CHAPTER 12.41

Uranium

Henry Schnell

Uranium is a naturally occurring element with an average concentration of 2.8 ppm in the earth's crust. Traces of it occur almost everywhere. It is more abundant than gold, silver, or mercury, with about the same abundance as tin but slightly less plentiful than cobalt, lead, or molybdenum. Vast amounts of uranium also occur in the world's oceans, but in very low concentrations.

Miners had noticed uranium minerals for a long time prior to the discovery of uranium in 1789. The uranium mineral pitchblende, also known as uraninite, was reported from the Krušnéhory (Ore Mountains), Germany, as early as 1565. Other early reports of pitchblende date from 1727 in Jáchymov (Czech Republic) and 1763 in Schwarzwald, or the Black Forest (Germany). Eugene M. Péligot isolated the element in 1841. Antoine H. Becquerel discovered its radioactivity in 1896. Before the discovery of nuclear fission by Otto Hahn and Fritz Strassmann in 1938, the principal use of uranium (chiefly as the oxides) was in pigments, ceramic glazes, and a yellow-green fluorescent glass, and as a source of radium for medical purposes. It has also been added to steels to increase their strength and toughness. However, because of the high toxicity (both chemical and radiological) of uranium and its compounds, and because of their importance as nuclear fuel, these earlier uses have been curtailed.

Today, the main market for uranium is for nuclear fuels and medical requirements for some of the associated isotopes. Uranium differs from many metal commodities, as its markets are energy demand based. The uranium market has had many ups and downs—affected by the energy crises of the 1970s and nuclear accidents such as Three Mile Island, Chernobyl, and, more recently, Fukushima—that continue to fuel the nuclear debate. However, demand for uranium continues to grow to supply the existing 447 worldwide nuclear reactors and 61 new reactors under construction in 15 countries.

Uranium minerals are very commonly distributed geographically and in a very wide range of mineral species and deposit grades. In the past 25 years, this diversity has been evident, with production in more than 20 countries and many more countries preparing for production to support their nuclear power industries. In 2014, approximately 54% of world production came from just 10 mines in six countries, with these six countries providing 85% of the world's mined uranium.

With uranium found in a large variety of settings, uranium processing must be adaptive to these settings and the large variety of mineralization. Historically, flow sheets were based on a tank leach process that tended to have very similar flow sheets that are referred to as "conventional." There has been a trending away from such a conventional flow sheet, or at least the technology within each unit operation, as high-grade deposits were exploited in Canada or as extreme low grades were considered in places such as Namibia.

The large diversity of mineral species and ore grades has resulted in an increased diversity of processing options and interesting adaptations of hydrometallurgical processes. The high grades found in Canadian ores have seen remote mining methods and special plant designs to address the challenges of these special ores. Kazakhstan has advanced the in situ leach/leaching (ISL) technologies, making it currently the largest producer. The low grades found in Namibia have resulted in higher tonnage operations (up to 100,000 t/d of ore) and significant technology advances of preconcentration, ion exchange (IX) adaptations, and other innovations. The advancement of resins has resulted in the reemergence of IX options. The ores of Niger similarly needed adaptation to deal with high clays found in these ores.

As concern about environmental impacts and climate change has grown, a number of high-profile environmentalists have decided that this is a more serious problem than their previous concerns with nuclear power. They have, to varying degrees, either changed their public stance or conceded that because nuclear power is virtually emission free regarding carbon dioxide (CO_2), it merits at least grudging support as part of the response to increasing atmospheric CO_2 levels and the depletion of fossil fuels. But public education and the mystique around radiation continue to play an important part in advancing uranium and nuclear projects.

Uranium will continue to lead the way in technology advances as by-product production gains importance. The environmental advances for these mines will result in

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adaptations for other metal resources. Hydrometallurgy will continue to provide new adaptations, and these advances will also flow into other metallurgical areas.

GEOLOGY, MINERALOGY, AND URANIUM DEPOSIT TYPES

History of Uranium Use

Uranium was discovered by H.M. Klaproth in 1789 in samples coming from the silver mines of Bohemia but was first isolated as a metal by E.M. Peligot, a chemist from the Baccarat glass factory in France. Uranium was initially used essentially because of the bright yellow color of the uranyl ions and hexavalent uranium minerals to color the glasses (autunite) until the early 1980s and was used for body paint by the early Navajos and Ute in Colorado (carnotite).

Exploration for uranium ores began after the French chemist Marie Curie discovered radium in 1898. About 1 g of radium was produced for every 8 t (metric tons) of uranium ore. The total amount extracted until 1939 did not exceed about 6,000 t U. Uranium ores were mined mainly from the former Belgian Congo (Shinkolobwe in the Democratic Republic of the Congo), Bohemia (Jáchymov), the Colorado Plateau in the United States, and Port Radium in Canada. Radium had a large variety of applications but was mainly used for radiotherapy.

A second rush for uranium exploration started at the end of the 1940s, essentially for military use after the discovery of atomic fission by Hahn and Strassmann in 1938 and development by the U.S. Atomic Energy Commission of the first atomic bombs by the end of World War II. For these bombs, the uranium was mainly from Shinkolobwe, a uranium inventory present on a Belgian ship rerouted to New York Harbor when the Germans invaded Belgium in 1939. Until the end of the 1960s, uranium was mined without economic considerations. In 1959, production reached a peak of 50,000 t U/yr.

The third and strongest rush occurred in the 1970s as a consequence of the huge increase of oil prices in 1973. A vast program of nuclear reactor construction started all over the world. Before 1973, uranium prices, which were stable and very low (~\$6/lb U_3O_8 [uranium oxide]), peaked in 1979 at \$44/lb U_3O_8. Another production peak was reached in 1980 at 66,000 t U/yr. World uranium production largely exceeded military and civil needs of about 20,000 t/yr until 1989. The prices declined to about \$10/lb U_3O_8 from 1990 to 2005, mainly because of the accumulation of huge uranium inventories and the 1993 Megatons to Megawatts Program agreement. This agreement, between Russia, RWE Nukem, AREVA, and Cameco for the dilution of military highly enriched uranium, had delivered the equivalent of 9,000 t of natural uranium per year on the market, until 2013. The decline of production created a strong unbalance (about 20,000 t/yr) with respect to the need of the nuclear reactors, during 24 years. After the oil crisis in 2004 and 2008, uranium production and uranium prices started again to increase, with a short peak at \$137/lb U_3O_8. Then the uranium price declined again because of stabilization followed by a strong decline in oil prices resulting from a world financial crisis, a strong increase of shale gas production in the United States, and the Fukushima nuclear accident. At a price of about \$40/lb U_3O_8, most uranium deposits cannot be mined in economic conditions.

Uranium Geochemistry

The initial uranium concentration for primitive Earth of 7 ppb has been progressively enriched through time, first in the mantle after the segregation of its Ni-Fe core and then in the continental crust by partial melting of the mantle and fractional crystallization. Average uranium concentration in the continental crust is 1.7 ppm and reaches 2.7 ppm in its upper part. Uranium tends to be enriched in the most silicic magmatic rocks (up to several tens to hundreds of parts per million), such as granites and rhyolites and in sedimentary rocks rich in organic carbon (black shale, lignite) or phosphorus (phosphorites). Uranium concentration in seawater is 3.3 ppb and only about one-tenth of parts per billion in lake and river waters.

Uranium has three natural radioactive isotopes: ^{238}U, ^{235}U, and ^{234}U with relative abundances of 99.274%, 0.720%, and 0.0055%, and half-lives of 4.469×10^9, 0.7038×10^9, and 246,000 years, respectively. The present $^{238}U/^{235}U$ ratio is 137.88 everywhere on the earth, except in some ore bodies at Oklo (Gabon, in Africa) where natural fission reactions have occurred. The decays of ^{235}U and ^{238}U are complex, with many intermediate daughter products until the final stable daughter isotope ^{207}Pb in the ^{235}U chain and ^{206}Pb in the ^{238}U decay chain are reached. These daughter elements have variable geochemical properties and thus can be differentially mobilized and dispersed into the environment, resulting in disequilibrium in the radioactivity of isotopes in the decay schemes. This disequilibrium must be assessed for using radiometric measurements in the evaluation of uranium ore grades in mining.

