Uranium extraction: the key process drivers

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Synopsis

This paper describes and sets in context the commercially proven uranium extraction processes. It also discusses some of the technologies that are currently being pursued with the aim of reducing costs and overcoming a number of technical issues that are faced by process designers and operators. In doing so, opportunities for the transfer of extraction technology from other industries/commodities are identified.

The key processing areas that are examined include:

- The influence of mineralogy. The type of uranium mineralization and the nature of its association within a particular deposit will have a major impact on process route selection. Uranium minerals are classified according to their influence on extractive metallurgy. As the uranium price has risen in recent times, so opportunities are increasing for the recovery of uranium from less conventional resources such as phosphates and fossil fuels.

- The opportunities for employing techniques aimed at reducing the capital and operating costs of the overall process such as heap leach and beneficiation. Preconcentration by screening, gravity separation and radiometric sorting can potentially reduce the mass flow to the downstream leach and recovery plants.

- Purification or separation of uranium in carbonate systems. While it is possible to directly precipitate uranium from clarified leachate, the presence of impurities such as vanadium, molybdenum and others may dictate that ion exchange or selective precipitation is employed. Advances have been made in other industries in terms of improved ion exchange equipment aimed at reducing resin inventories and hence costs.

- Separation in acid leach systems. Earlier developments included the use of combined SX-IX systems (Eluex or Bufflex) and the introduction of continuous countercurrent ion exchange (CCIX). Column SX has found a ready application to uranium in view of the rapid extraction kinetics. With the renewed interest in uranium processing, efforts are being redirected towards further development of resin-in-pulp (RIP) as a means of driving down capital and operating costs.

- Extraction in a chloride environment. Australia and Namibia are desert countries where good quality water comes at a cost. However, both have access to saline water from either the ocean or subterranean aquifers. The issues associated with operating a uranium extraction operation in such environments is discussed.

- Extraction from less conventional sources. Recent price increases for U3O8 have resulted in a number of companies examining the recovery of uranium from sources such as fossil fuels and phosphate deposits. Both of these areas were examined in the 1970s and 1980s and a number of approaches were commercialized. The paper reviews the extraction options.

Introduction

The uranium industry has experienced over two decades of inactivity with the closure of virtually all the smaller operations in North America, South Africa and Europe. During this period production has been sustained by a small group of companies that have dominated supply. More recently, the Beverley project in Australia and the Langer Heinrich project in Namibia have come into production and numerous new projects are in the basic engineering and feasibility study phases. With the promise of increased demand, especially from countries such as China and India, previously mothballed projects are being resuscitated and, for the first time in many decades, new uranium exploration targets are being drilled.

Historically, the most commonly encountered flowsheet for uranium processing comprises conventional mining followed by comminution, acid leaching, solid/liquid separation, SX purification and finally precipitation/recovery as indicated in Figure 1. A high ore carbonate content, hence high acid consumption, may dictate the use of alkaline leaching, and some guidelines for the design of such flowsheets are presented in this paper. It is also noteworthy that the uranium deposits currently under investigation worldwide are becoming increasingly diverse in their mineralogy, grade, and the inherent cost structure of the projects, many of which are in relatively remote locations.

This paper, therefore, focuses on an examination of those factors that affect flowsheet selection in an endeavour to provide some guidelines for process route selection.
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The key factors examined are:

➤ The influence of uranium mineralogy and the lithology and behaviour of the host rocks
➤ Means of effecting capital and operating cost reductions, particularly for lower grade deposits
➤ Flowsheet options for acid leaching and purification
➤ Flowsheet options for alkaline leaching and purification
➤ Impact of salinity on the process flowsheet
➤ Recovery of uranium from less conventional sources.

Influence of uranium mineralogy and lithology on the flowsheet

A classification of uranium ores according to their metallurgical behaviour has been provided by Lunt and Holden1. The key minerals are identified as follows:

Ores containing tetravalent uranium

Tetravalent uranium has a low solubility in both dilute acid and carbonate solutions. To achieve economic recovery, oxidation to the hexavalent state is essential. Ore mineralogy can be oxide or silicate based.

