

Mineralogy and uranium leaching response of low grade South African ores

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Received 8 May 2007; accepted 28 June 2007

Available online 8 August 2007

Abstract

The efficiency of uranium leaching determines the economic viability of treating low grade uranium deposits, and is quite sensitive to ore characteristics. The interrelationship between mineralogy, mineral liberation and the leaching behaviour of uranium is not well defined. Uraninite's leaching kinetics are well studied, but relatively little leaching research has been conducted for other uranium minerals.

Dissolutions higher than 90% are very difficult to achieve under the normal operating conditions employed for acid leaching of South African ores. In this paper, a mineralogy-leachability explanation is presented to rationalise the difficulty in exceeding 90% dissolution from low grade uranium ores in the Vaal River region (averaging 0.3 kg U₃O₈/t). Based on the findings, further discussed in the paper, it appears that brannerite's intrinsic inertness is responsible for not obtaining optimum recoveries.

It is also shown that for low grade uranium ores in the Witwatersrand context, the real value of uranium leaching could lie in the unlocking of extra gold (0.4–0.6 g/t) by the uranium leaching process that typically precedes cyanide leaching of gold.

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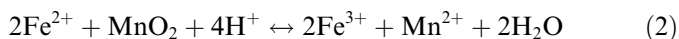
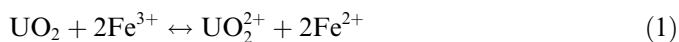
Keywords: Hydrometallurgy; Leaching

1. Introduction

Gold deposits in the Vaal River region, South Africa, also contain uranium-bearing minerals. Although the uranium content of the ore is very low by international standards (it averages only 0.3 kg U₃O₈/t), recovering uranium as a by-product of gold production improves the overall processing economics. The benefit is primarily due to the additional gold extraction following the acid pre-treatment of the ore during uranium leaching. However, the current high uranium price and increasing

demand due to renewed interest in nuclear power also add significantly to the benefit by making uranium production economic on its own.

Leaching as practiced in South African plants is accomplished by oxidation of the insoluble U⁴⁺ form to the acid-soluble U⁶⁺ form in an acidic environment (H₂SO₄), using ferric as an oxidant. To maintain reasonable reaction rates, depleted Fe²⁺ is continually converted to Fe³⁺ using pyrolusite (MnO₂-rich ore), i.e.:



The ferric/ferrous couple serves as an electron transfer catalyst between the solid oxidant (MnO₂) and UO₂. The electrochemical nature of the uranium leaching process means it is confined to a specific Eh–pH window (see Fig. 1). In sulphate systems, U⁶⁺ exists as UO₂(SO₄)_n^{2n–2} and not the

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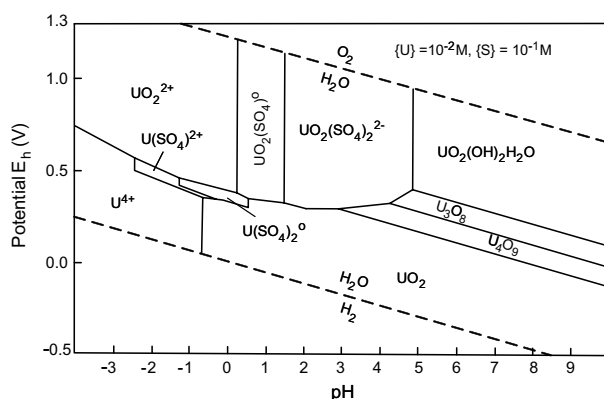


Fig. 1. Eh/pH diagram of a U–S–H₂O system @ 25 °C (Hayes, 2003).

uncomplexed UO_2^{2+} species. The efficiency of uranium recovery is influenced by the mineralogical characteristics of the ore. In particular, bulk composition affects reagent consumption and ferric generation, while uranium mineral composition and mode of occurrence influences uranium dissolution.

The uranium leaching response of three different ore types from the Vaal River area (further referred to as ores A, B and C) during atmospheric sulphuric acid leaching was investigated. The ores were studied in terms of their leaching responses to variations in acid addition, solid oxidant dosage, and temperature. The ultimate objective was to relate the leach responses to the ore mineralogies.

In the recovering of uranium as a by-product of gold production, two process routes can be considered with respect to gold recovery: (i) forward leaching (direct cyanide leaching of gold) and (ii) reverse leaching (sulphuric acid leaching of uranium followed by cyanide leaching of gold). It is generally accepted that leaching uranium before gold liberates more gold, improving gold recovery. Tests were done to quantify the benefit for the Vaal River ores.

