sized using wet screening showing the following size distribution:

Table 1: Size analysis results

| Screen Sizes (μm) | % Passing |
|--------------------------------|-----------|
| 75 | 97% |
| 45 | 97% |
| 38 | 83% |

A fire assay and mineral liberation analysis (MLA) revealed the following grade assay and mineral composition respectively:

Table 2: Precious metals grade assay

| Total (6E) g/t | Pt g/t | Pd g/t | Au g/t | Rh g/t | Ru g/t | Ir g/t |
|-------------------|-----------|-----------|-----------|-----------|-----------|-----------|
| ≈ 57 | 21 | 27 | 3.8 | 1.8 | 1.5 | 0.5 |

Table 3: Major base metals and gangue elements

| Cu % | Ni % | Fe % | Co % | Mg % | Al % | Ca % | Si % | Cr % | Total S % |
|---------|---------|---------|---------|---------|---------|---------|---------|---------|--------------|
| 2.3 | 3.4 | 16.4 | 0.1 | 10.4 | 1.4 | 4 | 17.3 | 0.2 | 8 |

Table 4: Major PGM minerals grouped by relative abundance

| Mineral Group | % Area |
|----------------|--------|
| Alloys | 20.8 |
| Sulphides | 28.6 |
| Arsenides | 23.8 |
| Sulpharsenides | 7.8 |
| Tellurides | 19.1 |

Among the above PGM minerals, sperrylite (PtAs_2) and cooperite (PtS) were present in by the far the largest portions of 19.6 % and 16.1 % respectively. The MLA also revealed that the PGMs were 61 % liberated.

Table 5: Major base metal and gangue minerals

| Minerals/Mineral Group | Wt % Abundance |
|------------------------|----------------|
| Chalcopyrite | 4.9 |
| Pentlandite | 7.7 |
| Pyrite | 1.9 |
| Pyrrhotite | 5.3 |
| Silicates | 66.5 |
| Others | 13.7 |

3. Experimental

3.1 High temperature heap bioleaching on concentrate: Experiment 1

Aim

The aim of this experiment was to determine the effectiveness of using a thermophilic bioleach process to extract copper and nickel from a platreef ore prior to a cyanide leach for precious metals. A low-grade flotation concentrate was used in this first test as a proxy, to obtain results in a shorter period of time than would be obtained with whole ore leaching and thus pre-determine optimal operating conditions.

Methods

Fig. 1 shows the columns used to simulate heap leaching in this experiment. Four samples of flotation concentrate weighing 650 g each were made into slurry using deionised water in a solid to liquid ratio of 5:3. The slurry was coated onto granite pebbles, packed into the columns and left over night to air dry. A solution containing 30 g/l H₂SO₄ was pumped into the columns from the top at a rate of 1 l/day (translated from an industrial flow rate of 5 l/m²/h), for 5 days in order to dissolve as much of the acid soluble BM minerals as possible, and the effluent was collected from the bottom. It was thereafter replaced with the main leaching solution containing 2 g/l Fe (1 g Fe³⁺ as ferric sulphate and 1 g Fe²⁺ as ferrous sulphate) and 10 g/l H₂SO₄, which was pumped into the column at the same rate and in the same manner. A mixed culture of thermophilic microorganisms was inoculated into the 4 columns, which were operated at temperatures of 65, 70, 75 and 80°C. Quantitative real time polymerase chain reaction (qRT PCR) was used to identify and quantify *Metallosphaera sedula* in the culture at 99% prevalence. The columns were aerated at a rate of 130 ml/min. Samples of solution were collected from the effluent at various intervals for atomic absorption spectroscopy (AAS) analysis of Cu, Ni, Co and Fe. The pH and redox potential (vs Ag/AgCl) were also measured and recorded each time a sample was taken, using a standard pH meter and redox probe, respectively. The temperatures were also monitored and recorded when samples were taken. Additionally samples were also collected for microscopic inspection, to determine the well being of the microorganisms. After 88 days of leaching, the extractions achieved were calculated both from the amounts present in solution and those determined by fire assay of the residual concentrate material.

On completion of the experiments the columns were emptied, and the concentrate was washed off the granite with water and recovered by sieving and pressure filtration. The concentrate was further washed with caustic solution to remove residual acid and then dried. From the dry samples, sub-samples were obtained for fire assays, x-ray diffraction (XRD) and MLA analyses, while the bulk of the samples were used in subsequent cyanide leaching experiments to extract the precious metals.