The silicates include uranophanite and coffinite which, having the zircon structure, would be expected to be refractory but, in practice, dissolve quite readily in oxidizing acid or carbonate solutions. Oxidants include pyrolusite, sodium chlorate, hydrogen peroxide, Caro’s acid, SO₂/air and ferric ion.

The oxides have a simple structure and uraninite predominates. Low carbonate ores tend to dissolve fairly readily in acid solution with oxidant addition. However, these oxides are often associated with carbonates, and hence consideration may need to be given to an alkaline leaching regime. In such situations the uranium can be quite slow to dissolve, demanding long leach times and possibly pressure leaching to achieve optimal recovery. Fine grinding may also be required with these ore types ahead of leaching in either agitated tanks or an autoclave. Pitchblende is the massive form of this mineral and frequently occurs in veins.

Ores containing hexavalent uranium

These ores are less common and usually of secondary origin. They are readily dissolved in dilute acid or carbonate solutions, although an oxidant may be needed to prevent reduction through the presence of other species or ferrous iron introduced through grinding media or present in the ore.

Carnotite, a complex of uranium and vanadium, is an example of such ores. Being oxidized, it is often found in association with highly friable, clayey minerals such as calcrete and the associated solid/liquid separation issues are a key factor in process route determination. In general, the issues surrounding the treatment of carnotite ores include the probable need to employ alkaline leaching (e.g. Langer Heinrich and Lake Way) the impact of slurry viscosity on pumping and solid/liquid separation and, invariably, the requirement to separate uranium from vanadium in the downstream recovery circuit.

Multiple oxides

These mineral species are complex and include associations of uranium with rare earths and the tin, tantalum, niobium, titanium family and, often, all three groups together. Their complex nature with extensive crystal lattice substitution means that they are highly refractory. This refractoriness increases with increasing niobium and tantalum content and decreases with increasing titanium, iron and uranium content.

Examples of multiple oxides include brannerite and davidite, and such ores were commonly encountered in the Elliot Lake region of Canada. They are also the focus of recent exploration targets in Zambia. These ores may require fine grinding, prolonged leaching in hot acid or leaching at elevated temperature in an autoclave.

Associations of uranium with carbon

Ores associated with carbon may be classified as two types. Firstly uraninite can occur with fossil organic matter, possibly due to replacement. These may require fine grinding but otherwise are reasonably soluble in acid and alkali. In the past, workers have proposed to recover uranium from the boiler ash derived from the use of uraniferous coals and lignites. Since the ash will usually be high in silica attention must be focused on the possibilities of gel formation and difficulties associated with solid/liquid separation.

Uranium can also occur as a complex with organic matter, possibly humates. Being hexavalent, such minerals dissolve readily. However, the carbon and organic matter can give problems downstream, for example in thickening and filtration, and may interfere with phase separation in solvent extraction. In extreme circumstances such ores may require calcining ahead of uranium leaching.

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Figure 1—Acid leach flowsheet
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**Associations of uranium with phosphates**
Association of uranium with apatite is common. Although somewhat refractory, a proportion of the uranium will dissolve in the production of wet process phosphoric acid and is recoverable. Previous plants that exploited this source of uranium invariably had to adapt the recovery circuit to the properties of the acid in terms of its organic content, concentration of P<sub>2</sub>O<sub>5</sub>, temperature, oxidation state of uranium, etc.

**Host rock**
The nature of the host rocks plays a major part in flowsheet selection. The presence of carbonate minerals in sufficient quantity to cause acid consumptions of greater than about 75 to 100 kg/t ore leached is likely to be the deciding factor in favour of carbonate leaching. This is equivalent to calcite content in the ore of 7% to 10%. Generally, the acid route exhibits advantages with respect to circuit simplicity and a greater range of purification options than alkaline leaching. However, each situation needs to be evaluated on its own merits. In summary, the ultimate process route selection is influenced by:

- The concentration of uranium in the ore with higher grade material being able to tolerate higher acid consumptions without having to contemplate alkaline leaching.
- More rapid kinetics of the acid leach over carbonate digestion of the same ore type. This has ramifications on the leaching step and also comminution, where acid leaching may not require such a fine grind.
- The presence of valuable by-products in the ore and the ability of either flowsheet to recover these species economically.
- The price of the reagents themselves and the relative transportation costs.
- The possibility of improving project economics by recovering energy from a sulphur-burning acid plant.
- Choice of purification step in acid leaching that is wider than that of alkaline leach circuits. The options for acid circuits include solid ion exchange (fixed bed, continuous counter-current, resin-in-pulp and the carousel) and SX (mixer-settler and pulsed column) and possibly combinations of IX/SX.