Uranium oxide has the lowest solubility in water in reducing conditions at neutral pH and low temperature. The solubility of uranium minerals in dilute waters in oxidizing conditions is controlled primarily by complexes of the uranyl cation UO_2^{2+}. Its solubility increases markedly in the presence of anions such as F^-, Cl^-, CO_3^{2-}, SO_4^{2-}, and PO_4^{3-}, and with increasing acidity or alkalinity of the waters (Langmuir 1978). For example, uranium is extremely soluble in the strongly acid, oxidizing water commonly associated with acid mine drainage because of the formation of uranyl sulfate complexes.

Uranium Deposit Types

The most recent and most widely used classification of uranium deposits is the one provided by the International Atomic Energy Agency (IAEA) in the last edition of the "*Red Book*" (NEA and IAEA 2016), mostly based on the work of Dahlkamp (2009), with 15 main types and 50 subtypes and classes. The classification is based on deposits listed in the IAEA World Distribution of Uranium Deposits (UDEPO) database (IAEA 2009) according to the nature of host rock lithologies. The 15 types of deposits are listed here from high temperature to surficial environments:

1. Intrusive-type deposits with two subtypes: (1.1) *intrusive anatectic* related to partial melting processes and hosted in granite-pegmatite from very high-grade metamorphic domains (e.g., Rössing and Husab in Namibia); and (1.2) *intrusive plutonic* related to magmatic differentiation processes, subdivided into three classes: (1.2.1) *quartz monzonites* associated with porphyry Cu deposits (Bingham, United States); (1.2.2) *peralkaline complexes* (Kvanefjeld, Greenland); and (1.2.3)

carbonatites (Palabora, Republic of South Africa). Resources are generally large, but the grades are generally low.

2. Granite-related deposits include veins hosted in granite or adjacent rocks and disseminated mineralization in episyenite bodies where quartz has been hydrothermally dissolved. In the Hercynian belt of Europe, these deposits are generally associated with large, peraluminous, two-mica granites. Two subtypes are defined: (2.1) *endogranitic*, occurring within the granitic pluton; and (2.2) *perigranitic*, occurring within the country rocks. Resources are small to large with low to high grades.
3. Polymetallic iron-oxide breccia complex deposits belong to the iron-oxide-copper-gold (IOCG) category. Olympic Dam (Australia) is the only IOCG deposit where uranium is extracted as a by-product of copper and gold. It is the world's largest uranium resource with more than 2 Mt (million metric tons) U at 230 ppm. These deposits occur as huge granite (Olympic Dam) or metasedimentary-metavolcanic (Salobo, Brazil) breccias associated with large quantities of magnetite or hematite. Uranium is disseminated as uraninite, coffinite, and U-Ti oxides.
4. Volcanic-related deposits occur within and near volcanic calderas filled with mafic to felsic volcanic lavas or pyroclastic rocks and interlayers of clastic sediments with (4.1) *structure-bound deposits* where uranium mineralization is controlled by veins and stockworks that may extend deeply into the basement; (4.2) *stratabound deposits* as disseminations in clastic sediments; the largest district of these subtypes is the Streltsovska caldera in Russia; and (4.3) *volcano-sedimentary deposits* where uranium mineralization is nearly concordant, low grade, in carbonaceous lacustrine sediments, with an important tuffaceous component (Anderson mine, United States). The association of U-Mo-F is quite common.
5. Metasomatite deposits occur in rocks affected by metasomatic alteration: (5.1) *Na-metasomatite subtype* (e.g., Novokonstantinovskoye deposit in Central Ukraine); (5.2) *K-metasomatite subtype* (only known in the Elkon District in Russia) that produced albitized or illitized rocks, respectively, along crustal-scale ductile to brittle structures; the resources range from medium to very large with average grades of about 0.1 wt % uranium; (5.3) *skarn subtype* is developed at the contact between granites and enclosing carbonate rocks (e.g., Mary Kathleen, Australia).
6. Metamorphite deposits occur are disseminations, veins, and shear zones within regionally metamorphosed rocks having no known relation to granitic intrusions, with three subtypes: (6.1) rare *stratabound* (e.g., Forstau, Austria); (6.2) *structure-bound* (e.g., Schwartzwalder, United States); and (6.3) *marble-hosted phosphate* (e.g., Itataia in Brazil). These deposits are highly variable in size and grade.
7. Proterozoic unconformity deposits occur below, along, or above the contact between an Archean to Paleoproterozoic crystalline basement and Proterozoic oxidized siliciclastic sediments. Deposits consist of pods or veins essentially consisting of massive uranium oxide. Strong quartz dissolution is generally associated with them. Three subtypes are distinguished: (7.1) *unconformity-contact* (e.g., Cigar Lake, Athabasca Basin, Canada); (7.2) *basement-hosted* (e.g., Ranger in Australia); and (7.3) *stratiform structure-controlled* (e.g., only at Chitral and Lambapur in India). The resources range from medium to large with very high grades in Canadian deposits but low grades in Indian deposits.
8. Collapse breccia pipe deposits are made of down-dropped fragments from overlying lithological sedimentary units filling large karst dissolution cavities developed in carbonate layers. Uranium oxides occur in the breccia matrix with sulfides or oxides of Cu, Fe, Zn, Pb, Ag, Mo, Ni, and Co. They are only known in the United States (Grand Canyon area). Resources are small to medium, but grades average 0.5% U.
9. Sandstone deposits occur in sandstones deposited in continental fluvial or marginal marine environments. They represent more than 600 deposits over the 1,580 registered in the UDEPO database. Five main subtypes are distinguished: (9.1) *basal channel deposits* occur within elongated channels filled with alluvial-fluvial sediments, and uranium is dominantly associated with plant debris (e.g., Khiagdinskoye, Vitim District, Russia); resources are large to medium, at grades ranging from 0.01% to 3% U; (9.2) *tabular deposits* are elongated parallel to the stratigraphy within reduced sediments (e.g., Akouta, Niger; and the Colorado Plateau, United States); individual deposits are small to large, at grades ranging from 0.05% to 0.5% U; (9.3) *roll-front deposits* with a crescent shape ore-body morphology (e.g., Inkai, Kazakhstan; and Crow Butte, United States); resources range from small to very large, with grades varying from 0.05% to 0.25% U; (9.4) *tectonic-lithologic deposits* occur along fault zones and adjacent sandstone beds where uranium is associated to hydrocarbons or detrital organic matter (e.g., Lodève District, France; Franceville Basin, Gabon); deposits are small to medium, and grades vary from 0.1 to 0.5% U; and (9.5) uranium mineralization associated to mafic dykes and sills in Proterozoic sandstones (Matoush, Otish Basin, Canada); small to medium deposits with low to medium grades (0.05%–0.40%).
10. Paleo quartz-pebble conglomerate deposits are characterized by uranium oxides associated with pyrite and detrital minerals, including gold, Fe-Ti oxides, and other sulfides in monomictic conglomerates, older than 2.4 Ga (billion years), deposited as basal (Elliot Lake District, Canada) or intraformational units (Witwatersrand Basin, Republic of South Africa) of fluvial to lacustrine braided streams. Hydrothermal remobilization may be important. Two subtypes are recognized: (10.1) *uranium dominant* associated with rare earth elements and Th (Elliot Lake), and (10.2) *Au-dominant* where uranium is a by-product (Witwatersrand).
11. Surficial deposits occur in near-surface sediments and soils, with the largest ones in calcretes. They occur (11.1) in *fluvial valleys* along drainage channels (e.g., Langer Heinrich, Namibia); (11.2) in *lacustrine-playa sediments* (e.g., Lake Way, Australia), where uranium always occurs as vanadates; (11.3) in *peat bogs* (e.g., Kamushanovskoye, Kyrgyzstan; (11.4) in *karst caverns* (Tyuya-Muyun, Kyrgyzstan); and (11.5) as *pedogenic/fracture-filling* (e.g., Beslet, Bulgaria).
12. Lignite-coal deposits correspond to elevated uranium contents in lignite or coal mixed with detrital minerals (silt, clay) adjacent to carbonaceous mud and silt or sandstone beds, with two subtypes: (12.1) *stratiform*

deposits (the Dakotas, United States), and (12.2) *fractured-controlled deposits* (e.g., Freital, Germany).