Figure 1: Columns used to simulate bioleaching

3.2 High temperature cyanide heap leaching

Aim

This test work was conducted to determine the effectiveness of using cyanide solution to leach PGMs from the residual material of a thermophilic bioleach process, following a high degree of extraction of the BMs.

Methods

Two samples of residual concentrate from the bioleach experiments were made into a slurry, coated onto granite pebbles and packed in columns as before. They were leached at a flow rate of 1 l/day. The columns were operated at a temperature of 50°C and aerated at a rate of 150 ml/min. The cyanide solution was recycled for 7 days after which it was exchanged with fresh solution. Samples were withdrawn at various intervals for inductively coupled plasma (ICP) analysis of precious metals, BMs and gangue elements. After 45 days of leaching the extractions achieved were determined by the metals in solution.

3.3 High temperature cyanide leaching in stirred tanks

Aim

This test work was undertaken to understand further the impact of BMs on a precious metals cyanide leach on the present ore, and compare the results to the literature on the same subject matter. There is popular consensus on the need to extract BM minerals before a cyanide leach to minimise scavenging of cyanide.

Methods

Three samples of concentrate were leached in 0.1 M cyanide solution for four days in 1 L Erlenmeyer flasks. An overhead stirrer was used to agitate the mixture and the temperature was maintained at 75°C using a water bath. Solution samples were withdrawn at various intervals over a 4 day period and filtered for ICP analysis. The three samples differed as follows:

Sample 1: A sample of fresh, untreated concentrate

Sample 2: A sample of residue from the bioleach process described above

Sample 3: A sample pre-treated with an acidic mixture of 40 g/l H₂SO₄ and 30 g/l HNO₃ in which a considerable amount of the Cu and Ni had been leached

4. Results and Discussion

High temperature bioleach

Amongst the four columns (Table 6), the 65°C column was the overall best performer contrary to the expectation that higher temperatures would produce better results. This temperature works in favour of the process, because temperatures exceeding 65°C would pose difficulties to the integrity of the materials of construction for the heap. The high density polyethylene (HDPE) used for heap pads, the plastics used for irrigation pipes, aeration pipes, liners and collection pipes would soften at temperatures above 65°C. Additionally, solution vapour losses would be substantially increased at higher temperatures. The results for all columns in Table 6 also point to the general amenability of this ore to high temperature bioleaching. Clearly the occurrence of these metals as sulphide minerals coupled with the presence of pyrite and pyrrhotite are the reasons for the success of this process. However, in addition to this, galvanic interaction between the various sulphide mineral phases may play a role (Bharathi et al., 2008). Galvanic leaching occurs when two mineral phases are in contact with each other and where one has a lower potential than the other. The mineral with the higher potential thus acts as a cathode while the one with the lower potential serves as an anode and experiences corrosion or leaching. The faster rate at which nickel leached as compared to copper (Fig. 1) can be explained thusly. Chalcopyrite in contact with pentlandite promotes faster leaching of nickel due to chalcopyrites lower potential (Bharathi et al., 2008; Mason and Rice, 2002). Chalcopyrite leaching is primarily promoted by its association with pyrite (Ekmekci and Demirel, 1997).

The lowest degree of co-leaching of the Pt and Pd were also experienced in the 65°C column. However, across all columns Rh and Ru were extracted in excess of 50 %. This is comparable to dissolution in acidic, aerated sulphate media currently experienced in a typical base metal refinery amongst some platinum producers in South Africa (Dorfling et al., 2010). Given that Rh and Ru are soluble in this acidic solution their extraction is unavoidable. However, considering the quantities extracted, recovery via methods such as solvent extraction, ion exchange and precipitation from the leach liquor should be evaluated (Kononova et al., 2011; Els et al., 2000; Bernardis et al., 2005; Seymour and O'Farrelly, 2001).

It was observed that after 43-46 days, iron precipitation occurred in all the columns following the trend in Fig. 2. It is believed that at this point the leaching of the iron sulphides was mostly complete and the source of the iron in the precipitate was mostly from the feed solution. MLA analysis on a sample of concentrate residue confirmed that the BM sulphides had been all but leached and there was hardly any elemental sulphur present. XRD and MLA analyses also confirmed that a considerable amount of the concentrate residue (35 %) was jarosite ($KFe_3(SO_4)_2(OH)_6$). The MLA analysis also indicated that the bioleaching had increased the PGM mineral liberation from 61 % to 81 %, but there was no change in the PGM mineralogy. The increased liberation can be attributed to the near complete oxidation of the sulphide minerals thereby exposing PGMs either encapsulated completely in sulphide mineral matrices or lying at the boundaries of sulphide and silicate minerals.