2. Experimental procedure

2.1. Mineralogical analyses

The mineralogical analyses were done on a mineral liberation analyzer (MLA) using polished section of the material investigated. The detailed uranium mineral evaluation uses an automated sparse phase liberation measurement to identify and characterize the grains of the mineral of interest. The particle detected is then analyzed using energy dispersive X-ray analyses.

2.2. Uranium leaching tests

A factorial design setup (2^3 design) was proposed for testing the dependence of the percentage uranium extraction on individual parameters. Key parameters investigated in the uranium leaching experiments were sulphuric acid dosage (9.9 kg/t, 12.8 kg/t, 16.3 kg/t), temperature (40 °C,

50 °C, 60 °C) and MnO_2 addition (2 kg/t, 3 kg/t, 4 kg/t – as 100% MnO_2). The ore samples were rod-milled to a P_{80} of $-75 \mu\text{m}$. Leaching experiments were carried out in 2 L water-jacketed batch reactors, which were mechanically agitated. The sulphuric acid was added at the beginning of each experiment. A H_2SO_4 solution concentration of 647 g/L was used. Pyrolusite containing 29.3% MnO_2 and 36.5% Fe was used as a solid oxidant and added 1.5 h after the addition of the acid. The absolute dosages were calculated on the basis of a pulp RD of 1.55 and a solids SG of 2.7. Stirring speed was kept constant at 6 rpm.

2.3. Forward and reverse leaching tests

The gold leaching experiments were carried out in 2 L water-jacketed batch reactors which were mechanically agitated. A pulp mixture consisting of 1310 g of the dried milled sample (P_{80} of $-75 \mu\text{m}$) and 1015 mL water was pre-heated to 30 °C. The pulp was agitated at 6 rpm to fully suspend the solids. The pH of the pulp was adjusted by addition of small amounts of lime, typical 0.1 g at a time using a spatula. The initial pH was set to ~ 10.5 . Reagent concentrations used were as follows: 40 g/L NaCN and 20 g/L CaOH. Before NaCN was added to the reactor, air was manually sparged. The air sparger was assembled with the baffle and left in the reactor till the end of the test. The test commenced after the addition of 0.65 g NaCN ($\sim 347 \text{ mg/L CN}^-$).

The reverse leaching experiments included sulphuric acid leaching of uranium followed by cyanide leaching of gold. The experimental methods are described in Sections 2.2 and 2.3.

3. Results and discussion

3.1. Uranium mineralogy

Like gold, uranium minerals are concentrated in a matrix of pebble-supported conglomerates (Smit, 1984). The bulk mineralogies of the three different ores investigated are fairly similar and consist primarily of quartz (70–80%), with lesser amounts of muscovite (8–11%). From Table 1 it can be seen that samples of ore A are slightly different, and contain less quartz, pyrite and chlorite; and more pyrophyllite, as compared to the other ores.

The uranium concentrations in the different ores also varied within the following ranges: ore A (270–330 ppm), ore B (290–450 ppm) and ore C (390–540 ppm). A detailed uranium mineralogical characterization of the ore was conducted and the uranium mineral distributions are shown in Table 2.

As expected from earlier studies by Smit (1984), it was found that uraninite as well as brannerite-type minerals ($\text{U}_{1-x}\text{Ti}_{2+x}\text{O}_6$) are jointly responsible for the major portion of uranium carriers in ore from the Witwatersrand basin. Table 2 shows that 80–90% of the uranium in the ores is

Table 1
Bulk mineralogy of the three Vaal River ores

Mineral	Formula	Ore A (%)	Ore B (%)	Ore C (%)
Quartz	SiO ₂	70.2	73.3	79.8
Muscovite	K–Al-silicate	10.1	11.3	8.2
Chlorite	Mg–Fe–Al-silicate	2.0	3.2	3.6
Pyrophyllite	Al ₄ (Si ₈ O ₂₀)(OH) ₄	9.7	2.5	1.1
Pyrite	FeS ₂	1.3	2.5	2.8
Albite	Na–Al-silicate	4.8	5.1	1.9