13. Carbonate deposits are hosted in limestones or dolostones. Mineralization can be (13.1) *syngenetic stratabound* in carbonate (e.g., Tumalapalle, India), (13.2) in *cataclastic carbonate* (e.g., Todilto Formation, United States), and (13.3) in *karst* (e.g., Pryor Mountains, United States).
14. Phosphate deposits with three subtypes: (14.1) *minerochemical phosphorite* in marine continental-shelf origin containing synsedimentary, stratiform, disseminated uranium in the structure of fine-grained apatite; these deposits represent millions of tons of uranium but at very low grade (0.005%–0.015% U); uranium is a by-product of phosphoric acid production (Phosphoria Formation, Idaho and Montana, United States; Gantour Basin, Morocco); (14.2) *organic phosphorite* in argillaceous marine sediments with uranium-rich fish remains (e.g., Melovoe District, Kazakhstan); and (14.3) *continental phosphate* (e.g., Bakouma, Central African Republic).
15. Black shale deposits include (15.1) *stratiform subtype* in organic-rich marine shale and coal-rich pyritic shale, with synsedimentary uranium adsorbed onto organic material and clay minerals (e.g., Haggan alum shale, Sweden; and (15.2) *stockwork sun-type* (e.g., Gera–Ronneburg District, Germany).

An alternative genetic classification has been proposed by Cuney (2009, 2011) and is based on the dominant process involved in concentrating the uranium in a deposit.

MINING OPTIONS

Underground mining—including the Canadian ores (McArthur River, Cigar Lake, and Rabbit Lake), Cominak (Compagnie minière d'Akokan) in Niger, and other smaller producers—represents a large source of uranium. An interesting development is the underground mining methods used for the very-high-grade ores in Canada, with the McArthur River mine using raise boring, eliminating the need for crushing, and the Cigar Lake mine using jet boring with water to recover the ore directly into a coarse slurry form.

The by-product production segment in Table 1 is primarily from the Olympic Dam operation in Australia that is a copper producer foremost, with significant uranium as a by-product or co-product. In recent years, ISL or in situ recovery has become a very significant source of uranium, most notably from Kazakhstan (Table 2). Uranium is produced not only by various methods but also in a wide variety of geographical locations, with Kazakhstan currently the largest producer.

Table 1 2016 Mining methods for uranium production

Method	Uranium, t	Percent
In situ leach	30,062	48
Underground and open pit (except Olympic Dam)*	29,030	47
By-product*	3,274	5

Source: WNA 2017

*Considering Olympic Dam as by-product rather than in underground category.

Table 2 2016 world distribution of uranium production

Country	Uranium, t	Percent
Kazakhstan	24,575	39
Canada	14,039	22
Australia	6,315	10
Namibia	3,654	6
Niger	3,479	5
Russia	3,004	5
Uzbekistan*	2,404	4
China*	1,616	3
United States	1,125	2
Ukraine*	1,005	2
Other	1,150	2
Total	**62,366**	**100**

Source: WNA 2017

*Estimates.

DEMAND AND MARKET CONSIDERATIONS

Historically, uranium production in developing countries has been dominated by underground or open pit mining and conventional milling. However, experimental ISL of uranium was begun as early as 1961 in Ukraine, and today ISL is the cornerstone of the uranium production industries of Kazakhstan and Uzbekistan. In addition, several other developing countries have either begun ISL operations or are assessing the amenability of their uranium resources to ISL extraction (WNA 2015).

Uranium processing tends to follow basic unit processes, as illustrated in Figure 1. The figure shows the relationships of various mining options to what referred to as "conventional processing."

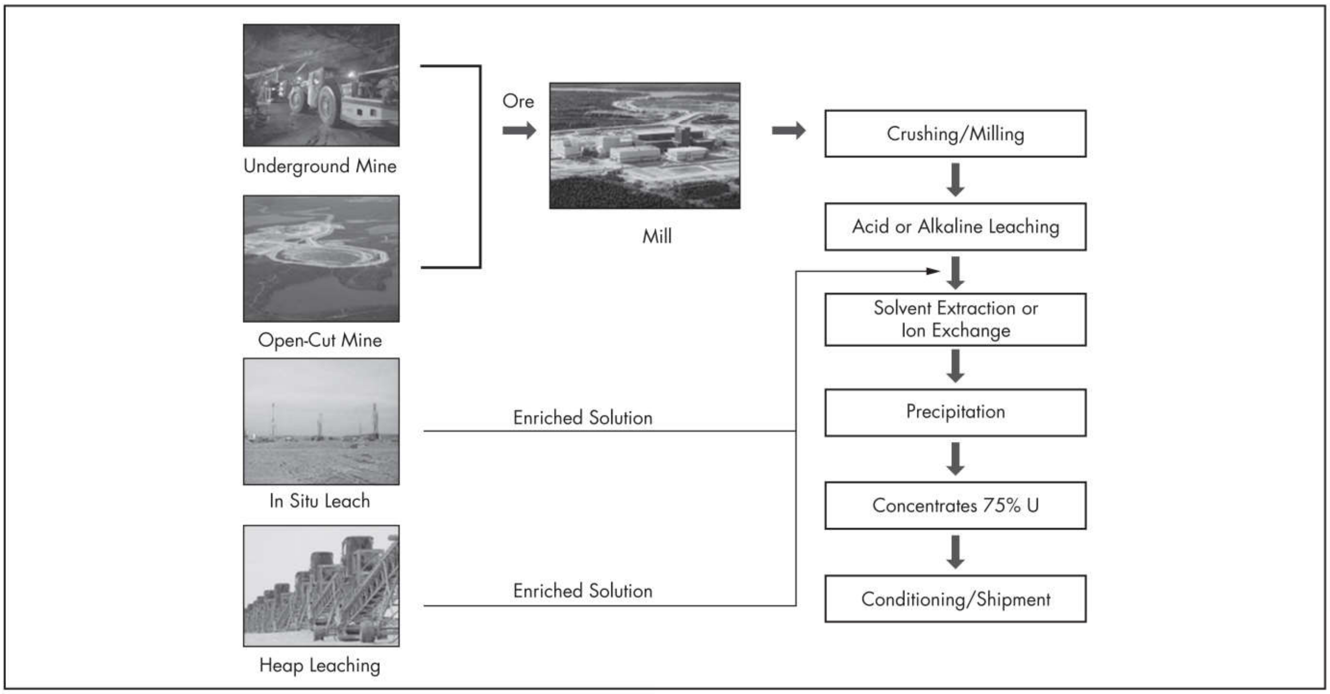
Although ISL has emerged as a prominent production alternative, there is a recent reemergence of heap leaching, which is different from historical uranium heap operations that used "dump leaching."

Markets and Prices

The market for uranium during the period 1945–1970 was primarily for nuclear weapons through various governments. Since that time, the market has consisted solely of nuclear fuel for nuclear power plants through both government and private entities. It is a well-known and easily calculated market given that the number and character of nuclear plants worldwide is well documented. As of mid-2015, there were 447 operable nuclear power plants in 30 countries. Uranium demand in 2015 was estimated to be 63,000 t (WNA 2017). Historical uranium production is illustrated in Figure 2.

