

The progression of metallurgical testwork during heap leach design

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The depletion of favourably located and higher-grade mineral deposits, combined with more favourable metal prices and higher capital costs, has stimulated research into heap leaching of lower grade ores and wastes. This has resulted in renewed interest in heap leaching of uranium ores, and in the development of processes for the bacterial heap leaching of low-grade chalcopyrite ores and the acid heap leaching of nickel laterites. A complete programme of heap leach testwork typically involves a sequence of bottle roll tests and column tests, followed by piloting on test heaps. The initial testwork is aimed at determining the amenability of the ore to heap leaching, which depends on the characteristic of the ore with respect to porosity and permeability to leach liquor, acid consumption, metal recovery and percolation. Subsequent phases of the testwork focus on the development of the process design criteria for the treatment of the ore. Basic techno-economic models have also been developed as a tool for the initial estimation of heap leach viability. Mintek has also developed in-heap monitoring probes, coupled to an operator guidance system, in order to assist the operator in achieving optimal conditions for bacterial growth and heat preservation during pilot-plant operation.

Introduction

Heap leaching is used extensively for the processing of copper and gold ores. Low capital and operating costs and simple atmospheric leach processes make heap leaching suitable for low-grade ores and small deposits. Disadvantages include low recoveries, long ramp-up times, large footprint and acid-mine drainage of wastes³. Factors that may make ores unsuitable for heap leaching are poor percolation due to the presence of swelling clays, and high gangue acid consumptions. Percolation may be improved by agglomerating with acid or binder, and by minimizing ore compaction during stacking, for example by using conveyors instead of trucks.

For ores that contain silicate gangue minerals, it may be possible to reduce gangue acid consumption by introducing the lixiviant more slowly, for example, by using higher irrigation rates at lower lixiviant acid strengths. However, it is usually not possible to achieve an

economically viable optimum between metal extraction and acid consumption if the ore contains highly reactive gangue such as carbonates. For high acid-consuming uranium ores, it is more economical to use sodium carbonate as lixiviant.

Nickel sulphide and laterite ores usually have high acid consumptions compared with copper ores, but this is offset by the higher nickel value. The cost of acid typically comprises about 40% of the total operating cost for nickel laterite ores, so acid needs to be produced on site and heaps need to be operated counter-currently to optimize acid utilization.

Bacterially assisted heap leaching has been applied extensively for secondary copper sulphide ores, where the bacteria assist in the oxidation of ferrous iron and sulphur species. If the ore contains pyrite, bacterial oxidation can be utilized to generate acid and exothermic heat, which may be preserved within the heap through the appropriate manipulation of aeration and irrigation rates⁹. Higher temperatures will improve kinetics for more refractory species such as chalcopyrite, which do not give economic leach rates at ambient temperatures.

This paper discusses the metallurgical testwork sequence for heap leach design, with specific reference to chalcopyrite, uranium and nickel laterite heap leaching.

Progression of metallurgical testwork

Figure 1 shows the typical progression of metallurgical heap leach testwork through subsequent phases of roll bottles, column tests and pilot heaps. Since the duration and cost of testwork increases progressively, it is beneficial to optimize as many parameters as possible earlier on in the testwork programme.

Table I gives a summary of outputs for various testwork stages. Roll bottle or shake-flask tests are normally performed first, to obtain an initial indication of maximum achievable extraction and acid consumptions. However, acid consumptions are normally overestimated in rolling bottles, hence the acid consumption results must be treated as semi-quantitative. The 1 metre columns provide more accurate extraction and acid consumption data under trickle bed