**Ability to concentrate**
There is potential to upgrade ores ahead of both acid and alkaline and acid leach circuits and this factor should always be examined in the evaluation of a new project or the expansion of an existing one. A number of techniques are possible including:

**Upgrading by screening**
Screening can be effective where uranium minerals report in significant quantities to the finer fraction of run of mine or primary crushed material. It has been applied in a number of situations and to a variety of uranium minerals.

The method finds particular applicability in the treatment of secondary minerals, particularly carnottite, in which the uranium tends to report with the fines. The Langer Heinrich uranium project incorporates primary crushing and scrubbing followed by screening and rejection of essentially +0.5 mm material.

There is potential to apply technology that has been adopted in other industries, particularly nickel laterite processing, in terms of both equipment and unit operations. Consideration should be given to mineral sizers (where these are compatible with ore competency and abrasion indices), scrubbers, attritioners to enhance recovery and log washers to improve separation of clayey undersize from competent material.

Screening at fine sizes can also present a challenge, particularly if the slurry has a high viscosity. Other industries such as gold CIP/CIL and nickel laterite processing have successfully employed Derrick screens for this application. These units have a small ‘footprint’ and an exceptionally high capacity per unit area.

**Upgrading by radiometric sorting**
Prior to the mid-1980s downturn in the uranium industry, radiometric sorting was beginning to make significant inroads into the flowsheet. Production units of the Ore Sorters Model 17 were installed at the Mary Kathleen, Western Deep Levels, West Rand Cons, Cenex and St Anthony mines. These units generally treated ore in size ranges between 25 mm and 150 mm. At the coarser end of this range throughputs of up to 100 t/h were being achieved.

Radiometric sorting is particularly applicable to vein type deposits where high grade mineral is discrete and easily separated from low grade material and gangue. The radiometric sorter at Mary Kathleen achieved recoveries of about 90% of the uranium in the sorter feed into 40% of the mass. The sorter product ran about 2200 ppm U<sub>2</sub>O<sub>5</sub>.

This technology should be a consideration and especially so in situations where the downstream process is likely to be more costly than the norm. For example, with refractory ores that demand long leach retention times or the use of high temperature autoclaves. However, with the increasing price of uranium that has occurred in recent months, the economic necessity of rejecting a low grade fraction has diminished.

A number of the larger producers are looking to radiometric sorting as a means of increasing output at existing facilities or maintaining output in the face of declining grades. A demonstration scale unit has been installed by Ultra Sort at Rossing Uranium in Namibia (refer Figure 2) and will treat crushed feed in two size ranges between 80 mm and 300 mm. Details of the installation have been published by Gordon and Heuer.2 Earlier units were controlled on the basis of a cut-point based on grades of product and rejects. Modern equipment has advanced in sophistication. Input and output tonnages and grades can be integrated such that the units can be controlled on the basis of the U<sub>2</sub>O<sub>5</sub> upgrade ratio. This is particularly advantageous in situations where the mineralization is fairly highly concentrated close to the economic cut-point and where a rapid response is required in the event of grade changes.

**Other concentration techniques**
Gravity concentration methods are not readily applicable to uranium minerals that are usually finely disseminated in the
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ores. A possible exception is uranium associated with pegmatites. Heavy media separation methods have been employed, most notably at Radium Hill in Australia where a -25 mm +10 mm feed was concentrated ahead of leaching.