Table 2
Uranium mineral distribution (area %) of feed samples

Mineral	Formula	Ore A (%)	Ore B (%)	Ore C (%)
Uraninite	UO ₂	47.6 (84.9)	42.1 (79.7)	52.8 (89.2)
Brannerite	(U,Th,Ca)(Ti,Fe) ₂ O ₆	42.2 (12.9)	49.6 (18.5)	32.3 (7.7)
U-Phosphate	(U, Cl)PO ₄	3.1 (0.2)	2.5 (0.1)	5.7 (0.4)
Coffinite	U(SiO) _{41-x} (OH) _{4x}	7.1 (2.0)	5.8 (1.7)	9.2 (2.7)

Values in brackets represent uranium metal deportment, in %, calculated from ideal mineral uranium contents, assuming spherical shapes to convert area to volume and then to mass of mineral using their ideal densities. There is no 'standard' U-phosphate mineral, so its uranium concentration was assumed to be the same as that of brannerite and its density that of apatite. Densities used were 10.88, 5.2, 3.19 and 5.44 g/cm³ for uraninite, brannerite, apatite and coffinite, respectively. Uranium contents used were 88.15, 33.54, and 72.63% for uraninite, brannerite, and coffinite, respectively (data mainly from www.webmineral.com).

contained as uraninite, 8–19% as brannerite, and the balance as traces of coffinite and uranium phosphates.

Uraninite dissolves readily in the presence of a lixiviant provided that the required conditions of extraction are met. Brannerite-type minerals, unlike uraninite, are not readily leachable in sulphuric acid and therefore are referred to as refractory. Liebenberg (1955) distinguished between two uraniferous titanates in Witwatersrand ore: uraniferous leucoxene and brannerite which have UO₂:TiO₂ mole ratios of <1 and >1, respectively. One would expect variability in the leaching response amongst brannerite-type minerals, but at this stage of the study different brannerite-types were not investigated. Previous work done by Glatthaar and Duchovny (1979) indicated that Vaal River ores mostly consist of brannerite associated with leucoxene and other titaniferous minerals (termed uraniferous leucoxene) which have a more loosely knit appearance and probably are more readily available for dissolution as compared to brannerite associated with silicates (termed brannerite), which occurs as minute, compact crystals intergrown in the siliceous material. This however, is not indicative that the different types of brannerite minerals will dissolve.

Small amounts of coffinite were present. Coffinite is generally more reactive than brannerite to oxidative sulphuric acid leaching, but less reactive as compared to uraninite. In ore from the Elliot Lake district, which also consists mostly of brannerite and uraninite, secondary coffinite intergrowths are enhancing the overall kinetics of brannerite by accelerating leaching pit formation (Ifill et al., 1996). Further, in the ore samples investigated there were also

traces of uranium associated with monazite, which may be inert or reactive, depending on whether uranium is a substitutional impurity or is adsorptively associated with monazite. Ford and Gould (1994) found that the amount of inert uranium is, in absolute terms, fairly similar for all Witwatersrand ores, at about 0.015 kg/t to 0.030 kg/t, suggesting that there is always an amount which is very inert.

The mineralogy investigations focused on uraninite and brannerite, as they form the bulk of the uranium-bearing minerals. Table 3 shows the uranium mineral associations. Most of the unliberated uraninite is associated with silicates, carbon, or has a ternary association. Ore C also has a high percentage uraninite associated with phosphates. Association with carbon is quite low for brannerite.

Uranium grain sizes were found to be very small, with 50% of the particles passing 19.4, 21.3 and 23.2 µm for ore A, ore B and ore C, respectively. The degree of liberation of the uranium-bearing minerals was low (see Table 3), between 11% and 45%, and expectedly increased as particle size decreased. It is important to realize that surface area exposure may be a more useful indicator of leachability of uranium minerals, as opposed to intrinsic liberation. Minerals with exposed surface area are technically leachable as they can be accessed by a lixiviant. Tables 4a and 4b show that between 87% and 93% of the uraninite particles and 71% to 86% of the brannerite particles have more than 10% of their surfaces exposed, and even higher proportions have more than 5% of their surfaces exposed.

Fig. 2 shows the distinction between liberation and surface exposure. It is suggested that the measurement of area exposure of the uranium minerals (especially the fraction with >10% of surface exposed) is a very good indicator of their leachability, because only intrinsic inertness to leaching reagents can cause an exposed mineral to remain unleached.

Uranium minerals seem to have a high percentage surface area exposure, despite poor liberation. This suggests that the breakage of the ore particles occurs near the uranium grains. It needs to be established whether this is due to association with soft gangue components.