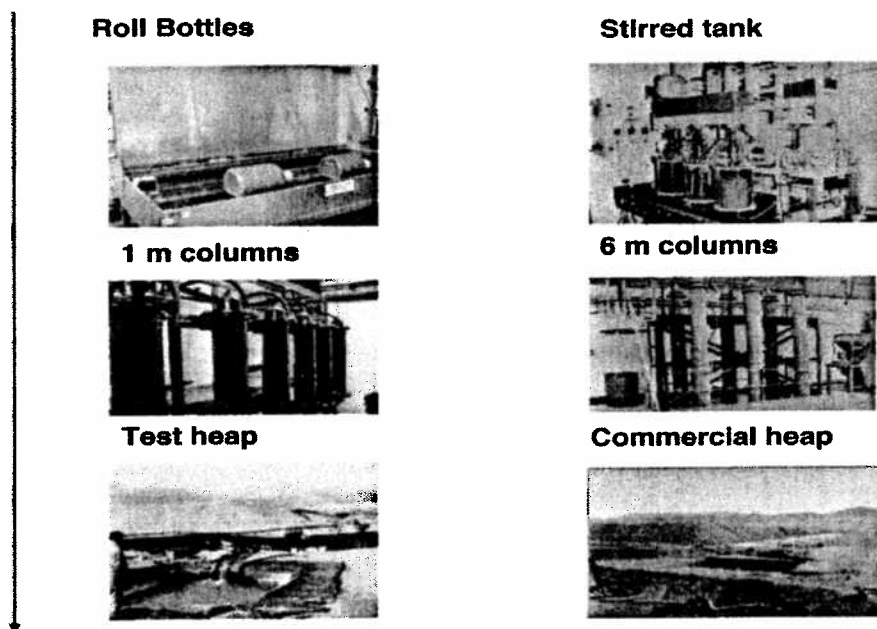


Figure 1. Staged approach to heap leach testwork and design

Table I
Metallurgical testwork programme

Stage	Outputs	Approximate duration days
Roll bottle, shake-flask or stirred tank	<ul style="list-style-type: none"> • benchmark recoveries and reagent consumptions • effect of mineralogy (laterites) • effect of crush size, lixiviant strength, redox and temperature 	<ul style="list-style-type: none"> • Cu oxides — 7 D • Cu sec. sulphides – 14 • Cu prim. sulphides – 30 • Ni laterites – 14
1 m columns	<ul style="list-style-type: none"> • identify severe percolation problems • maximum extraction and reagent consumptions under percolation-contact mode • optimal agglomerate moisture content • isothermal or temperature adjustment with heat model • aeration, inoculation and bacterial activity 	<ul style="list-style-type: none"> • Cu oxides – 50 d • Cu sec. sulphides – 80 • Cu prim. sulphides – 150 • Ni laterites – 80–150
6 m columns	<ul style="list-style-type: none"> • extraction kinetics, reagent consumptions under typical lift height of heap • impurity build-up in recycled solutions • more reliable simulation of compaction and percolation • neutralizing potential of ore for counter-current operation (laterites) • effect of bacterial activity on temperature behaviour predicted by heat model 	<ul style="list-style-type: none"> • Cu oxides – 150–200 • Cu sec. sulphides – 250–300 • Cu prim. sulphides – 300–365 • Ni laterites – 250–500
Test heaps	<ul style="list-style-type: none"> • best judgement on heap permeability in the absence of column side support • dynamic temperature variation as a result of bacterial activity demonstrated • net reagent consumption and impurity build-up in closed circuit with recovery step • counter-current operation to maximize copper recovery and minimize gangue reaction 	<ul style="list-style-type: none"> • Cu oxides – 150–200 • Cu sec. sulphides – 250–300 • Cu prim. sulphides – 300–365 • Ni laterites – 250–500

conditions, as well as an initial indication of possible severe percolation problems. But for a given irrigation rate (in terms of l/h/m²), the extraction kinetics from the column of ore (in terms of % extraction per day) are accelerated compared to the extraction kinetics that will be obtained from a taller heap of ore.

Since the kinetics and pregnant leach solution composition vary with lift height in commercial size heaps, a more realistic indication of leach kinetics and impurity build-up is obtained in taller columns, at a lift height equivalent of an actual heap (typically 6 metres). Although the taller columns provide an indication of slumping and permeability, the side support from the column may result in less compaction than will be observed on an actual pilot or commercial heap. Therefore, whereas solution build-up in columns provide a definite indication that percolation problems will be experienced at larger scale, trouble-free percolation in columns unfortunately does not guarantee successful percolation for the given irrigation rate on the commercial heaps.

As shown in Table I, the leach cycle increases from 150–200 days for oxides to 250–300 days for secondary sulphides and 300–365 days for primary copper sulphides (chalcopyrite). In order to demonstrate the effect of leach cycle on the pad area and mining rate, a production calculation is shown in Table II for a heap leach plant producing 20000 tonnes per annum copper, with a head grade of 0.6% copper. The chalcopyrite material has a longer leach cycle and lower recovery, resulting in bigger pad area and ore processing rate. This will result in larger capital costs for the mining, crushing and leaching sections.