Rationalization of the leaching process

A number of means are available for reducing the capital and operating costs of the total project by rationalizing the leaching process and/or integrating the leaching step with downstream processes. Incorporation of these leaching operations has the potential to significantly reduce both the capital and operating costs of a uranium project. Three options are discussed below:

Heap leaching

Heap leaching was successfully employed on some of the earlier uranium projects. While the technique was primarily aimed at boosting output by exploiting low grade and near grade ores, such as at Narbalek in Australia, some projects in South America and elsewhere have used heap leaching as the major dissolution step. Clearly the method can have advantages in rationalizing and reducing the capital and operating costs, as indicated in Figure 3. The unit operations of milling, solid/liquid separation and clarification can potentially be discarded (see Figure 1). Heap leaching does, however, require attention to a number of issues, i.e.:

➤ Leaching kinetics that can be significantly slower than for other metal oxides such as copper. This requires a focus on solution management in order to maintain acid and oxidant strengths and the maintenance of leachant concentrations

➤ Ensuring the absolute integrity of the heap leach membrane to prevent any solution leakage. Double membranes are likely to be the minimal requirement for uranium leaching. These could be of plastic construction, e.g. various grades of HDPE or possibly clay in combination with HDPE. Leak detection systems will be required

➤ In arid areas and for health, safety and environmental reasons it will be necessary to minimize evaporative loss through the use of drippers as opposed to spray type systems and even to cover the heap with a layer of crushed waste

➤ The closure plan for the heap will need to be carefully planned and executed. Upon termination of the project the heap will need to be irrigated to the point where the solution tenor is below statutory limits for discharge. Following this the heap will need to be capped to prevent dusting, although it has to be said that such requirements will also apply to the tailings storage facility in the conventional milling circuit. Capping is typically done with a layer of topsoil overlaying crushed rock to prevent dusting

➤ While the earlier applications of this technology focused on acid leaching, consideration is being given to alkaline leaching on heaps. This is the case on the Trekkopje project, Namibia, that is currently being investigated.

The design of heap leach systems, both in terms of the physical construction and the metallurgical development programme, can draw from the depth of experience in other industries, particularly gold and copper heap leaching, and increasingly from the experience in leaching of the kinetically slower nickel laterites. This first nickel laterite leaching system is presently being commissioned in Turkey and a number of others are planned in Australia and South America.

In situ leaching

In situ leaching has been employed extensively in the past where the deposit and surrounding aquifer permit its consideration. Numerous in situ operations were operated in
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Wyoming, New Mexico, Portugal and elsewhere. More recently, the Beverley project in South Australia has adopted the technique successfully. The projects have all employed acid as the lixiviant in view of the nature of the host rock that is often a sandstone type having low acid consumption. The in situ leach process reduces the number of unit operations to a minimum, with the facilities simply including purification of leach solution and recovery of yellow cake by various precipitation methods. The mining operation reduces to a well field with the wells typically geometrically spaced between 10 and 20 m. The wells are either injection wells through which the lixiviant is pumped or production wells from which the uranium-bearing liquor is recovered.

The issues to be addressed include:

➤ The permeability of the orebody and the nature of the underlying rock that should ideally provide a barrier to solution migration. Often the uranium-bearing ore is contained in sandstone strata that is sandwiched between less pervious rock layers such as shale or mudstone.

➤ The hydrology of the deposit and the depth of the aquifer that is typically less than 200 m below surface.

➤ The volumetric flow, U₃O₈ tenor and impurity levels of the leach liquor that will affect the selected downstream purification and recovery unit operations. For acid leaching the choice will usually reside between SX and IX. Fixed bed ion exchange is a robust process that can accommodate wide variations in solution tenor in view of the relatively large inventory of resin, and this was the driver for the selection of this process at Beverley uranium.

Resin-in-pulp (RIP)

This is another process that was quite commonly used, especially in the USA, during the 1970s. RIP technology was also extensively used in Russia for the extraction of uranium, copper and other metals. The adoption of this unit operation eliminates the solid/liquid separation step downstream of leaching and has the potential to combine the leaching and purification steps, hence reducing both capital and operating costs. The early plants employed resin enclosed in baskets such as shale or mudstone.