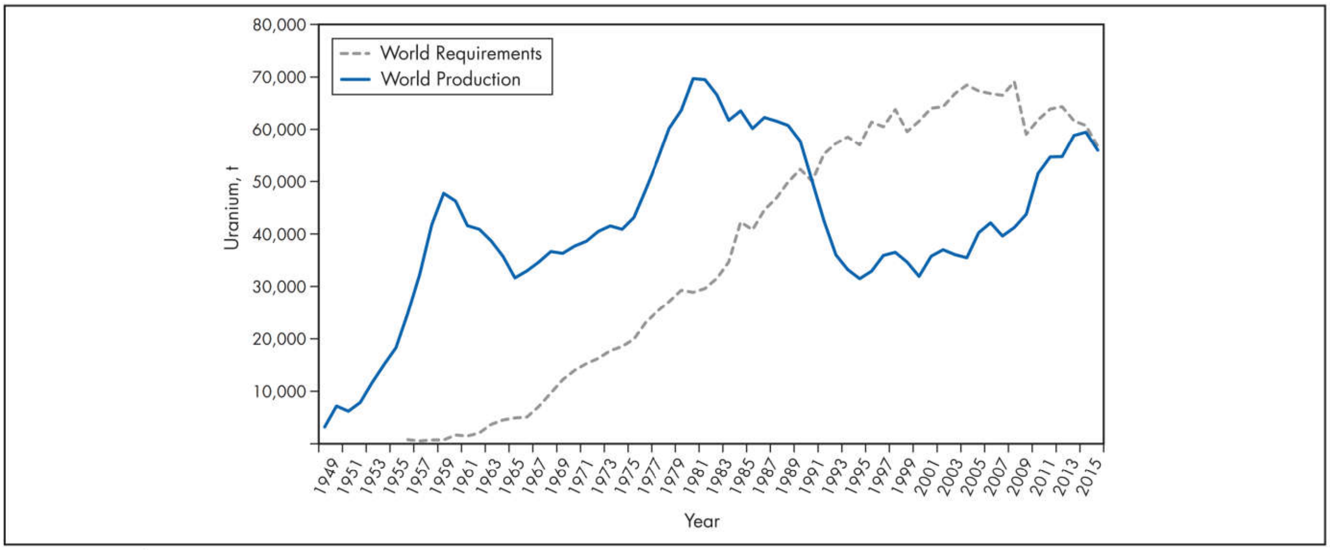
These uranium requirements are filled from two sources: primary production and secondary supply. Primary production consists of newly mined uranium. Secondary supply consists of previously mined uranium in a variety of forms, including inventories, blended-down nuclear weapons, tailings from the uranium enrichment process, and reprocessed spent fuel. Secondary supply currently amounts to about 17,000 t U/yr and is expected to remain relatively constant at this level for the foreseeable future. Thus, its market share is expected to decline. Future uranium requirements, as projected by the IAEA, are set forth in Figure 3.

Sales of uranium to owners of nuclear power plants are made mainly under long-term (5- to 10-year) contracts, in which future delivered prices might be (1) base price escalated, (2) market based, or (3) some combination of the two. Deliveries of uranium under long-term contracts typically



Source: Schnell 2009

Figure 1 General uranium flow sheet



Source: NEA and IAEA 2016

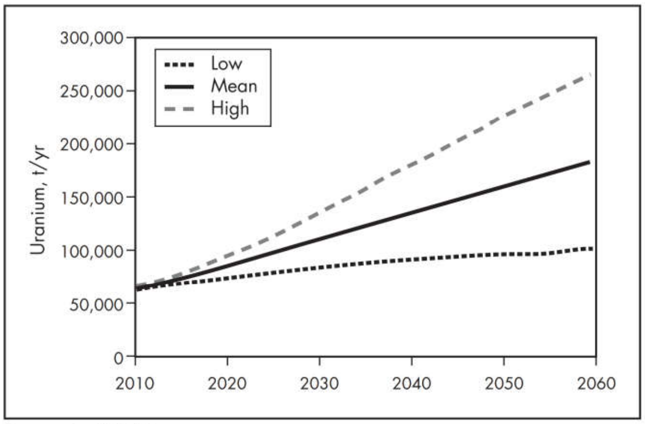
Figure 2 Annual uranium production and demand

cover about 75% of the market, but pricing terms are rarely made public. Prices in the spot market, however, are widely available and form the basis for most analyses. World production of uranium is expressed in metric terms as tons or kilograms of uranium although some producers will use U_3O_8 to exaggerate reserves or production. The spot price, a historical carryover, is expressed as pounds of U_3O_8 (typically in U.S. dollars). (For reference, 1 kg U = 2.6 lb U_3O_8.) Historical spot market prices are shown in Figure 4.

URANIUM UNIT OPERATIONS

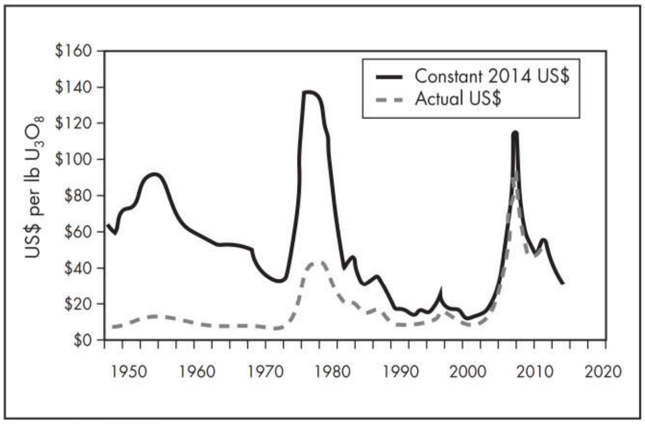
Uranium is a relatively common element in the crust of the earth (very much more than in the mantle). It is a metal approximately as common as tin or zinc, and it is a constituent of most rocks and even of the sea. Some typical concentrations are given in Table 3.

Uranium is found in a large variety of settings; consequently, uranium processing must be adaptive to these settings. Historically, flow sheets based on a tank leach process tended to be very similar and are referred to as conventional. There



Source: Pool 2013

Figure 3 Future nuclear reactor uranium requirements



Source: Pool 2013

Figure 4 Historical uranium price

is a trending away from such conventional flow sheets, or at least the technology within each unit operation is changing. The range of processing now varies from the extreme high-grade deposits in Canada to the other low-grade extremes in places such as Namibia.

When determining a flow sheet for a particular ore, many factors need to be considered: mining method, deposit type, deposit size, mineralogy, uranium grade, geographical location, water balance, production capacity, costs (capital and operating) schedule, and other considerations. Most notably, costs are a result, although the deposit size and production target play an important role. Many of the unit processes depend upon the host rock or gangue characteristics rather than uranium mineralogy.

Mineral Variability

The known types of uranium mineralization are overwhelming because of the large uranium atom with four electrons loosely bound to the nucleus. Uranium is therefore found in a large range of minerals—vanadates, carbonates, oxides, phosphates, silicates, arsenates, sulfates, and molybdates. These minerals are all potential uranium sources, if found in sufficient abundance. The principal minerals exploited today are uraninite (pitchblende), carnotite, coffinite, brannerite, torbernite, and wulfenite.

Process Choice

There are three basic process alternatives—conventional, ISL (or in situ recovery), and heap leach. Although one speaks of conventional, there are no two processing plants that are the same today in the world. Most similar, but still distinct, are the Somair and Cominak operations in Niger. Reviewing the three sets of unit operations, as shown in Figure 5, heap leach has the advantage of not requiring grinding and eliminating solid–liquid separation. In ISL, the mine is eliminated, but it is now the leach section of the plant with less control.

It is commonly thought that ISL and heap leach have much lower costs, both capital expenditures (CAPEX) and operating expenditures (OPEX), but with the complex back end of the process costing about 50% of investment, cost differences of the three basic flow sheets are not huge. Certainly, ISL is the lowest cost option for uranium production, with conventional agitated leach being the highest cost, and the choice must be balanced against risk and recovery.

Table 3 Natural uranium concentration occurrences

Very high-grade ore (Canada), 20% U	200,000 ppm U
High-grade ore, 2% U	20,000 ppm U
Low-grade ore, 0.1% U	1,000 ppm U
Very low-grade ore (Namibia), 0.01% U	100 ppm U
Granite	3–5 ppm U
Sedimentary rock	2–3 ppm U
Earth's continental crust (average)	2.8 ppm U
Seawater	0.003 ppm U

Source: WNA 2016

A comparison of the main variables for each process option is shown in Table 4 and Figure 6. The other considerations are costs, with the key OPEX and CAPEX element proportions surprisingly similar, regardless of location. The highest CAPEX components are the leaching and solid–liquid separation areas. The highest OPEX components are power, labor, and reagents in similar proportions.