Table 6: Extractions of BMs over 88 days

| Columns | Temperature °C | Cu % | Ni % | Co % | Fe % |
|---------|-------------------|---------|---------|---------|---------|
| 1 | 65 | 91.1 | 98.5 | 83.5 | 38.4 |
| 2 | 70 | 65.5 | 96.9 | 86.1 | 30.4 |
| 3 | 75 | 85.4 | 98 | 82.7 | 31.2 |
| 4 | 80 | 56.8 | 93 | 76.8 | 46.5 |

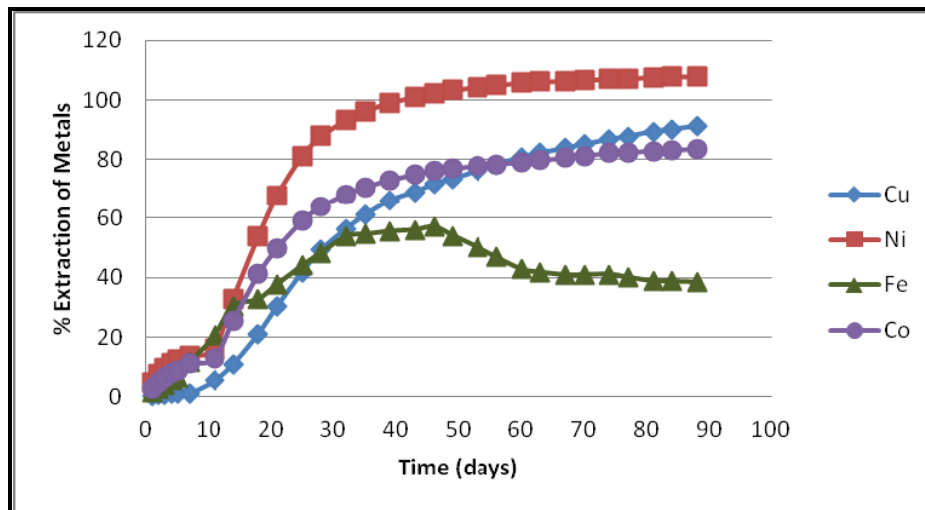


Figure 2: Extraction curves in column 1

It must be noted that the extractions from Table 6 were calculated using fire assay results from concentrate samples before and after the bioleach, while the curves in Figure 2 were drawn from ICP analyses of effluent leachate samples. There was a slight difference between the two, particularly Ni which in Figure 2 appears to have gone over 100 % extraction.

As expected in an acid leach, gangue element dissolution was high in all columns (Table 7), but of particular interest are Mg and Al. At levels of 10-12 000 mg/l these cations can inhibit ferrous oxidation (Ojumu et al., 2008). The average concentrations of

Mg and Al in this experiment were 243.6 mg/l and 38.9 mg/l respectively. However this was a once through operation with no recycle of leach solution; a full scale heap operation with recycle may have to incorporate a method to prevent build up of these cations after several cycles.

Table 7: Gangue element dissolution over 88 days

| Columns | Mg % | Si % | Cr % | Al % | Ca % |
|----------------|-----------------|-----------------|-----------------|-----------------|-----------------|
| 1 | 47.2 | 7.4 | 40.7 | 54.8 | 61.4 |
| 2 | 47 | 4.4 | 30.8 | 52.7 | 62.2 |
| 3 | 50 | 6.4 | 53.4 | 51.4 | 65.5 |
| 4 | 52 | 11.7 | 40.5 | 40.3 | 71.4 |

High temperature cyanide leach

Assays conducted on the two residual samples from the bioleach experiment, subsequently used in the follow-up cyanide leach experiment, had the following grades:

Table 8: PGM grade of residual concentrate samples

| Columns | Pt g/t | Pd g/t | Au g/t | Rh g/t | Ru g/t |
|----------------|-------------------|-------------------|-------------------|-------------------|-------------------|
| A | 15.5 | 21.5 | 1.7 | 1.1 | 0.2 |
| B | 18 | 25.3 | 1.9 | 1.7 | 0.7 |