➤ The efficiency of separation of the ore particles and the resin. The available size of resin beads will dictate the grind size adopted and influence the ease or otherwise of the screening operation

➤ The attrition rate of the resin that necessitates that it is screened and pumped carefully. Historically, resin attrition has been an impediment to the introduction of the technology. However, suppliers are now developing harder resins that exhibit lower abrasive losses

➤ The ability to achieve product grade yellow cake using an IX process on its own. If the impurity levels are such that an SX step is needed downstream then the economics of a two-step separation process versus solid/liquid separation and SX will need to be evaluated

➤ The necessity of recirculating barren liquor to the process. This applies particularly to alkaline leaching with sodium carbonate concentrations of around 40–50 g/l. It is mandatory to recirculate such solutions for reuse and this will generally preclude the use of RIP.

A number of suppliers and laboratories are developing new and more robust resins specifically for RIP applications. Permusoft Proprietary Limited, who are the Australian distributors for Rohm and Haas resins, are conducting extensive testing in Rohm and Haas laboratories in France and are developing a series of resins specifically for RIP.

Overall circuit selection—acid leaching

The main circuit options for acid leaching are described in Figure 4. These tend to be driven by the choice of purification circuit that, in turn, has an influence on the selection of the solid/liquid separation unit operation utilized. Separation technologies include:

➤ Straight ion exchange. A variety of resin types are available and the contacting technology includes fixed bed (FIX) and continuous countercurrent ion exchange (CCIX) and the resin-in-pulp (RIP) process discussed previously. The CCIX technology may still have applicability to low U₃O₈ tenor solutions and a number of designs appeared in the late 1970s. The advantages of the CCIX units over FIX included lower capital costs.
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and a significant reduction in the resin inventory, and hence a lower operating cost. A further advantage of CCIX over FIX is that it is capable of handling unclarified solutions with, in practice, solids contents of up to 1–2%. The ultimate solids loading will be dependent on the size analysis of suspended solids.

- A more recent technology that seeks to achieve continuous (or at least semi-batch) operation and large reductions in resin inventory is the SepTor contactor developed and marketed by Outokumpu/SepTor Technology. A second company, IONEX separations, supply a competing ion exchange contactor. These units operate in a carousel arrangement and are essentially multiple fixed beds. They therefore require a clarified feed. They have an increasing track record across a range of industries and have succeeded in achieving the main objective of a low resin inventory.

- Solvent extraction (SX). The ubiquitous extractant is a tertiary amine that supplanted the cationic D2EHPA systems in the 1950s. The anionic reagent has a greater selectivity for uranium than strong base resins and even the more selective weak base resins. A wide variety of equipment is available to the engineer including various designs of mixer-settler and SX columns. The latter contactor is particularly applicable to uranium SX where the extraction kinetics are extremely rapid.

- Eluex (Bufflex in South Africa). This system combines the ion exchange and solvent extraction operations into a sequential system. It can achieve greater selectivity than the individual processes.

**U₃O₈ leachate concentration and project capacity**

One of the major factors affecting the selection of a purification process is the flow rate of pregnant liquor, which has a significant impact on both capital and operating costs, particularly in the case of SX and FIX, but to a lesser degree with CCIX.

The use of an Eluex circuit incorporating strong base resin CCIX followed by SX using mixer-settlers was shown to have an economic advantage in the treatment of high solution throughputs and a number of such circuits have been used. One such configuration was adopted at Rössing. With the advent of column SX, and the economies that these contactors can achieve, it is likely that the crossover point in terms of plant throughput will have moved in favour of direct SX over Eluex.

In the treatment of pregnant liquors having high U₃O₈ concentrations the SX route will, in any event, become progressively more attractive and would undoubtedly give superior economic returns at tenors greater than 0.9 g/l U₃O₈. This can be explained by the fact that full advantage of CCIX as a cheap uranium preconcentration step is not realized unless significant reductions in the size of the downstream SX are achieved. The preconcentration ability of CCIX is limited by two factors, i.e.:

- The maximum concentration of U₃O₈ that can be achieved in the CCIX eluate. Historically plants have operated with eluates in the range of 4–7 g/l U₃O₈

- The loading capacity of amine extractants has an upper limit of around 8–10 g/l U₃O₈ beyond which phase separation problems can arise.

**Selectivity**

Acid leach purification processes can be ranked in order of decreasing selectivity as follows:

- Eluex > SX and weak base IX > strong base IX

Strong base IX is the least selective acid leach purification process, and with the trend towards tighter specifications for impurities it may be unsuitable unless used as the initial operation in an Eluex process or in association with a very selective final precipitation process such as hydrogen peroxide precipitation.