Table II
Effect of leach cycle on pad area and mining rate

	Oxide	Sec. sulphide	Prim. sulphide	Units
Cu production	20 000	20 000	20 000	tpa Cu
Cu grade	0.60	0.60	0.60	% Cu
Leach cycle	200	280	365	days
Cu recovery	80.0	80.0	60.0	% Cu leached
Mining/stacking rate	4 166 667	4 166 667	5 555 556	tpa ore
Ore mass under leach	2 283 105	3 196 347	5 555 556	tonnes
Stacked bulk density	1.8	1.8	1.8	t/m ³
Ore volume under leach	1 268 392	1 775 748	3 086 420	m ³
Lift height	6.0	6.0	6.0	m
Surface area under leach	211 399	295 958	514 403	m ²

Table III
Operating stages during heap leach cycle

Stage	Conditions	Purpose
pH reduction	High irrigation rate, moderate acid strength (ILS), GL: GA ~ 10	Create suitable pH gradient for bacterial activity
Early leach	Moderate irrigation rate, moderate acid strength (ILS), GL: GA ~ 5	Maximize copper recovery, limit gangue reaction, promote bacterial activity, preserve heat
Late leach	Low irrigation rate, strong acid (raffinate)	Maintain the copper tenor while the leach rate is slowing
Rinse	High irrigation rate, low copper tenor (raffinate)	Recover entrained copper prior to heap closure

Low grade chalcopyrite heap bioleaching

Chalcopyrite is the most abundant copper mineral, comprising about 70% of the world's known copper reserves. Many low-grade reserves are untreatable with conventional heap leaching due to the slow heap leach kinetics at ambient temperatures. Since the leach kinetics improve with temperature, it is possible to improve kinetics by generating exothermic heat through the bacterial-assisted conversion of sulphide species (most importantly pyrite), and to preserve the heat within the heap through the appropriate manipulation of aeration and irrigation rates. The following approach is followed during metallurgical testwork for chalcopyrite ores:

- Column tests are performed in conjunction with a heat balance, which simulates the axial temperature profile in the core of a heap in the absence of side effects. Daily measured oxygen consumption data are input into the heat balance, and the temperatures in each jacketed column section is adjusted to simulate dynamic heap temperature behaviour. Columns are inoculated and aerated to promote bacterial activity
- The process is demonstrated on test heaps, typically 25000 tonnes, 40 m x 60 m x 6 m. Crushed ore is mixed with acid and inoculum in an agglomeration drum. The base of the pile is fitted with aeration pipes connected to a low pressure blower. The pilot heap is operated in closed circuit with a solvent extraction/electrowinning plant
- The heap life cycle is divided into distinct stages, in order to meet the combined objectives of maximizing copper recovery, minimizing gangue reaction and creating conditions for bacterial growth and heat preservation. Each stage has its own set of criteria for the control of the irrigation rate (G_L , kg/m²/h), aeration rate (G_A , kg/m²/h) and lixiviant acid strength as described in the Table III. 'New' heaps are irrigated with intermediate leach solution (ILS) whereas 'old' heaps are irrigated with return raffinate from the solvent extraction plant.

Mintek has developed in-heap probes (lysimeters), which allow solution and gas-phase samples to be taken for daily analysis at various depths in the heap. The lysimeters are placed in augured holes, and are also fitted with thermocouples. An example of the development of pH, redox and temperature profiles in the core of the heap as measured by the in-heap probes is shown in Figure 2. The reduction in pH gradient in the heap creates suitable conditions for bacterial growth. The increase in bacterial activity is evidenced by a corresponding increase in redox potential and temperature. Temperatures of about 40°C were maintained in the bottom half of the heap.

The lysimeter measurements assist the site engineer in making decisions concerning changes in irrigation and aeration rates, and changeover between subsequent stages in the leach cycle. Mintek has developed software for logging the daily measurements and assay results for multiple heaps. Each day, the software produces a report of the valve and flow adjustments required on the plant, as well as a log sheet for the measurements and assays required for the day. This operates in conjunction with a particular irrigation manifold and valve arrangement, which simplifies the task of visualizing the daily irrigation valve adjustments required, which is illustrated on the daily report. The heap software can also be applied to oxide and secondary sulphide ores.