Table 9: Base metal and gangue element grade of residual concentrate samples

| Columns | Cu % | Ni % | Co % | Fe % | Mg % | Al % | Ca % | Si % | Cr % | Total S % |
|----------------|-----------------|-----------------|-----------------|-----------------|-----------------|-----------------|-----------------|-----------------|-----------------|----------------------|
| A | 0.3 | 0.1 | <0.05 | 15.8 | 6.3 | 0.8 | 1.7 | 19.0 | 0.1 | 4.8 |
| B | 0.9 | 0.3 | <0.05 | 14.0 | 6.6 | 1.1 | 1.5 | 20.3 | 0.1 | 4.9 |

Fig. 3 shows that although severally lagging behind the complete and near complete extraction levels of Pd and Au (Table 10), the Pt extraction curves were continually increasing, at a slower rate than initially. This suggests that perhaps over a longer period of time a higher extraction could be achieved. It was observed that in the first instance it took 7 days of recycling the solution to reach maximum Pt extraction, but after that it only took 4 days of recycling the solution. In the second week it appeared that back precipitation of the Pt occurred after 4 days and in subsequent weeks the Pt extraction level stayed constant after 4 days. This may also be a case of cyanide depletion at this stage (after 4 days). However, Fig. 3 may also be an indication of a mineralogical limitation. The more cyanide soluble platinum minerals were dissolved in the first 15 days, leaving the less soluble ones behind. The refractory nature of certain gold bismuths, tellurides and arsenides to direct cyanide leaching under mild conditions of temperature (up to 60°C), atmospheric pressure and dilute reagent concentrations (2-5 g/L), is well reported (Jayasekera et al., 1996; Adams, 2005; Climo et al., 2000; Henley et al., 2001). It may be that this process is suffering a similar mineralogical limitation. Further test work and mineralogical analysis was conducted to confirm this.

Table 10: Precious metals extractions after 45 days

| Columns | Pt % | Pd % | Au % |
|---------|------|------|------|
| A | 34.3 | 96.5 | 63.4 |
| B | 32.2 | 92.5 | 97.5 |