Weak base IX and direct SX have comparable selectivities and are usually capable of achieving nuclear grade uranium on their own. Weak base IX does, however, suffer from severe silica fouling were this species is present in the leachate and this has severely limited the application of this resin.

Eluex is the most selective of all processes combining the selectivities of both IX and SX.

A great deal could be written about the ability of the various processes to deal with impurities. It is instructive to summarize the ability of the SX and IX processes to reject the commonly encountered impurities. This is done in Table I.

**Overall circuit selection—carbonate leaching**

The major flowsheet options for alkaline uranium leaching are illustrated in Figure 5. The alternatives for extraction are essentially either direct precipitation from leachate or extraction via IX. The equipment variants for the latter process are basically similar to those for acid leaching and include FIX, CCIX and the recently developed SepTor/IONEX carousel systems. The use of RIP will preclude recirculation of the leachant and is likely to be uneconomic or at least reduce the returns of the project dramatically.

The key operating cost in the alkaline leach circuit is carbonate consumption and hence this factor, along with the selectivity of the extraction circuit, are the key issues that need to be addressed by the engineer.

**Selectivity**

The methods for handling impurities in IX systems are essentially similar to the methods advocated in Table I. For direct precipitation of uranium from carbonate leach solutions then the methods given in Table II can be considered.

It is unlikely that direct precipitation from carbonate liquor will produce a marketable product without resorting to redisolution and reprecipitation or the use of more selective precipitation techniques such as hydrogen peroxide precipitation of uranium peroxide. This last method is somewhat selective over impurities such as vanadium, which is commonly associated with secondary uranium mineralization.

**Sodium carbonate consumption**

This is arguably one of the most critical factors in establishing a viable flowsheet whether this be direct precipitation from leachate or recovery from an ion exchange eluate.
With respect to a basic flowsheet incorporating CCD and IX (refer Figure 5), the circuit configuration must strive to maximize the input of fresh wash water (or tailings dam return water) to the final stage of the CCD. Unless this is achieved, soluble carbonate/bicarbonate will report to the tailings dam with the final CCD underflow. Depending on the percentage return of tailings solution to the process, this can lead to a significant reagent make-up cost.

The following options can be considered as a means of maximizing the amount of water that can be used as CCD (or belt filter) wash:

➤ Recirculate a portion of the IX barren solution for use as mill circuit water addition. Interpose a thickener between the mill and leach circuits and recirculate thickener overflow to the mill

➤ Utilize IX barren solution in the penultimate CCD stage as wash and use only fresh water as the final stage wash solution

➤ Dissolve make-up sodium carbonate and bicarbonate in recirculating liquor as opposed to water or consider direct addition to the leach circuit as powder

➤ Minimize the use of pump gland seal water around the process plant

➤ Consider indirect heat transfer instead of direct injection of steam into the leach tanks. However, this

<table>
<thead>
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<th>Impurity</th>
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<th>IX Separation technique</th>
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<tbody>
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<td>Vanadium</td>
<td>Reduction to the tetravalent state prior to SX</td>
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<td>Molybdenum</td>
<td>Separate U and Mo stripping circuits</td>
<td>Prior precipitation from pregnant solutions</td>
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<td>Activated carbon adsorption of Mo</td>
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<td>Reductive scrubbing</td>
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<td>Use of secondary amines</td>
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<td>Arsenic</td>
<td>Crowd off during extraction</td>
<td>Precipitation from pregnant solution</td>
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<td>Low pH scrubbing after extraction</td>
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<td>Silica</td>
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<td>Scrubbing post the extraction step</td>
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<td>Ferric iron</td>
<td>Crowd off during extraction</td>
<td>Crowd off during adsorption</td>
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<td>- Thermal methods</td>
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<td>- Dissolution and reprecipitation</td>
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<td>- Precipitation prior to uranium, e.g. with FeSO₄ or PbSO₄</td>
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<td>- Precipitation with peroxide</td>
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<td>Molybdenum</td>
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<td>Sodium</td>
<td>Dissolution of sodium diuranate and reprecipitation</td>
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<td>- Use of hydrogen peroxide precipitation of ferric molybdate</td>
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needs to be tempered by further consideration of slurry viscosity and the propensity for scaling of exchanger surfaces

➢ Consider the use of belt filters as an alternative to CCD if filtration rates are favourable.