Uranium ore

The rise in uranium prices has resulted in renewed exploration, expansion and construction of new projects such as Langer Heinrich (Namibia), Keyelekera (Malawi), Ezulwini (South Africa), and Buffelsfontein (South Africa). There is also renewed focus on exploitation of previously uneconomical ores and wastes such as Rossing (heap leaching of low grade ore) and Buffelsfontein (treatment of pyrite-gold tailings). Expansion through heap leaching is being investigated at Ranger (Australia) and construction of a heap leach plant at Arlit (Niger)⁴, amongst others.

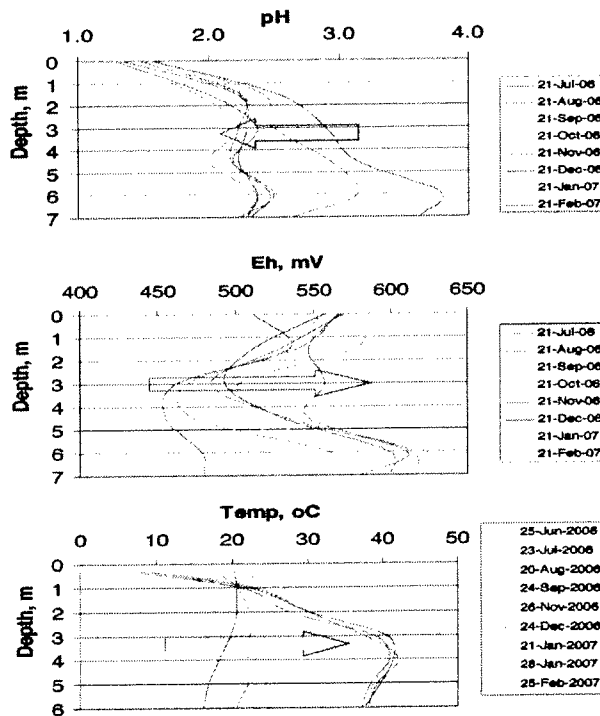


Figure 2. Progression of Eh, pH and temperature in bioheap core

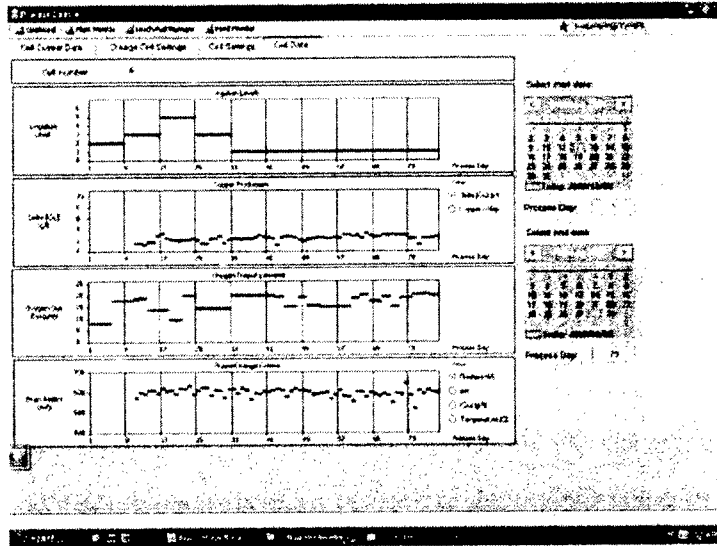


Figure 3. Heap leach control software

Table IV
Common uranium minerals

	Mineral	Formula	Operation
Leachable oxides	Uraninite ^{TL}	$U^{+4}_{1-x}U^{+6}_xO_{2+x}$	Rossing, Dominion Reefs, Ezulwini Narbalek, Kintyre
	Pitchblende ^{TL}	UO_2 to $UO_{2.25}$	
Leachable silicates	Coffinite ^{TL}	$U(SiO_4)_{1-x}(OH)_{4x}$	Rystkuil
Refractory complex oxides	Brannerite ^{Tr}	$(U,Ca,Fe,Th,Y)(Ti,Fe)_2O_6$	Elliot Lake Radium Hill
	Davidite ^{Tr}	$(La,Ce,Ca)(Y,U)(Ti,Fe^{3+})_{20}O_{38}$	
Hydrated oxides	Becquerelite ^{HL}	$7UO_2 \cdot 11H_2O$	
	Gummite ^{HL}	$UO_3 \cdot nH_2O$	
Silicates	Uranophane ^{HL}	$Ca(UO_2)_2Si_2O_7 \cdot 6H_2O$	Rossing Dominion Reefs
	Uranothorite ^{HL}	$(U,Th)SiO_4$	
	Sklodowskite ^{HL}	$(H_3O)_2Mg(UO_2)_2(SiO_4)_2 \cdot 2H_2O$	
Vanadates	Carnotite ^{HL}	$K_2(UO_2)_2(VO_4)_2 \cdot 3H_2O$	Langer Heinrich
	Tyuyamunite ^{HL}	$Ca(UO_2)_2(VO_4)_2 \cdot 8H_2O$	
Phosphates	Torbernite ^{HL}	$Cu(UO_2)_2(PO_4)_2 \cdot 10H_2O$	Rum Jungle Rum Jungle
	Autunite ^{HL}	$Ca(UO_2)_2(PO_4)_2 \cdot 11H_2O$	
Carbonates	Schroekingerite ^{HL}	$NaCa_3(UO_2)(CO_3)_3(SO_4)F \cdot 10H_2O$	
Arsenates	Zeunarite ^{HL}	$Cu(UO_2)_2(AsO_4)_2 \cdot 10-12H_2O$	
Hydrocarbons	Thucholite ^{TL}		

HL—hexavalent readily acid leachable without oxidation
 TL—tetravalent readily acid leachable with oxidation
 TR—tetravalent refractory