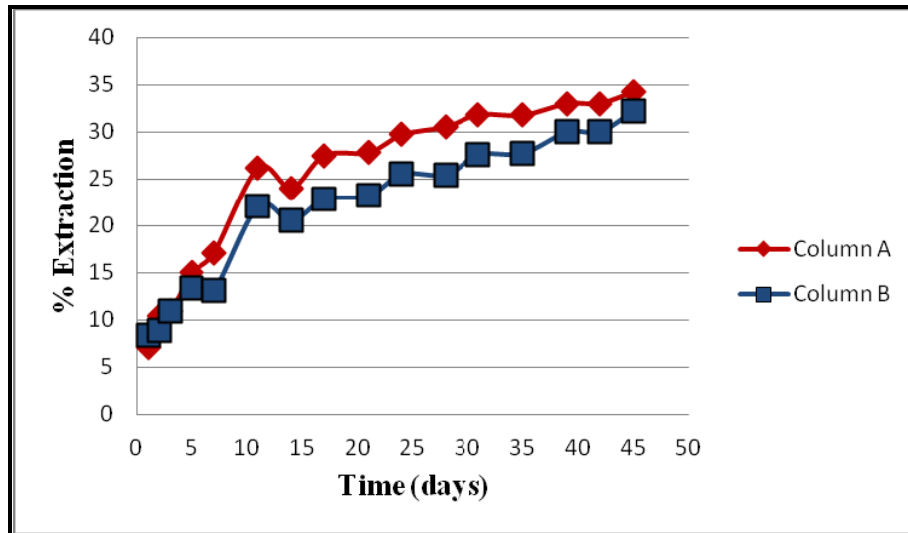


Figure 3: Platinum extraction curves

A sub-sample of 100 g of residue concentrate from Column B was leached in 0.5 M of sodium cyanide solution in an Erlenmeyer flask at a pH of 11 for 21 days. The temperature was kept constant at 75°C using a water bath. These slightly more aggressive conditions were used to obtain results in a shorter period of time and hopefully obtain maximum extraction without resorting to high pressure and temperature leaching. Of the 68 % Pt left in the concentrate the new test only achieved a further 6 % extraction. An MLA analysis on a sample of the residual material from the cyanide heap leach indicated that 78 % of the remaining Pt was in the form of sperrylite (PtAs₂) and a further 9 % were in the form of sulpharsenides. It also showed that at this stage the remaining PGMs were 91 % liberated. Looking back at Figure 3 it is hence clear that the more soluble Pt minerals leached out early, leaving sperrylite which appears refractory to cyanide leaching under conditions of low to medium temperatures. Typically gold telluride and arsenide minerals refractory to direct cyanidation are leached using high concentrations of cyanide (5-15 g/L) at high temperatures (>100°C) and pressures (>2 MPa) or by roasting before cyanidation (Adams, 2005; Marsden and House, 2006). These process parameters are not suitable in a heap leach context. Instead a different pure hydrometallurgical solution is required that can be applied to a heap leach scenario. Some avenues that are proposed for exploration include; adding hydrogen peroxide to the cyanide solution to oxidize the arsenic (Vladmir and Oran, 2006), pre-treating the concentrate with either a chloride solution (Rapson, 1997) or an alkaline sodium sulphide solution (Anderson, 2010).

Another problem encountered was the considerable presence of sulphur, which complexed with cyanide ions to form thiocyanate (SCN⁻), at levels of up to 5000 ppm. The thiocyanate was identified by HPLC and represents a considerable amount of the cyanide consumed. Based on the MLA analysis results post-bioleach it was concluded that the source of the sulphur was the jarosite as most of the sulphides had been leached and there was little or no elemental sulphur. Ciftci and Akcil (2010) report a similar effect in their test work which involved the cyanide leaching of bioleached residue concentrate. The first stage of bioleaching was intended to liberate refractory gold minerals for cyanide leaching. They observed jarosite formation in the concentrate after the bioleach and high free cyanide ligand consumption in the subsequent cyanide leach. They reported the formation of thiocyanate but attributed the high cyanide consumption to formation of ferrocyanide instead; formed from the release of iron from the jarosite. However, in this case Table 11 below shows that very little iron went into solution (ICP analysis indicated it peaked at 50 ppm) as compared to the sulphur. In this context, it is proposed that the mechanism for thiocyanate formation is similar to that observed in the decomposition of natrojarosite, argentojarosite and plumbojarosite in cyanide and other alkaline media (Roca et al., 1993; Patino et al., 1998). The jarosite decomposes releasing sulphate ions as illustrated in equation 1 (Patino et al., 1998), which react with the cyanide to form thiocyanate.



Table 11: Extraction of major BMs

| Columns | Cu % | Ni % | Fe % | Co % |
|----------------|----------------|----------------|----------------|----------------|
| A | 18.3 | 32.3 | 0 | 0.8 |
| B | 17 | 16.1 | 0 | 0 |

Although the percentage extractions of the BMs appear low (Table 11), especially considering how much copper and nickel was extracted in the bioleach, the concentration levels exceeded 100 ppm in the first 14 days. Marsden and House (2006) report that this amount of copper usually has a negative influence on gold recovery via adsorption to carbon, and it may influence PGM recovery similarly. Nickel on the other hand displays less ability to adsorb to carbon than copper so the observed amount of nickel may not cause problems during the precious metal recovery stage (Marsden and House, 2006). It appears that aerating the columns has aided the dissolution of the copper and nickel, but only for the first 14 days; after that the levels dropped to the 20-50 ppm range which does not have any negative impact thereafter. It is postulated that perhaps the initially high levels could have been due to the presence of more soluble minerals, such as oxides, leaving the less soluble sulphides thereafter (Habashi 1999). This can perhaps be remediated with a longer acid wash before commencing the bioleach process. It is, however, encouraging that the iron extraction remained low and as before this is attributed to the formation of jarosite during the bioleach process.

Table 12: extraction of major gangue elements

| Columns | Mg % | Al % | Si % | Ca % | Cr % |
|----------------|----------------|----------------|----------------|----------------|----------------|
| A | 0.01 | 0.1 | 0 | 0.1 | 0.8 |
| B | 0 | 0.2 | 0.01 | 0.1 | 1.4 |

As expected from the literature (Drew, 1972) the dissolution of gangue elements was relatively insignificant, and did not impact the leaching of precious metals (Table 12). Separate test work will determine their impact on recovery of the precious metals.