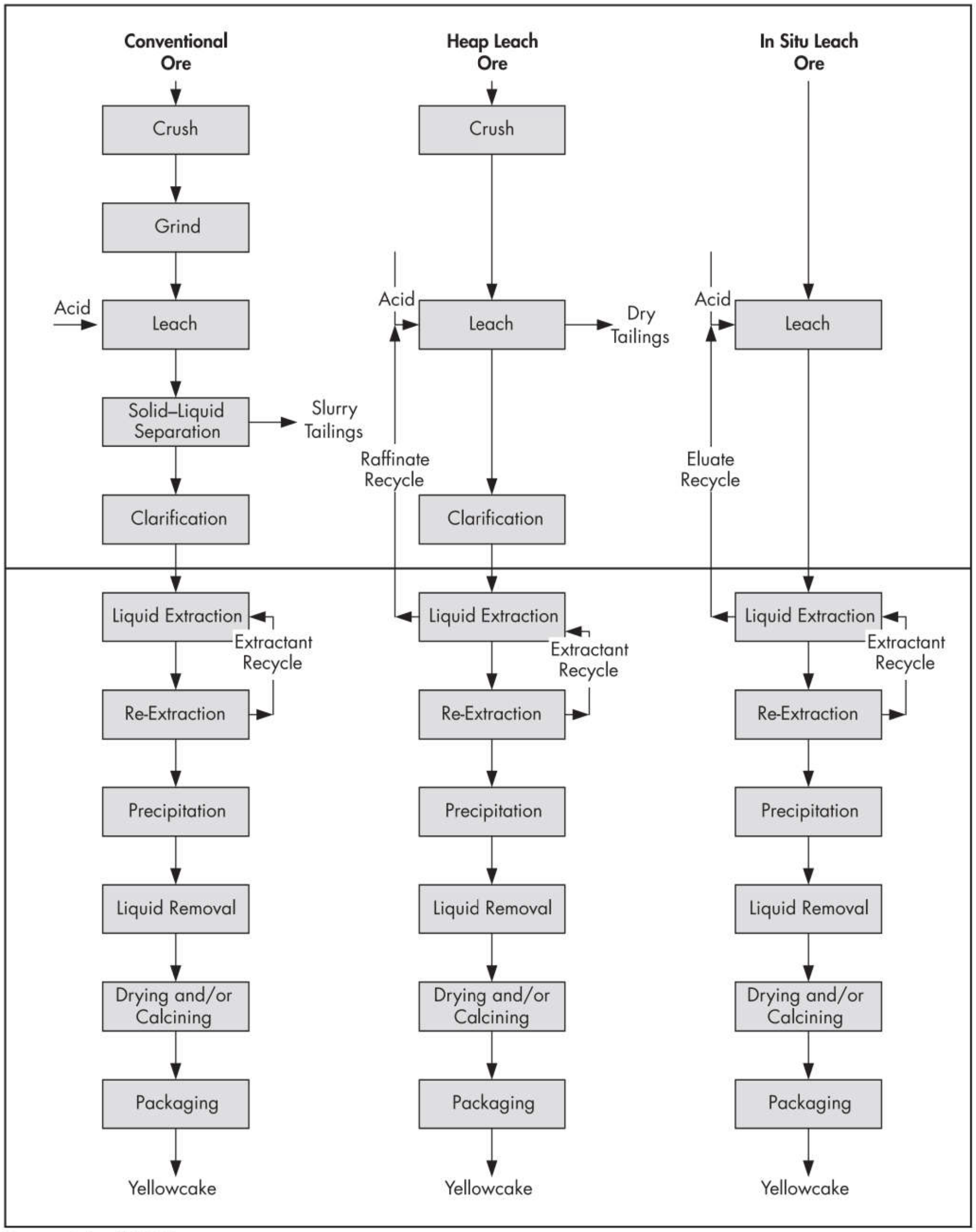
Conventional Uranium Process

The following subsections discuss each unit process, with leaching separated into conventional, heap leach, and ISL. Further technical details can be found in other chapters of this handbook.

Crushing

Uranium most commonly occurs along the fissures or cracks within the ore or gangue matrix, and crushing is preferential at these fissures and cracks in the ore matrix, thereby exposing the uranium minerals. Gyratory crusher installations are no longer typical because costs and uranium throughput tonnages are generally small. Jaw crushers are still common and are used at the Langer Heinrich operation, but there is a trend toward the use of sizers for low-grade operations such as Trekkopje in Namibia or Kayelekera in Malawi due to their lower capital cost.

Sizers require testing with the specific ores and have limited application for hard-rock ores. Under development, with



Source: Schnell 2009

Figure 5 Principle uranium process flow sheets

uses in gold and coal, are high-pressure grinding rolls that pass the ore through a set of opposite rotation rolls under high pressure. This has been considered for uranium, but to date, there are no commercial uranium-based installations.

Crushing is generally limited down to the 10-to-15-mm size, which is usually not sufficient to expose all the uranium minerals.

Ore Stockpile

A stockpile is generally used to separate the crushing operation from the mine and reduce the downstream plant size. Ore stockpiles have been open air using either a reclaim system under the stockpile or a set of feeders, but best practice today is for a covered stockpile requiring consideration for radon buildup in such enclosures while providing personnel access.

Preconcentration

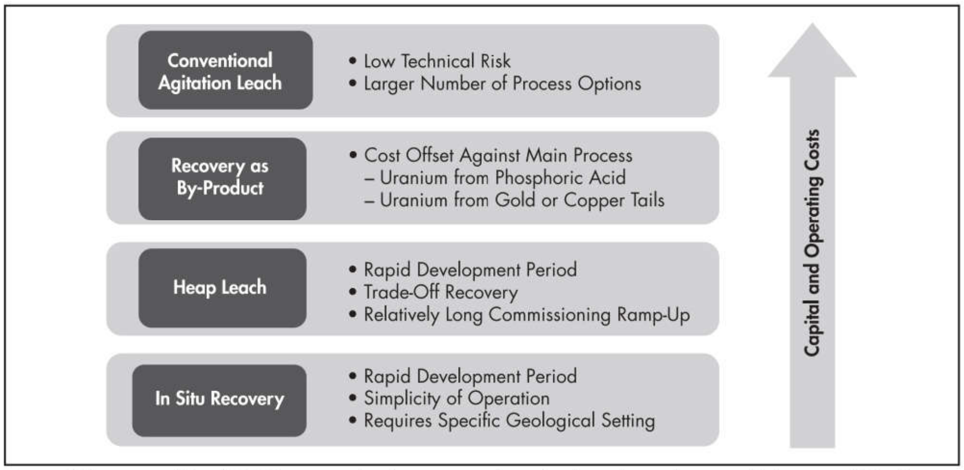
Historically, radiometric sorting has been used to reduce the tonnage to the main processing plant, and this option is still studied, with several pilot ore sorters installed, at the Ranger operation in Australia (ERA 2008) and at Rössing in Namibia (Rössing Uranium Ltd. 2007). However, neither of these installations have been commercially accepted.

The preconcentration step is employed at the Langer Heinrich operation in Namibia, with uranium as carnotite hosted in gypcrete ore. Lunt et al. (2007) use an attrition grind

Table 4 Leach option comparison

Conventional	Heap Leach	In Situ Leach
• Conventional mining • Higher capital cost • Higher operating cost • Best operating control • Short leach times • Best recovery, +95% • Adaptable to most ores	• Conventional mining • Reduced capital cost – No grinding – No solid–liquid separation • Reduced operating cost • Long leach times • Reduced reagent consumption • Reduced leach control • Reduced recovery, ~70% • Not applicable to all ore types	• No mining • Plant capital cost similar to heap leach; reduced overall capital cost • Operating costs reduced (no mining) • Long leach times • Reduced reagent consumption • No real leach control • Recovery questionable • Only applicable to specific ore/geological settings

Source: Schnell 2009



Source: El-Ansary and Lund 2008, reprinted with permission from the Australasian Institute of Mining and Metallurgy

Figure 6 Capital and operating cost considerations for process options

followed by sizing of the various ore fractions to reduce ore tonnage.

Gravity separation has been applied to uranium ores in the past (Schnell 2011) with some success but is associated with serious radiation issues and, consequently, is currently not used. One exception is the use of a Knelson concentrator in the ore receiving circuit of Key Lake to help reduce cement carryover coming from the McArthur River underground mine backfill (Cameco Corporation 2009).

Grinding

Ore sizes of 10–15 mm are then further reduced with the use of grinding mills, but final grind size is coarse (100–500 µm) compared to other metals operations, given that only mineral exposure is required, rather than mineral liberation.

Most grinding operations are wet and then use either coarse ore to grind the fine ore (semiautogenous grinding, or SAG) or steel rods or balls in more conventional grinding. SAG milling is only applicable to specific hard ores after suitable testing. An exception is dry grinding found in the Somair and Cominak operations in Niger. Dry grinding is specific to the Niger ores, which require a very intense leach reagent concentration to liberate the uranium minerals from an expanding clay matrix.

Grinding for high-grade deposits generally uses small autogenous mills, as in the case of McClean Lake in Canada. McArthur River and Cigar Lake operations in Canada have small autogenous mills placed underground to avoid high-grade ore hoisting. In these cases, the ore is transported to surface as a ground slurry with positive displacement pumps. Both operations then transport the ground slurry by truck in special dual-containment containers to mills more than 80 km away.