Uranium occurs in primary (tetravalent) and secondary (hexavalent) forms. Primary minerals require oxidation during hydrometallurgical extraction. The Table IV provides a summary of common uranium minerals. Uranium ores are usually processed by atmospheric tank leaching, pressure leaching, *in situ* leaching or heap leaching. Pressure leaching and bacterial heap leaching are applied where oxidation of pyrite is required, such as the

Buffelsfontein pyritic gold tailings project or Urgeirica heap leach (Portugal). Whereas tank leaching requires crushing and milling steps, heap leaching is performed on crushed or run of mine ore.

The most common leaching medium is sulphuric acid with ferric sulphate as oxidant (e.g. Rossing). Ferric sulphate is regenerated by addition of an oxidizing agent such as MnO_2 , H_2O_2 , or $NaClO_3$. Where the ore has high gangue acid consumption (>90 kg/t), leaching is performed in carbonate medium (e.g. Langer Heinrich tank leach, Trekkopje heap leach). Uranium recovery from the leach solution is usually performed by solvent extraction, ion exchange, a combination of solvent extraction and ion exchange (Eluex/Buflex), or direct precipitation. Uranium is precipitated from the eluate or strip liquor and calcined to produce U_3O_8 . Figure 4 shows the normal sequence for hydrometallurgical processing of uranium ore.

Most uranium heap leach plants have been satellites to existing operations, for example for treatment of low-grade waste with acidified ferric iron bleed streams (e.g. Rossing, Narbalek), and have been applied to ores with grades of below 0.1% U_3O_8 . However, current high uranium prices and plant capital costs makes uranium heap leaching attractive as a primary (greenfield) process (e.g. Trekkoppje, Arlit, Chirundu), provided reagent costs can be kept low.

For a standalone operation, ferric sulphate may be added as reagent (typically 0. g/L Fe, ORP 475–425 mV), or sufficient iron may be leached from the gangue minerals or by leaching pyrite acid plant calcine. A suitable oxidizing agent such as hydrogen peroxide, sodium chlorate, pyrolusite, or caro's acid is added to regenerate the ferric iron.