High temperature cyanide leach in stirred tanks

The percentage extractions achieved from the leach test are presented below:

Table 13: Percentage extractions

| | Pt % | Pd % | Au % | Rh % | Ru % | Cu % | Ni % | Fe % |
|-----------------|----------------|----------------|----------------|----------------|----------------|----------------|----------------|----------------|
| Sample 1 | 10.3 | 35.1 | 95 | 26.4 | 39.6 | 17.86 | 27.55 | 31.07 |
| Sample 2 | 20.4 | 36.9 | 100 | 44.6 | 54.4 | 1.45 | 3.21 | 3.21 |
| Sample 3 | 51.1 | 94.2 | 100 | 35.2 | 18.1 | 7.28 | 1.78 | 2.71 |

Sample 1: fresh untreated concentrate; Sample 2: residual concentrate from bioleach process; Sample 3: concentrate sample pre-treated with acidic mixture to extract bulk of BMs

The extraction levels of gold are not diminished by the presence of the BMs. However, removal of these elements during pre-leaching has had a positive influence on the extraction of PGMs, specifically Pt and Rh. At first glance it may seem that the pre-treatment used on sample 3 is superior to the bioleaching, resulting in substantially higher Pt and Pd extractions; but the fact is that the aggressive pre-leach for sample 3 had reduced the PGM grade by as much as half. The actual amounts of Pt and Pd extracted from the two samples were more or less identical. Table 13 also illustrates the importance of high extraction of BMs before cyanidation. This is clear from the high amounts of Cu, Ni and Fe extracted from Sample 1 in comparison to Samples 2 and 3, which would result in high consumptions of cyanide reagent.

5. Preliminary conclusion

This first set of experiments has shown that a two stage heap leach process consisting of a first stage of bioleaching using thermophiles at 65°C to extract BMs, followed by a cyanide heap leach to extract precious metals, is a potential route to accompany the standard process route for extracting PGMs from a Platreef ore. This is on condition that two obstacles are overcome; the first being the formation of large amounts of jarosite during the bioleach stage, which is the source of sulphur consuming cyanide by formation of thiocyanate in the subsequent cyanidation step. The second being the introduction of a hydrometallurgical pre-treatment stage before cyanide leaching to convert the refractory sperrylite to a cyanide soluble compound.

6. Heap bioleaching on concentrate: experiment 2

Aims

Based on the results of experiment 1, this experiment was conducted with the aim of achieving similar BM extraction levels but with less jarosite formation by manipulating feed composition; specifically the amount of iron and the pH (by adjusting the amount of acid in the feed).

Methods

The procedure, process conditions, culture of microorganisms and sampling procedures used were identical to the ones detailed in the first bioleach experiment, with the exception that a temperature of 65°C was used in all columns and the feed composition across the columns varied as follows:

Column 1: Was a repeat of column 1 from experiment 1, the feed solution consisted of 2 g/l Fe (1 g ferrous and 1 g ferric) and 10 g/L H₂SO₄. After 43-46 days the experiment was stopped to see, by XRD analysis, if any jarosite had formed. Also important to note is that in the previous experiment most of the Cu, Ni and Co had been leached at this point. The sample would then be leached with cyanide in a stirred tank reactor to observe the levels of sulphur as well as the precious metal extraction levels. The results would be compared with the first set of stirred tank tests conducted on the residue from the bioleach column run under similar conditions. The hypothesis in this case is that if all oxides and significant copper is extracted, the remaining sulphides may not interfere significantly in the cyanide leach.

Column 2 and 3: Were run with 0.5 g/l ferrous, no ferric; with column 2 having 10 g/l H₂SO₄ and column 3 having 20 g/l H₂SO₄. These tests reduce the amount of iron right at the beginning in case jarosite formation starts earlier in the experiment. The higher acidity in column 3 was a contingency against iron precipitation.

Column 4: Was run with the standard feed solution of 2 g/l Fe (1 g ferrous and 1 g ferric) and 10 g/L H₂SO₄ for 43-46 days after which the solution was changed to 0.5 g/l ferrous, no ferric and 20 g/l H₂SO₄. This was to determine if the remaining base metals (Cu and Co) can be completely extracted without forming jarosite after 40 days.