Closed-circuit operations using cyclones are preferred so that the higher specific gravity uranium minerals will be ground finer than the gangue, improving leaching.

Conventional Leach Circuits

Following comminution, the uranium minerals are exposed and available for dissolution of uranium in a hydrometallurgical process. There are two basic uranium leaching alternatives—acid or alkaline process, with U^{6+} dissolving as either a sulfate or a carbonate.

The gangue of the ore will define whether to use acid or alkaline leaching, primarily depending upon reagent consumption costs. Conventional leaching is most common with sulfuric acid (25–200 kg/t acid) and typically at about 55°C. Higher temperatures, as used in pressure leaching, will result in silica dissolution, making the subsequent unit operations more difficult.

Leaching can be carried out in a large variety of equipment types—heap, vat, agitated, autoclave, and so on—all dependent upon many variables and determined by laboratory testing. The interdependence of time/temperature/pressure results in the increase of one variable while decreasing the other variables. Other variables are similar, but their effects are perhaps not as dramatic.

Table 5 Typical conventional uranium leach parameters

	Typical Leach Parameters	
Parameter	**Acid Leach**	**Alkaline Leach**
Reagent	25–200 kg H_2SO_4/t	$Na_2CO_3/NaHCO_3$
Time	8–12 hours	48–96 hours
Pressure	Atmospheric	Pachuca or autoclave
Oxidant	H_2O_2, O_2, MnO_2	O_2
Temperature	50°–60°C	85°–90°C
Recovery	93%–97%	85%–90%

The basic acid leach chemistry dissolves the U^{6+} directly, while the U^{4+} species is oxidized. Typical oxidants in use are sodium chlorate, manganese dioxide, hydrogen peroxide, and oxygen. Oxygen has become most desirable because of low-cost oxygen production using vacuum swing adsorption technology. There is a general move away from the use of chlorates due to environmental and corrosion issues. Pyrolucite (MnO_2) is still common where it is economically available. Traditionally, the oxidation process found in most textbooks depends upon first oxidizing a ferrous iron to ferric and then the ferric oxidizing the uranium, although there are ores where direct oxidation takes place. The simplified chemistry of alkaline leaching is oxidation followed by formation of a uranyl–carbonate complex.

Acid leach reactions are as follows:

$$UO_2^{2+} + SO_4^{2-} \leftrightarrow UO_2SO_4$$

$$UO_2SO_4 + SO_4^{2-} \leftrightarrow [UO_2(SO_4)_2]^{3-}$$

$$[UO_2(SO_4)_2]^{3-} + SO_4^{2-} \leftrightarrow [UO_2(SO_4)_3]^{4-}$$

$$6Fe^{2+} + NaClO_3 + 6H^+ \leftrightarrow 6Fe^{3+} + NaCl + 3H_2O$$

$$UO_2 + 2Fe^{3+} \leftrightarrow UO_2^{2+} + 2Fe^{2+}$$

Alkaline leach reactions are as follows:

$$2UO_2 + O_2 \leftrightarrow 2UO_3$$

$$UO_3 + Na_2CO_3 + 2NaHCO_3 \leftrightarrow Na_4UO_2(CO_3)_3 + H_2O$$

Continuous tank leaching typically uses an agitated system of no fewer than five or six reactors in series to eliminate short-circuiting. Typical acid leach times are 8–12 hours in a 35%–50% solids slurry at 50°–60°C. Most leach circuits are arranged to advance the slurry by gravity. Some leach circuits use a two-stage leach so that the first stage uses the discharge solution from the second stage to help reduce acid consumption, providing a pre-leach. Typical parameters for conventional leaching are given in Table 5.

An alternative to the typical agitated tank leach are the Somair and Cominak operations that use a pug leach process. Using this method, the dry ore is combined with concentrated sulfuric acid and nitric acid in a drum and then the moist mixture is cured using a slow-moving conveyor for 2 hours before entering agitated tanks and diluted before solid–liquid separation (Thiry and Roche 2000).

Carbonate leaching has reemerged with the start-up of the Langer Heinrich operation in Namibia, and Trekkopje will be the first application using carbonate in a heap leach operation (Schnell 2010).

Leaching equipment ranges from pressure leach (Key Lake and McClean Lake) to agitated tank leaching (Langer Heinrich, Rössing, etc.) and a recent return to pachuca leaching (Cameco Corporation 2012).

Leaching of uranium would not be complete without considering the pugging-and-curing flow sheet used at the Somair and Cominak operations in Niger. These ores in Niger contain the mineral structure of siliceous clay that requires a high-acid curing step to achieve good metal recovery.

Heap Leach

Although some heap leach work was carried out in the 1970s and 1980s, uranium heap and dump leaching did not develop as it did in gold and copper, primarily because of market conditions and the availability of higher-grade deposits. Recent increased interest in lower-grade deposits and the efforts of countries to secure internal uranium production has resulted in heap leach application in uranium production. Heap leaching based on the fine crushed, agglomerated technology for copper operations in Chile with sulfuric acid has been applied for many years at the Caetité operation in Brazil (Gomiero et al. 2010) and more recently in Niger at Somair for the processing of mineralized waste (Durupt et al. 2013). In addition, the large Imouraren project in Niger will also be using crushed agglomerated heap leach (Thiry and Bustos 2014). Many junior companies have been looking at acid heap leaching to exploit low-grade deposits, and more heap leach uranium operations will be seen in the future as the application of this technology to uranium matures. The Trekkopje project is unique in applying crushed agglomerated heap leach in carbonate (alkaline) leach chemistry.

The objectives of a good heap leach operation are

- Recoverable metal to solution of 80% plus,
- Leach times of <300 days,
- Results that are consistent and independent of location within the heap,
- Virtually no solids in the pregnant leach solution, or PLS (no requirement for solid–liquid separation step),
- Little sensitivity to head grade, and
- Heap stability.

The most difficult objective is achieving the “consistent and independent of location” result within a heap. This challenge requires that leach conditions are the same everywhere on the heap and the ore is also equally responsive.

When considering the heap leach process option, the following must be true:

- Months (years) of careful column test work have confirmed good results in all areas of the ore body.
- The ore body has been carefully sampled for variability.
- Initial test work is generally performed on a blended sample, but variability testing is required to better predict actual performance.

The general operating conditions for uranium heap leach are as presented in Table 6.

Heap leaching seems simple, but there are some pitfalls, such as poor heap permeability due to compaction or jarosite formation. Compaction can cause possible increased phreatic head, resulting in heap failures; heap stability and extraordinary measures are required to correct such problems. Solution flow considerations, including rain events, require a design that considers the inherent solution energy and avoids solution discharges into the environment. Figure 7 presents a typical acid heap leach flow sheet.

Table 6 General uranium heap leach operating conditions

Description	Acid Heap Leach	Alkaline Heap Leach
Typical uranium grade	200–500 ppm	150–500 ppm
Crush size	10–25 mm	10–25 mm
Clay content	<35%	<35%
Agglomeration time	45–90 seconds	45–90 seconds
Agglomeration moisture	7%–10%	7%–10%
Heap height	6–9 m	7–9 m
Irrigation rate	4–10 L/min/m^2	5–10 L/min/m^2
Leach reagent	10 g/L H_2SO_4	40 g/L CO_3 and 10 g/L HCO_3
Leach cycle	1 or 2 countercurrent	3–4 countercurrent
Leach time	30–90 days	120–180 days
Recovery	80%–85%	70%–80%

In Situ Leach

The in situ mining method for uranium is in sandstone deposits with typical low-grade uranium concentrations of (0.06%–0.2% U). ISL is currently responsible for more than 40% of world uranium production. Mining and uranium leaching take place in the deposit.

In order to consider an ISL option, the deposits must be located in a permeable formation, water should normally be over the formation, and the deposit must be located between confining layers to be able to contain the solutions. Figure 8 shows a conceptual model of a typical ISL formation as found in Kazakhstan, Australia, and the United States.

The leach process is conducted within the ore formation, and the leach chemistry is either with acid or carbonate solutions, similar to conventional uranium leach procedures but with much lower reagent concentrations and requiring very long leach times, in the three-to-five-year time frame. Specifically, there are two leach options: (1) Oxygen, carbon dioxide, and sodium bicarbonate are added to the native groundwater, or (2) weak sulfuric acid is pumped into formation, and at times with use of an oxidant. Recoveries should normally be in the +75% range.