Extraction in a chloride environment

This issue is of relevance to projects contemplated in Australia where groundwater is highly saline (often three times that of seawater) and countries such as Namibia where water is a scarce and valuable commodity but where the uranium deposits are frequently located near the ocean. Chloride anions, as would be expected, load readily onto the tertiary amine indeed the chloride ion is more strongly extracted than the bisulphate anion. The result is competition for active extractant sites with the uranyl sulphate complex. Cognis® has observed that the presence of 2.5 g/l Cl in Olympic Dam pregnant liquor (derived from the sodium chloride oxidant) reduces uranium loadings by as much as 20%. The level of chloride in waters of the Eastern Goldfields region of Australia can be 50 times higher than this.

For projects faced with this issue, a possible solution could be to employ an Elux circuit with a catonic resin (aminodiacetic acid type) employed in the primary extraction stage. This resin, of which a number are commercially available, extracts uranium as the cation (UO2++) preferentially to anionic species including chlorides. The resin can then be stripped with dilute sulphuric acid and the uranium re-extracted selectively using SX. Such a system was contemplated in the early studies for the Key Lake® uranium project.

A similar system might be contemplated using D2EHPA (the Dupex Process) SX. As noted previously, this extraction technique was employed prior to the commercial development of tertiary amine solvents. It has also been reported by ANSTO that the use of a synergistic mixture of tertiary amine and D2EHPA has a much greater tolerance for chloride anions and behaves in a similar manner, from the viewpoint of extraction, to the tertiary amine. This organization is continuing to develop this avenue for extraction in chloride environments.

Recovery from phosphate sources

Most phosphate rocks contain uranium that digests during the production of wet process phosphoric acid (WPPA). Typically dehydrate WPPA (30% P2O5) derived from Moroccan or Florida rock contains around 80 to 150 mg/l U3O8 hence the recovery of uranium from apatite sources has focussed on extraction from WPPA. In the heyday of this process about 1800 t/a U3O8 was produced from phosphoric acid at a dozen facilities worldwide. At first sight the economics of uranium recovery from this source look attractive since the mining, comminution, leaching and solid/liquid separation costs are born by the phosphoric acid production process itself. It would, therefore appear to be a relatively straightforward exercise to recover uranium using, for example, solvent extraction either before or after the WPPA concentration step. In practice, however, there are technical challenges associated with recovery of uranium from WPPA including:

➢ The presence of organics in the WPPA derived from humates in the rock itself and antifoaming agents added during the leaching process. The organic content of so-called black acid can be high and can cause severe interference with phase disengagement in the uranium SX process

➢ The valence state of the uranium in the WPPA. Since it is not necessary to add an oxidant in the leach process, uranium is frequently found in the tetravalent state. This has led to the development of a number of two-stage SX processes in which uranium is recovered as the cation (UO2++), stripped and then re-extracted in a second circuit using another extractant to achieve selectivity over other cationic metallic species. The most commonly encountered process was two-stage SX using DEHPA-TOPO

➢ The presence of fine solids that required extensive clarification ahead of SX and crud treatment facilities within the uranium plant.

Some deposits such as Bakouma in the Central African Republic have an unusually high uranium content that demands a fresh approach to the potential of recovering uranium as the primary metal. This project is currently under detailed study.

Conclusions

The uranium industry is again experiencing a period of unprecedented growth. The selection of an optimal flowsheet will depend on a number of factors as defined in this paper. Experience of flowsheets developed two to three decades ago is useful. However, process development is again moving forward and advances made in other industries in the intervening period can be effectively utilized in uranium extraction. Specific issues such as the need to operate in saline environments, recovery of uranium from unconventional sources and the development of uranium specific technologies such as RIP are being addressed by a number of engineering companies and technology suppliers. The authors would like to thank the management of GRD Minpro Limited for permission to publish this paper and to acknowledge the input of their colleagues in undertaking uranium projects and uranium feasibility studies.

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