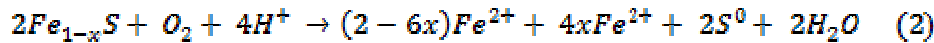
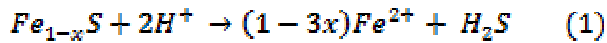
6.1 Results and discussion

ICP analysis of leachate samples and fire assays of the residual concentrate material were used to calculate percentage extractions which were as follows:

Table 14: Percentage extractions achieved

| Columns | Duration days | Cu % | Ni % | Co % | Fe % |
|------------------|------------------|---------|---------|---------|---------|
| 1 (experiment 1) | 88 | 91.1 | 98.5 | 83.5 | 38.4 |
| 1 | 40 | 80 | 92.8 | 93.7 | 46 |
| 2 | 89 | 90.1 | 98.6 | 97.5 | 71 |
| 3 | 89 | 93.1 | 99.1 | 98.6 | 90.9 |
| 4 | 89 | 93.7 | 98.9 | 98.7 | 86.3 |

As per the plan, Table 14 shows the extractions in columns 2-4 match those achieved in experiment 1. This is particularly interesting in the case of columns 2 and 3 which had much less Fe in the feed. It is clear that the Fe^{2+} released from the pyrite and more particularly the pyrrhotite is enough to sustain the bacterial metabolic growth. As stated earlier, galvanic interactions are likely to be occurring between inter-grown sulphide grains, in this case between pentlandite and pyrrhotite. Lu et al. (2000) showed that under acidic conditions pentlandite has a higher potential than pyrrhotite, driving the latter to be dissolved faster. Santos et al. (2006) proposed the following reactions for pyrrhotite dissolution:



Eqs. (1) and (2) show the dissolution of pyrrhotite producing Fe^{3+} which acts as an oxidant for sulphide minerals and Fe^{2+} which is oxidised by bacteria to produce Fe^{3+} for the leaching process. However the main aim was the reduction of jarosite and sulphur compounds. XRD analysis and LECO combustion test showed the following results:

Table 15: XRD Results

| Columns | Duration of leach days | Jarosite % | Total Sulphur % | Elemental Sulphur % |
|------------------|------------------------------|---------------|-----------------------|---------------------------|
| 1 (experiment 1) | 88 | 35.4 | 4.4 | 0 |
| 1 | 40 | 11.9 | 5.6 | 3.3 |
| 2 | 89 | 16.2 | 2.6 | 0.4 |
| 3 | 89 | 21 | 4.1 | 1.6 |
| 4 | 89 | 18.1 | 3.1 | 0.8 |

Table 15 shows that overall Column 2 has the least amount of sulphur compounds. Additionally Column 1 shows that formation of jarosite is unavoidable. The ICP analysis of the effluent leachate samples showed that Column 1 of this experiment was a successful repeat of Column 1 from experiment 1. The extractions for the BMs and the extraction trends were similar at the 40 day mark. It is therefore concluded that if allowed to go beyond 40 days, the jarosite formation in Column 1 would have eventually reached the levels that were attained in the first experiment. Fig. 4 shows the Fe extraction curves calculated and drawn using data from the analysis of the effluent

leachate samples from the columns. These amounts were calculated based on the Fe in solution minus the initial amounts in the feed. Even though the curves for columns 2-4 show near complete leaching of the Fe, the XRD analysis and fire assays confirm the presence of Fe in the residual material (mainly in the form of the jarosite) confirming that some jarosite formation is unavoidable.

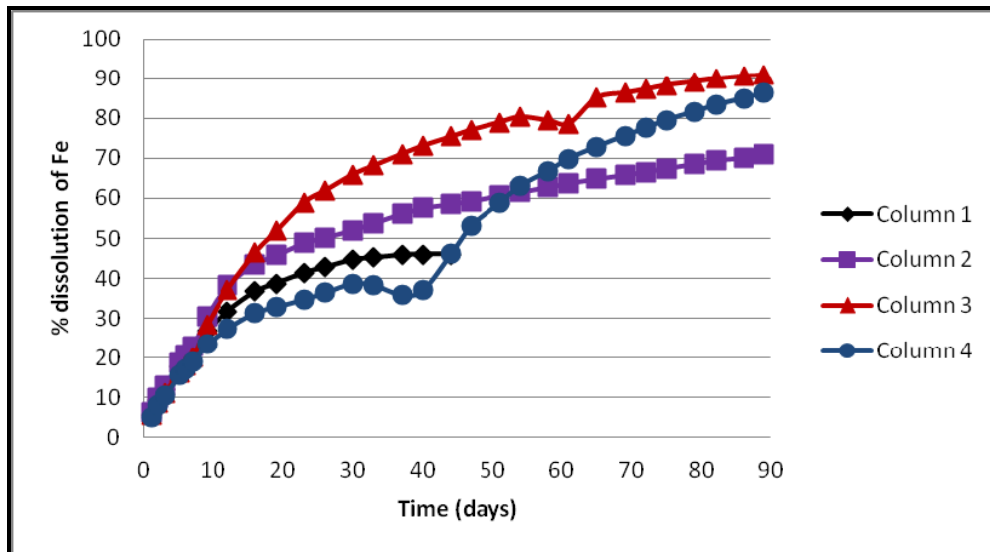


Figure 4: Fe extraction curves

6.2 Conclusion

Jarosite formation in this heap bioleach process cannot be avoided entirely but can be minimised by using less iron in the feed. In a full scale operation perhaps no Fe would be required in the feed as at a large scale the Fe leached from the sulphides would be enough fuel for the process.