Table 3
Brannerite and uraninite associations

	Brannerite			Uraninite		
	Ore A	Ore B	Ore C	Ore A	Ore B	Ore C
Binary association	67.8	85.8	67.7	86.0	85.2	86.6
Liberated	11.8	11.1	13.4	30.6	30.9	42.3
Uranium minerals	1.7	7.5	15.7	11.8	7.1	6.5
Silicates	46.8	59.0	28.7	27.4	20.6	24.0
BMS	1.0	2.1	4.1	0.3	0.5	0.7
Oxides	3.6	4.0	4.6	0.1	0.1	0.7
REE-Phosphate	0.0	0.0	0.2	0.0	1.2	8.0
Carbonate	0.0	0.0	0.0	0.0	0.0	0.0
Carbon	2.9	2.1	1.0	15.8	24.8	4.3
Ternary association	32.2	14.2	32.3	14.0	14.8	13.4
Total	100	100	100	100	100	100

Table 4a
% Uraninite liberation and % of uraninite with 5 and 10% surface exposure

	Ore A (%)	Ore B (%)	Ore C (%)
Liberated	30.6	30.9	42.3
Middlings	26.1	31.6	31.2
Locked	43.3	37.5	26.5
Total	100	100	100
5% Surface exposure	96.4	96.4	98.3
10% Surface exposure	88.9	87.4	93.2

Table 4b
% Brannerite liberation and % of brannerite with 5 and 10% surface exposure

	Ore A (%)	Ore B (%)	Ore C (%)
Liberated	11.8	11.1	13.4
Middlings	38.7	53.4	29.6
Locked	49.5	35.5	57.0
Total	100	100	100
5% Surface exposure	93.9	94.2	87.8
10% Surface exposure	79.3	86.1	71.3

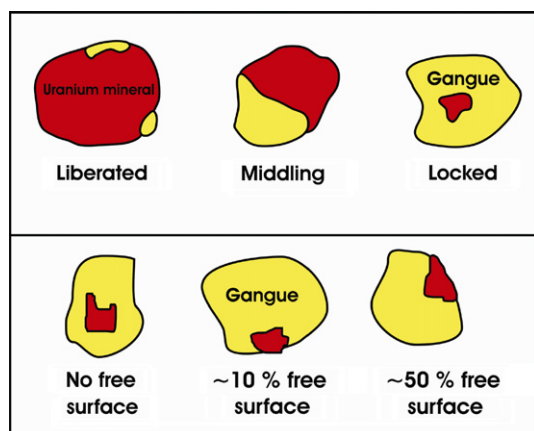


Fig. 2. Distinction between liberation and surface exposure.

This means that coarser grinds can be tolerated for uranium leaching than one would suspect from their grain sizes alone.

3.2. Uranium dissolution

Standard leaching tests done showed that the maximum uranium recovery within the chosen parameter space never exceeded 90%, for a 24 h leaching period. A summary is presented in Tables 5a, 5b and 5c. The independent estimate of error (based on centre-point triplicates; temp: 50 °C, MnO₂: 3 kg/t, H₂SO₄: 12.8 kg/t) determined for ore A, ore B and ore C for 24 h are 1.5, 0.4 and 2% points, respectively.

The dissolution curve as well as the potential profile, for the aggressive leaching conditions, over a 24 h period is presented in Figs. 3 and 4. Note that the increase in potential after 1.5 h is due to the addition of the oxidant.

Acid dosage was found to have the greatest effect on uranium dissolution. The insensitivity of the uranium dissolution to temperature suggested mass-transfer (either internal or external) control, but since there are a lot of variables their interaction has to be understood before making definite conclusions, especially because the temperature range tested is relatively narrow (20 °C).

There can be various reasons for not achieving dissolutions higher than 90%. Since acid concentration is the most influential parameter, and to ensure that the uranium dissolution reaction was not acid limited, acid dosage was increased to 25 kg/t. However, there was no significant improvement beyond 90% (see Table 6).

Although there is still uncertainty regarding the dissolution of brannerite, some of the brannerite must dissolve in order to achieve ~90% total uranium dissolution, because it cannot be accounted for by uraninite dissolution alone (refer to Table 2). Mineralogical analysis (Table 7) showed that uraninite has been mostly dissolved (only small amounts remained in the residue) and that the largest fraction of unleached uranium present in the residue exists as brannerite.