Bacterial heap leaching requires minimal additional capital cost, namely the installation of aeration piping and low pressure blowers. If the ore already contains pyrite, it may be possible to reduce the overall acid consumption by bacterial oxidation of the pyrite, which also results in exothermic temperature increase in the heap which may improve kinetics. At the same time ferric iron is regenerated by bacterial oxidation in the pile, resulting in reduced cost of oxidizing agent. The bacterial action throughout the heap maintains the iron in the ferric form, and since ferric iron precipitates above pH 2 (effectively yielding acid during precipitation), the pH throughout the heap is more rapidly buffered at a lower pH value when there is

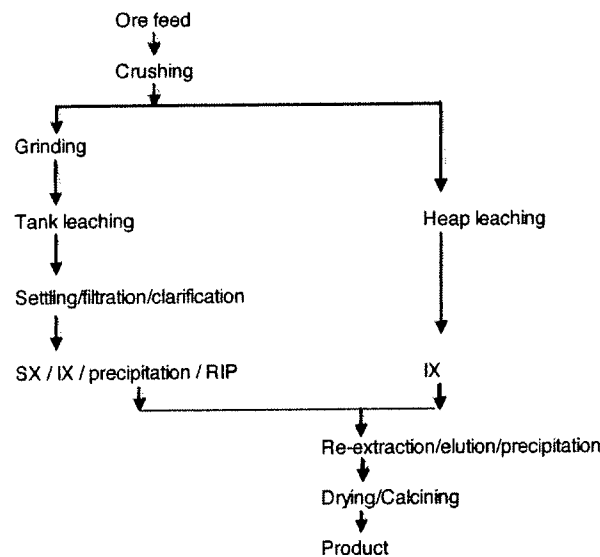


Figure 4. Uranium processing sequence

bacterial activity in the heap. Minerals indicated in Table IV as 'readily leachable' are acid leached at pH 1.5–2.0 and 35–60°C, which are suitable conditions for bioleaching. Minerals indicated in Table IV as 'refractory' require higher temperature (60–80°C) and stronger acid (up to 50g/L).

Figure 5 illustrates the effect of bacterial leaching versus chemical leaching for a uraninite/uranophane ore. Tests were performed in 1 metre columns, irrigated with 0.5 g/L Fe at pH 1.6. For the chemical leach column, the redox potential in the feed was maintained at 470 mV with hydrogen peroxide. For the bacterial leach, 2% pyrite was mixed into the ore, and the column was aerated and inoculated. The redox potential of the feed solution was initially adjusted to 450 mV, whereafter the redox in the recirculating solution was allowed to increase. Whereas the leach kinetics and uranium recoveries were similar, the acid consumption (measured as the acid in the feed solution minus the acid in the drainage) was about 35% lower for the bacterial column.

Nickel laterite ores

Laterites are usually processed with ammonia or high pressure acid leaching (suitable for limonites with high Fe, low Mg, low Si), or with smelting for ferronickel production (suitable for saprolites with high Mg, high Si). Whereas laterites were previously thought to be unsuitable for heap leaching due to the high clay content and acid consumption, the demonstration of a nickel laterite heap leach pilot plant in Caldag, Turkey, has resulted in a number of subsequent projects, e.g. Jump-Up dam, Canegrass and Murrin-Murrin. However, to date only one brownfield project is operational at Murrin Murrin. Laterites normally have very high acid consumptions (typically 500 kg/t), so a low-cost acid source or on-site acid plant is required. The leach solution also contains high levels of impurities such as iron that need to be precipitated prior to nickel recovery. The limonite fraction normally has poorer leach kinetics, so heap leaching may be suitable as an 'add-on' to an existing PAL process, for example heap leaching of a saprolite fraction to neutralize the pressure acid leach liquor.

Figure 6 shows the linear behaviour that was observed between nickel recovery and acid consumption in column tests performed on nickel laterite ore. This means that faster nickel recoveries can be achieved by increasing the rate at which acid is added, which may be achieved by speeding up the irrigation rate, or by increasing the acid strength in the lixiviant, or by admixing a large acid dosage to the ore before stacking of the heaps.