The PLS from the well fields will have also leached other contaminants, mostly iron in the case of acid leach since the leach solution is low grade. Ion exchange is normally used to fix the uranium to a resin with the PLS feeding IX columns. The uranium is complexed with an organic radical and then is absorbed onto styrene beads. Most operations in Australia and the United States use fixed columns, while pinned-bed columns are used in Kazakhstan. The uranium is then eluted from the resin with a carbonate of acid solution. This elute maybe then be precipitated directly (alkaline leach) or forwarded to solvent extraction (SX) for iron removal or a double precipitation (pH ~3.4 to precipitate iron, followed by pH +8 for uranium precipitation) may be required prior to final yellowcake product precipitation. The water from ISL needs to be treated before being recycled back into the aquifer, or some bleed may need to be discharged into the environment. The available alternatives for water treatment are deep well disposal, reverse osmosis treatment, or chemical treatment.

The environmental considerations for ISL can be onerous. In the case of alkaline leach, the well field needs to be washed or rinsed after leaching is complete to reduce contamination. Reverse osmosis is used to recycle this rinse solution over a period of several years. For acid ISL, the dissolved iron will precipitate as the leach solution advances out of the well field, creating a natural bearer for solution contamination. When the groundwater remediation is complete, piping is removed, topsoil is replaced, and the area is revegetated.

The advantages of ISL are that the employees have minimal radiation exposure, ground surfaces have little permanent disturbance, capital and operating costs are low, and low-grade ore bodies can be economically mined. But one must keep in mind that this method is only applicable to select types of deposits contained in permeable sandstone.

Solid–Liquid Separation

After leaching the uranium into a soluble complex, it must be separated from the residue with a physical separation using either decantation or filtration. The solid–liquid separation choices for uranium are dependent on the characteristics of the leached pulp—basically, how the pulp filters or decants. The options used are numerous, such as filtration (belt, drum, pressure), countercurrent decantation (CCD), countercurrent cyclones, countercurrent classifiers, or resin-in-pulp (RIP).

RIP has recently been applied in the Kayelekera mine in Malawi, with the development of modern resins where the uranium complex is adsorbed or complexed with a resin and the pulp is then screened to separate the resin bead (larger in diameter than the ore solids) by screening (Figure 9).

Solution Clarification

Uranium operations generally have low throughput compared to base metal operations, and, subsequently, clarification is a viable option in most conventional and heap leach operations. The leach solutions (PLS) will commonly contain fine solids that needs to be clarified prior to solution purification. Conventional clarifiers, sand filters, and sand filtration are typical. Some continuous sand filtration types have shown poor availability, but the use of pinned-bed clarifiers is being more recently applied for high-PLS flows (Thiry and Bustos 2014).

Thickening

Settling of solids from solution typically uses a thickener or decanter or settler (different terms are often used for the same equipment) with the solids raked into the center and with the uranium-bearing solution overflowing. The decantation circuits operate as CCD units, typically using five to eight thickeners in series with a water consumption of 1–2 m^3/t. A recent variation has been applied at the McClean Lake operation in Canada with the use of "high rate" thickeners where the pulp bed is partially used as a filter to improve solid–liquid separation, thereby reducing thickener diameters.

Filtration

Filtration is another option to separate the uranium solution from the pulp and wash the pulp to reduce uranium losses. A wide variety of filtration equipment is available. Most commonly used in uranium processing are horizontal belt filters as used at the Somair and Cominak operations in Niger. The solids from a belt filter are typically discharged directly from the filter onto a conveyor belt to send the washed pulp to a dry tailings disposal area.

Solvent Extraction

Various other metals are dissolved during leaching, requiring purification of the uranium-bearing solutions. The purity requirements of the final uranium precipitate are becoming

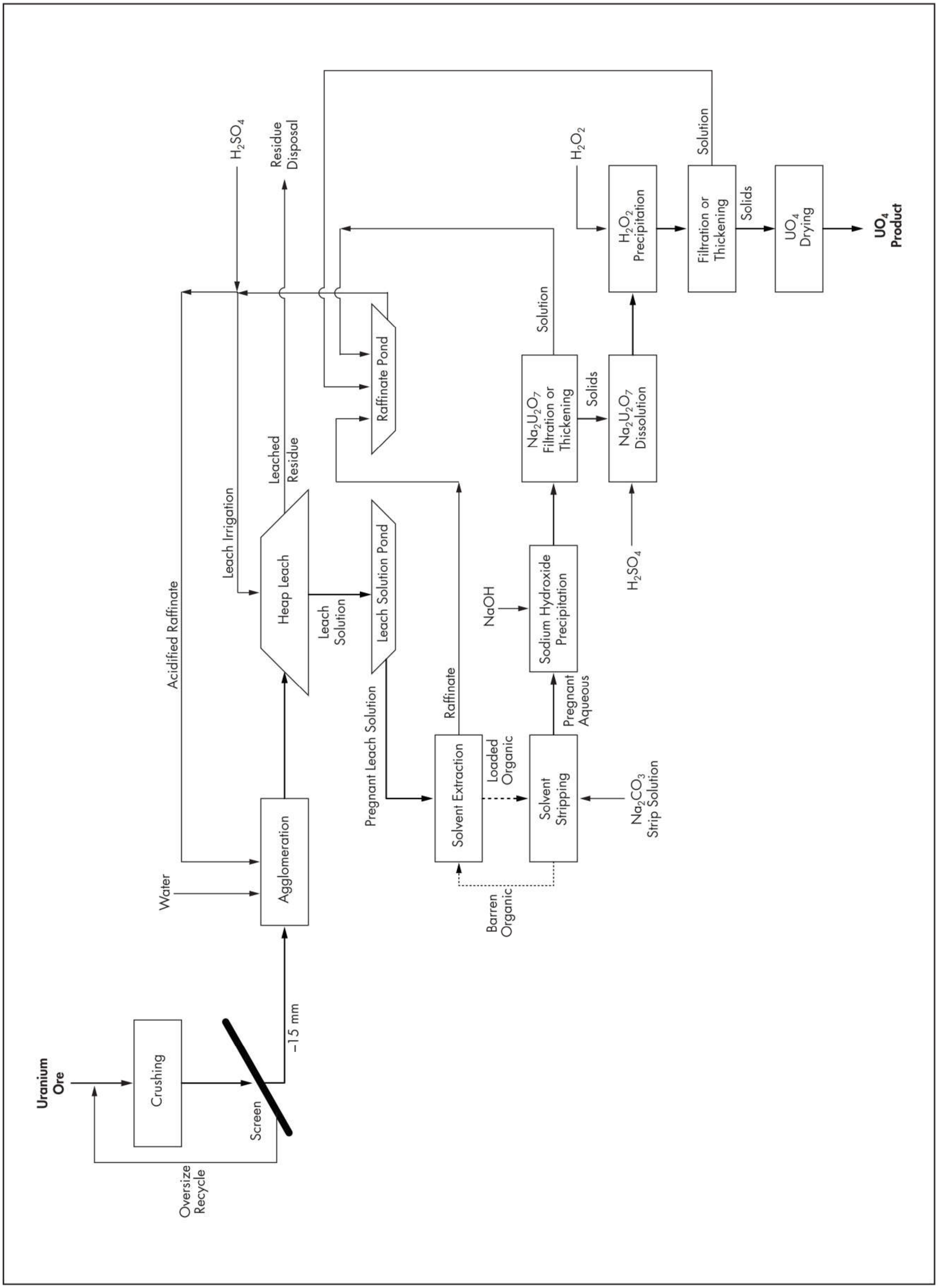
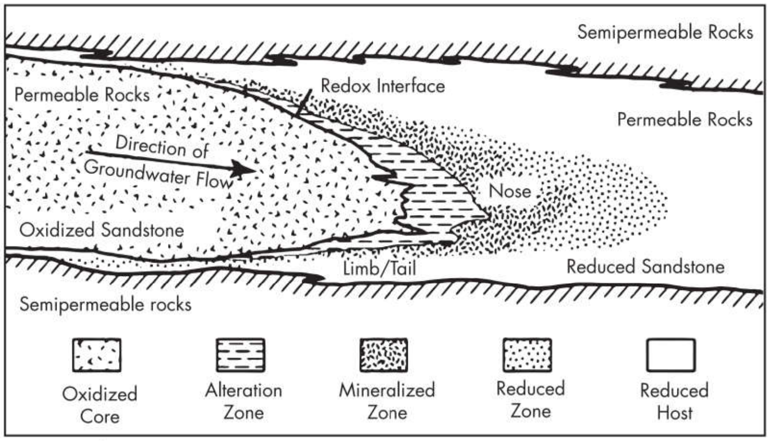
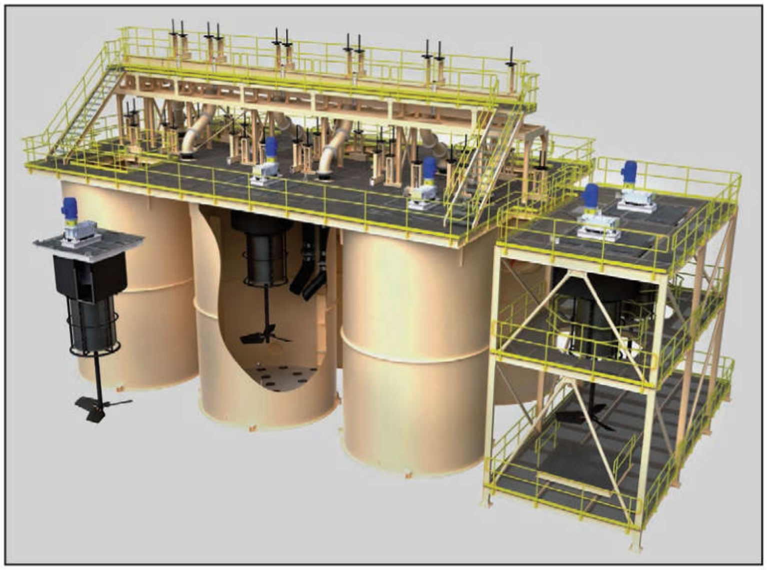