Table 5a
Percentage uranium dissolution after 24 h for ore A. Acid consumption is included in kg/t in brackets

	High temp (60 °C)	High temp (60 °C)	Low temp (40 °C)	Low temp (40 °C)
High Acid (16.3 kg/t)	90.3 (11.2)	86.6 (13.3)	85.2 (12.8)	87.4 (10.8)
Low Acid (9.9 kg/t)	63.8 (9.2)	68.3 (8.3)	63.8 (8.7)	68.0 (8.0)
	High MnO ₂ (4 kg/t)	Low MnO ₂ (2 kg/t)	High MnO ₂ (4 kg/t)	Low MnO ₂ (2 kg/t)

Table 5b
Percentage uranium dissolution after 24 h for ore B. Acid consumption is included in kg/t in brackets

	High temp (60 °C)	High temp (60 °C)	Low temp (40 °C)	Low temp (40 °C)
High Acid (16.3 kg/t)	83.5 (14.8)	86.2 (14.2)	86.7 (11.9)	83.1 (11.6)
Low Acid (9.9 kg/t)	71.8 (9.6)	73.7 (9.2)	76.5 (9.0)	69.8 (8.6)
	High MnO ₂ (4 kg/t)	Low MnO ₂ (2 kg/t)	High MnO ₂ (4 kg/t)	Low MnO ₂ (2 kg/t)

Table 5c

Percentage uranium dissolution after 24 h for ore C. Acid consumption is included in kg/t in brackets

	High temp (60 °C)	High temp (60 °C)	Low temp (40 °C)	Low temp (40 °C)
High Acid (16.3 kg/t)	80.5 (15.9)	66.7 (15.4)	80.0 (13.7)	79.8 (12.6)
Low Acid (9.9 kg/t)	67.9 (9.6)	70.9 (9.4)	70.3 (9.3)	70.9 (8.9)
	High MnO ₂ (4 kg/t)	Low MnO ₂ (2 kg/t)	High MnO ₂ (4 kg/t)	Low MnO ₂ (2 kg/t)

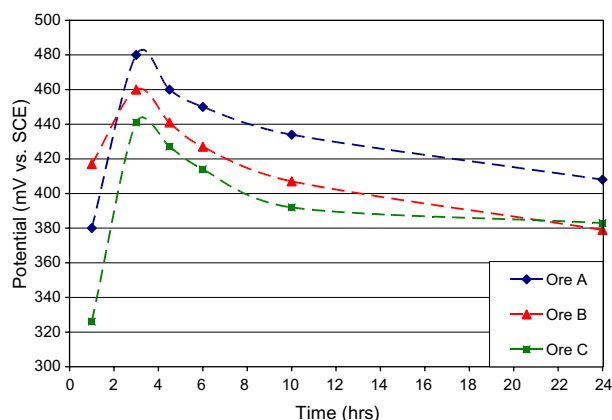
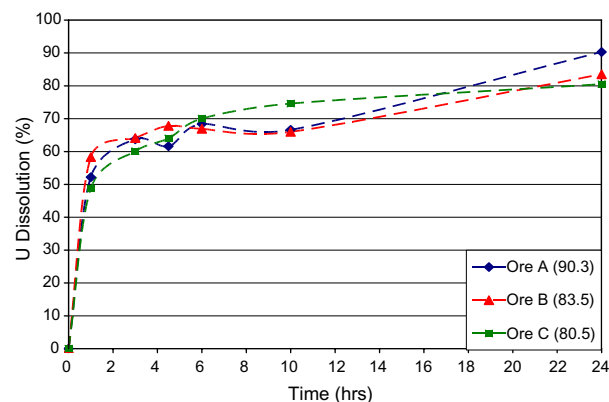
Fig. 3. Potential profile for aggressive leaching conditions (H₂SO₄: 16.3 kg/t, MnO₂: 4 kg/t, temp: 60 °C).Fig. 4. Uranium dissolution versus time for aggressive leaching conditions (H₂SO₄: 16.3 kg/t, MnO₂: 4 kg/t, temp: 60 °C) End dissolutions are included in brackets.