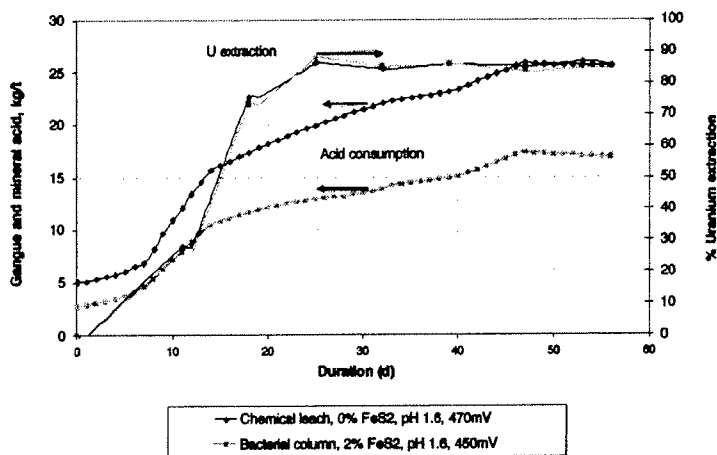


Figure 5. Bacterial versus chemical leaching of uranium ore

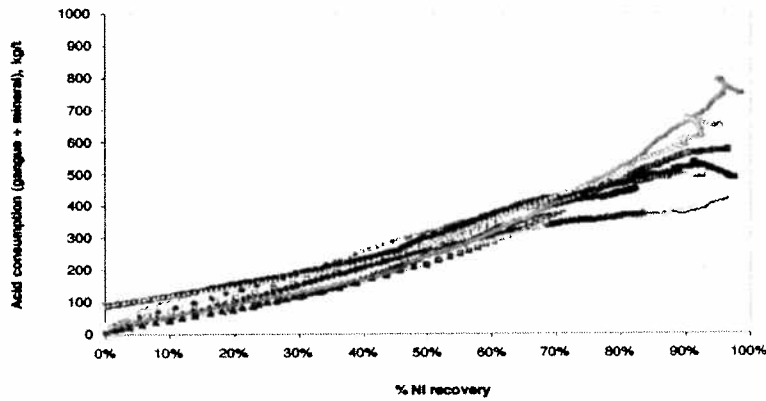


Figure 6. Acid consumption vs. nickel recovery

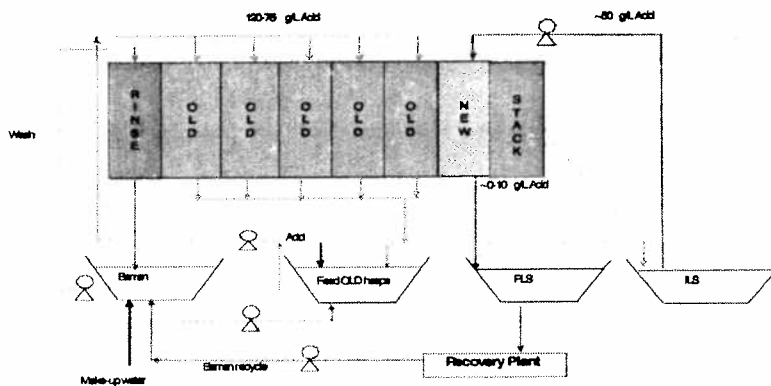


Figure 7. Proposed heap leach plant schematic

Figure 7 shows a proposed nickel laterite heap leach arrangement. The plant is divided into a series of 'old heaps' and 'fresh heaps'. In order to maximize leach kinetics, a high acid strength (75–120 g/L) is used over the 'old heaps'. New heaps coming into production are irrigated with intermediate leach solution draining from the 'old heaps' and the acid is neutralized over the fresh heaps. Any residual acid remaining in the drainage from the 'fresh heaps' has to be neutralized prior to nickel hydroxide precipitation, resulting in increased neutralizing agent and acid costs. An important part of column testwork is therefore to determine the neutralizing potential of the fresh heaps, so that the acid strength can be maximized, while at the same time ensuring complete neutralization of residual acid over the fresh heaps.

Figure 8 shows the experimental determination of the acid neutralizing potential of the laterite material in 6 metre columns. In order to simulate the arrangement in Figure 7, the column testwork was divided into 'fresh heap' and 'old heap' stages. Initially the heap was irrigated at 50 g/L acid, to simulate neutralization of intermediate leach solution (ILS) over 'fresh heaps'. As indicated, this acid concentration was completely neutralized over the fresh ore. After 80 days, the feed acid concentration was increased to 75g/L, and later to 100 g/L and 150 g/L, to optimize the acid concentration for the 'old heaps'. The aim was to produce drainage with an average acid content of 50 g/L over the entire 'old-heap' part of the leach cycle of the ore.