7. Final Conclusion

A two stage heap leach process in which a bioleach first extracts BMs followed by a cyanide leach to extract precious metals is a potential method to accompany the conventional method of processing PGMs, to achieve full value of the Platreef ore considered here. An initial acid wash and bioleach to extract high levels of BMs has shown to be beneficial to Pt and Rh leaching. However, this is on condition that a hydrometallurgical process to convert the sperrylite to a cyanide soluble form can be developed to precede the cyanide leach. This will be the focus of future work on this project as well as an in-depth discussion on the role of the mineralogy in this process and testing this process on coarse crushed ore particles. These aspects will be explored in future papers in this project. Key to this research work and its industrial application is the use of high temperatures (50-70°C), from a heap leach applications perspective, for both the base and precious metals extraction. The authors (Eksteen et al., 2011) show how this can be practically achieved using solar heating and potential heat integration between the heap bioleach and heap cyanide leach operations (Figure 8), as the heap bioleach (BHL) is quite exothermic and the cyanide heap leaching operation (CHL) is

roughly energy-neutral, except for environmental and evaporative losses. Solar heating, either directly for alkaline cyanide solutions, or indirectly via intermediate heat exchange provides environmentally friendly ways to heat the solution and obtain acceptable extraction kinetics for the PGMs and Au. It matches a low intensity heating mechanism to a low intensity leaching process with a large thermal inventory. As the barren leached heaps are to be considered for mine backfill, the heaps are foreseen to be temporary heaps with loading and unloading onto permanent well-engineered leach pads.

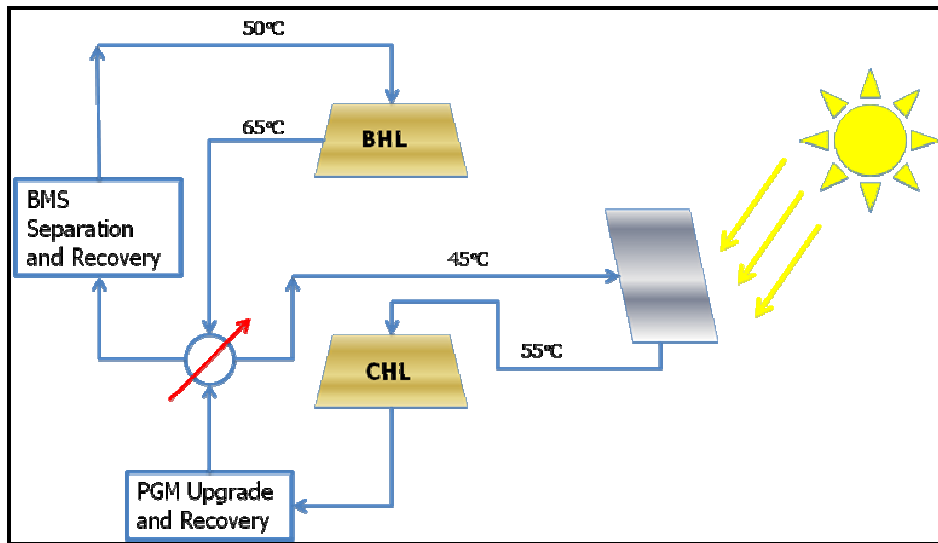


Figure 8: Heat management in heap bioleach and cyanide heap leach (Eksteen et al., 2011)

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[IEYHDhAeBvfDMBQ&sa=X&oi=book_result&ct=result&resnum=1&ved=0CByQ6AEwAA#v=onepage&q=silicate%20minerals%20cyanide%20dissolution&f=false](http://books.google.co.za/books?id=IEYHDhAeBvfDMBQ&sa=X&oi=book_result&ct=result&resnum=1&ved=0CByQ6AEwAA#v=onepage&q=silicate%20minerals%20cyanide%20dissolution&f=false)

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