Table 6

Comparison of uranium dissolutions between aggressive (Acid: 16.3 kg/t) and most aggressive (Acid: 25 kg/t) conditions at 60 °C and MnO₂ addition of 4 kg/t

Ore type	Uranium dissolution at aggressive conditions (%)	Uranium dissolution at most aggressive conditions (%)
Ore A	90.3 (11.2 kg/t)	89.0 (18.6 kg/t)
Ore B	83.5 (14.8 kg/t)	85.5 (22.2 kg/t)
Ore C	80.5 (15.9 kg/t)	84.0 (22.5 kg/t)

The acid consumptions are included in brackets in kg/t.

Table 7

Uranium mineral distribution (%) in the residue samples of the aggressive leaching tests H₂SO₄: 16.3 kg/t, MnO₂: 4 kg/t, temp: 60 °C for all three ores

Mineral	Formula	Ore A (%)	Ore B (%)	Ore C (%)
Uraninite	UO ₂	9.5	1.0	10.2
Brannerite	(U,Th,Ca)(Ti,Fe) ₂ O ₆	69.9	91.2	73.6
U-Phosphate	(U, Cl)PO ₄	15.8	7.3	10.4
Coffinite	U(SiO ₄) _{1-x} (OH) _{4x}	4.8	0.5	5.8

Table 8

% Brannerite liberation and % of brannerite with 5 and 10% surface exposure for residue samples of the aggressive leaching tests H₂SO₄: 16.3 kg/t, MnO₂: 4 kg/t, temp: 60 °C for all three ores

Brannerite	Ore A (%)	Ore B (%)	Ore C (%)
Liberated	26.5	11.9	40.6
Free surface >5%	88.9	92.3	89.9
Free surface >10%	86.2	85.2	85.7

Investigating the liberation and surface exposure of the residue brannerite, it seems that brannerite is exposed to the lixiviant (Table 8). Therefore, neither liberation nor surface area exposure is expected to influence optimum uranium recovery.

The fact that the largest fraction of unleached uranium present in the residue exists as brannerite (Table 7), which is in fact exposed to the lixiviant (Table 8), is an indication of the slow reaction kinetics of brannerite. In a theoretical study to determine if brannerite is intrinsically inert or thermodynamically limited it was found that it maybe possible to extend brannerite dissolution by employing extreme leaching conditions (e.g. pressure leaching at high potentials and temperature). It complies with the findings by Zhang et al. (2003) that the rate of uranium release from brannerite is an order of magnitude lower than the release from uraninite.

In the experiments, a simple doubling of residence time to 48 h and use of nitric acid did not show any improvement in overall uranium dissolution and therefore, a more in-depth study is necessary to understand the leaching kinetics of brannerite. It must be noted that these dissolutions are actually quite high in absolute terms, considering that the feed grade is typically what would report to the residue in most international uranium mines.

Table 9

Residue bulk mineralogies of the three ores for most aggressive leaching conditions H₂SO₄: 25 kg/t, MnO₂: 4 kg/t, temp: 60 °C

	Ore A		Ore B		Ore C	
	Bulk (%)	Residue (%)	Bulk (%)	Residue (%)	Bulk (%)	Residue (%)
Quartz	70.2	70.2	73.3	73.3	79.8	79.8
Muscovite	10.1	13.9	11.3	12.4	8.2	12.9
Chlorite	2.0	0.6	3.2	0.8	3.6	1.4
Pyrophyllite	9.7	11.8	2.5	2.8	1.1	2.2
Pyrite	1.3	0.6	2.5	1.1	2.8	1.5
Carbonates	0.1	0.0	0.2	0.0	0.1	0.0
Albite	4.8	7.6	5.1	6.1	1.9	5.2
K-Feldspar	0.8	3.4	0.4	2.7	1.3	2.1
REE-phosphates	0.1	0.0	0.2	0.0	0.1	0.0
Other silicates	0.1	1.2	0.2	2.4	0.3	2.7
Other oxides	0.4	0.6	0.4	0.4	0.2	0.5
Copper	0.0	0.0	0.0	0.0	0.1	0.0
Carbon	0.2	0.3	0.5	0.6	0.3	0.5

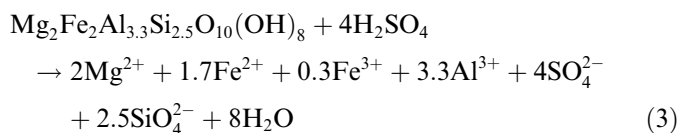
Note that the residue values were re-normalized to quartz (not expected to dissolve) to compare feed and residue bulk mineralogies.