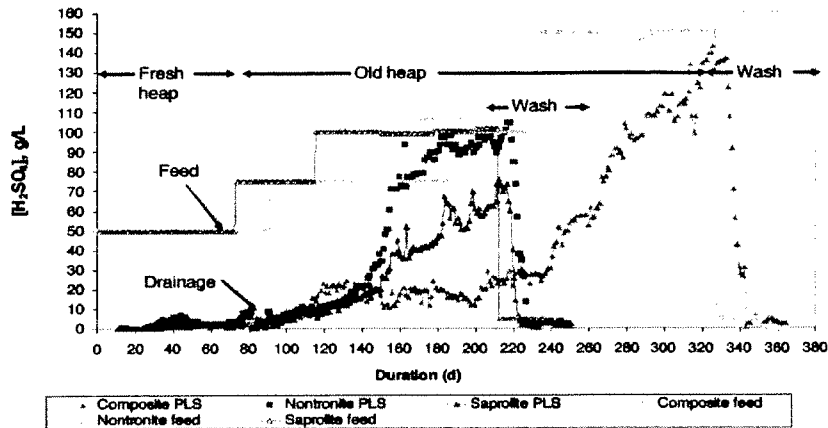


Figure 8. Determination of neutralizing potential of ore in 6 metre columns

From the data in Figure 8, it appears that an average of 50 g/L acid in the drainage of 'old' nontronite and sapolite ores is obtained by irrigating it with 100 g/L acid solution, and to obtain the same on composite material (consisting of limonite, nontronite and sapolites) irrigant acid strength of 100 to 150 g/L is required. These figures provided the first iterations of the liquor and acid balances over the process.

In addition to chemical acid leaching, there are also two potential routes for bioleaching of laterites:

- In order to save on the capital cost of an acid plant, elemental sulphur may be mixed in with the ore during agglomeration, and sulphuric acid may be generated bacterially within the heaps. Evidence from past heap leaching testwork has shown that bacteria can generate and tolerate up to around 70 g/L acid produced from commercial-grade pelletized sulphur added to the heap. This approach has, however, not been demonstrated or optimized in large-scale piloting or commercial-scale applications. There may be a capital cost saving from not building an acid plant; on the other hand, the cost of sulphur could be the same or even higher if the sulphur utilization is poor.
- Fungi have been used to generate organic acids (citric, oxalic) from a carbon source such as molasses, and nickel recovery from laterites of up to 60% has been reported⁷. There are, however, serious disadvantages associated with the use of fungal organisms on industrial scale. The cost of the organic carbon used as energy source makes their application very expensive and unless it can be replaced by inexpensive waste organic products, it would not be a viable process. Selective growth of the fungi would also be impossible under commercial conditions due to contamination with other undesirable microorganisms.

Summary and conclusions

Longer leach cycles and lower recoveries associated with chalcopyrite heap leaching will typically result in larger pad capital costs and larger ore processing rates for a given copper recovery. Chalcopyrite heap leaching may therefore be more suitable to brownfield applications, for example where existing solvent extraction and electrowinning infrastructure on brownfield installations is used. It is also necessary that the ore contains sufficient pyrite, in order to generate exothermic heat through bacterial oxidation.

The application of bio-heap leaching to copper ores provides flexibility to treat a blend of secondary copper sulphides and oxides with minimal additional capital cost, and an existing operation could also be adapted to process primary copper sulphides once the oxides and secondary sulphides become depleted. However, not all oxide/secondary sulphide copper ores are suitable to heap leaching, especially if the ore contains highly reactive gangue minerals.

Most uranium heap leach plants have been satellites to existing operations, for example for treatment of low-grade waste with ferric iron bleed streams. However, current high uranium prices and plant capital costs has resulted in standalone uranium heap leaching processes being developed (e.g. Trekkopje, Arlit), provided reagent costs can be kept low. Bioleaching offers a possible route with little additional capital expenditure for reducing oxidizing agent costs and also acid addition if the ore contains pyrite.

Due to the high costs of sulphuric acid and poor leaching characteristics of the limonitic fraction of the laterite mineralogy, laterite heap leaching may also find application as an add-on process, for example neutralizing acid liquor from a pressure leach plant over saprolite ore. Since the capital cost of the leaching section may be a small percentage of the overall capital cost of a greenfield project with large infrastructure requirements, the overall savings on capital cost may not be significant. As a result, laterite heap leaching will be more suitable to small projects with low infrastructure requirements, producing an intermediate product (e.g. Caldag), or where existing infrastructure is used, such as Murrin Murrin.

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