3.3. Reagent consumption

Reagent consumption varied for the three ores. The results indicated that ore A consumes less acid, as compared to the others. Comparing the residue analysis to the bulk mineralogy, reasons for different acid consumptions can be established.

Quartz, muscovite, pyrophyllite and albite are found not to be major acid consumers, while chlorite is the most important acid consumer (see Table 9).

The lower chlorite concentration in ore A is thought to be partly responsible for the lower acid consumption (refer to Tables 5 and 6) as well as lower uranium concentrations. Although chlorite is an acid consumer, it is also an iron producer providing the necessary iron for uranium leaching before the addition of a solid oxidant. The idealized chlorite reaction is presented in reaction (3).



For oxidant requirements, pyrolusite is mostly used in the mining industry, since it is inexpensive. MnO₂ has the thermodynamic potential to oxidize U⁴⁺, but since it is kinetically difficult for a solid to oxidize another solid, manganese dioxide works via iron (Fe²⁺). Before introducing additional iron to the system it is important to establish if the soluble iron content in the ore is sufficient or not. In the absence of interfering ions, iron from chlorite will result in an initial Eh of ~414 mV before addition of pyrolusite and hence initiate dissolution of fast leaching uranium fractions. Pyrite on the contrary consumes Fe³⁺ but generates Fe²⁺ as well as H₂SO₄ (see reaction (4)).

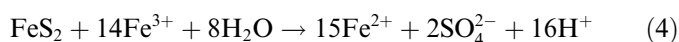


Table 10

Gold residue grades for the forward and reverse leach method for 8 and 24 h

	Au residue grade (g/t)						
	Forward leach			Reverse leach			Δ
	Au Grade	8 h	24 h	Au Grade	8 h	24 h	
Ore A: Test 1	7.99	0.77	0.48	7.66	0.19	0.12	0.36
Test 2	8.29	0.71	0.64	8.48	0.28	0.13	0.51
Ore B: Test 1	11.60	1.00	0.69	9.71	0.32	0.27	0.42
Test 2	10.60	0.86	0.68	7.96	0.36	0.23	0.45
Ore C: Test 1	12.60	0.97	0.93	12.70	0.67	0.31	0.62
Test 2	11.70	0.84	0.71	14.00	0.38	0.28	0.43

Analysis of the Fe:U mole ratio indicated sufficient iron being available throughout the leaching process for the uranium leaching reactions.

3.4. Effect of reverse leaching on gold recovery

Gold is generally insoluble in acid solutions in the absence of a suitable lixiviant. Therefore, through acid pre-treatment it is possible to dissolve gangue minerals to liberate gold without actually dissolving the gold itself. Results obtained for forward leaching (direct cyanide leaching of gold) and reverse leaching (sulphuric acid leaching of uranium followed by cyanide leaching of gold) are presented in Table 10. (Tests were done in duplicate.)

A gold recovery benefit of between 0.4 and 0.6 g/t was measured for the Vaal River ores. This improves gold recovery by 3–4% points to a total gold recovery of ~98% which have a definite monetary benefit.

Future work includes complete mass balances as well as a detailed comparative study of the kinetics of electrochemical dissolution of uraninite and brannerite minerals, to help establish optimum operating strategies for maximum uranium and gold dissolution from Witwatersrand ore.

4. Conclusions

It was found that it is very difficult to achieve uranium dissolutions higher than 90% for the Vaal River ores under conventional uranium leaching conditions. The predominant uranium mineral in the residue samples is brannerite, which is highly exposed, and some of it even fully liberated. The dissolutions achieved are in the region of the uranium fraction associated with uraninite in the feed. The fraction of brannerite with free surface exposure >10% in the residue samples varies between ~58% and 92%, as in the feed samples. Therefore, brannerite intrinsically reacts very slowly. Under the normal operating window, i.e. not considering uneconomical leaching conditions, ~90% dissolution will be a maximum. Aggressive leaching conditions as defined in this paper are thought to be suitable for extracting maximum value out of an atmospheric leach set-up. If ~100% extraction is desired, more extreme conditions (probably pressure leaching) will be necessary.

The leaching of uranium before gold results in improvements in gold recovery by 3–4% points to a total gold recovery of 98%.

Acknowledgements

This work was financially supported by AngloGold Ashanti and the authors would like to gratefully acknowledge the technical assistance provided by AngloGold Ashanti's senior personnel, Mr. Basie Maree and Mr. Bryan Penny.

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