Figure 7 Acid heap leach flow sheet



Source: Cyclone Uranium Corporation 2012

Figure 8 Conceptual model of a roll-front deposit



Source: Kemix Pty Ltd. 2015

Figure 9 Resin-in-pulp example

stricter, and most uranium conversion plants require a precipitate quality better than the ASTM International standard. The most common impurities associated with uranium are molybdenum and vanadium, but care must be taken to also consider other metal contaminants, most notably zirconium, arsenic, copper, and nickel.

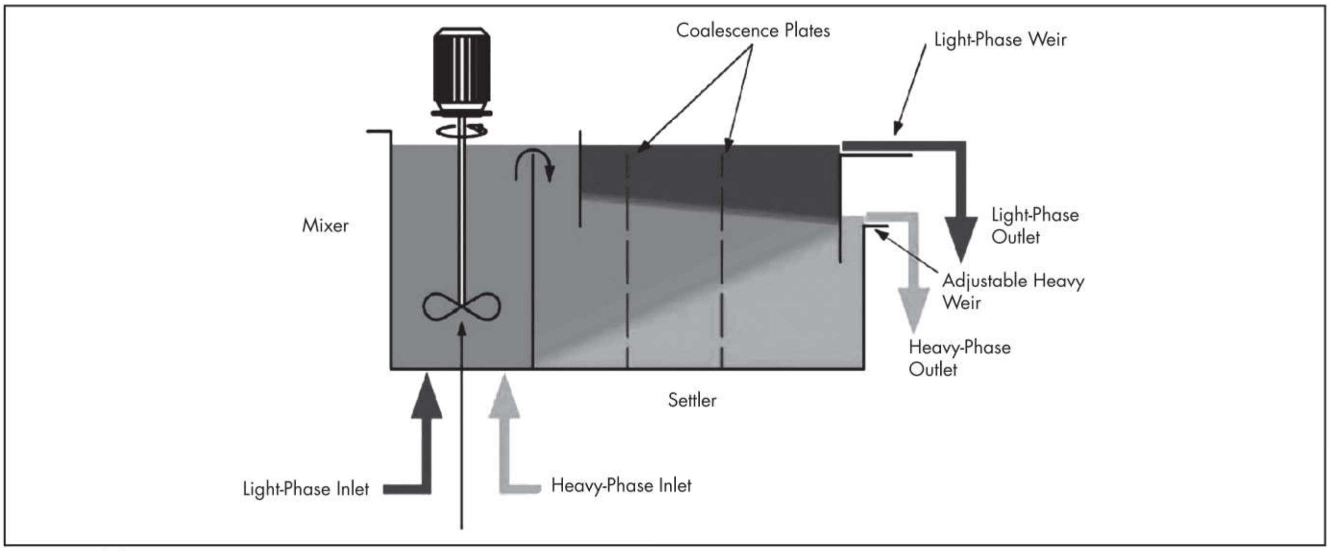
Solvent extraction is typical for uranium, but little progress has been made for extractants and modifiers and the tertiary amines predominate with some use of the organophosphorus compound DEPHA. Equipment is usually conventional mixer-settlers or Krebs mixer-settlers, as used in Canada (McClean Lake and Key Lake; see Figure 10). At Olympic Dam in Australia, Bateman columns have been applied (Bateman Litwin 2012). Future projects are also considering vertical smooth flow technology (Paatero et al. 2010).

The uranium stripping in solvent extraction is with ammonium sulfate (Key Lake, McClean Lake), sodium carbonate (Somair), or strong acid (Rabbit Lake). In one interesting alternative, the uranium is precipitated directly in the organic extractant. Extractants are generally tertiary amines with typical concentrations in the 5% amine/g/L U levels, although for higher-grade solutions, extractant concentrations can be in the 10% amine/g/L U. Stripping generally produces 35 g/L U concentrations but can be up to 100 g/L for high-grade operations. Figure 11 depicts a typical solvent extraction pulse column.

Ion Exchange

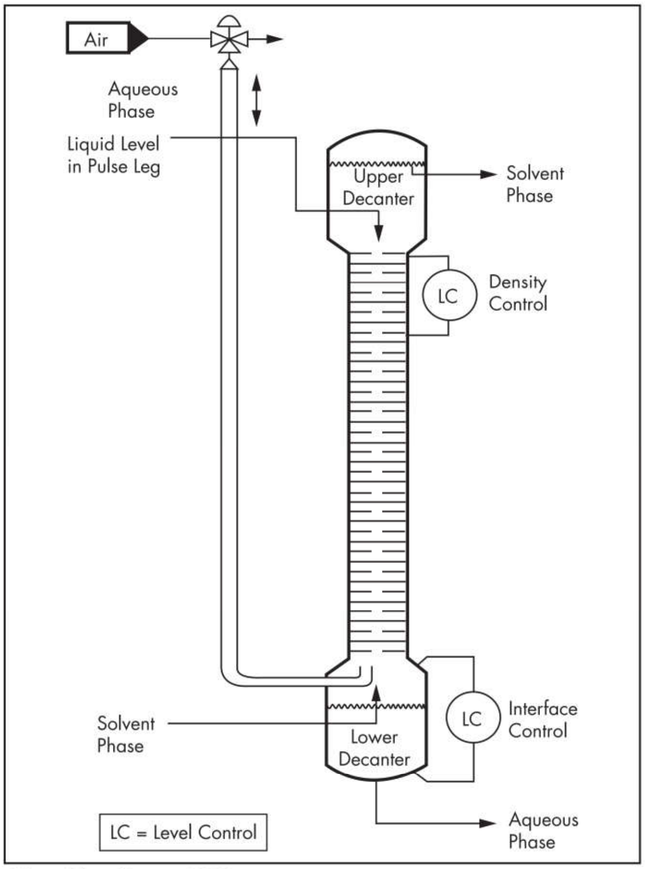
The IX process has progressed both in the resin and equipment design areas and in the recovery of uranium from lower-grade deposits. ISL operations with lower-grade solutions make ion exchange the preferred choice.

Normally, ion exchange is chosen for uranium solution grades at the <1.0-g/L range; however, slightly lower solution grades may use the SX option. In the case of ion exchange, additional solution purification may be needed, which usually adds a small SX circuit, a process known as the Eluex process



Source: Perdula 2013

Figure 10 Typical solvent extraction mixer-settler



Adapted from Vancas 2003

Figure 11 Solvent extraction pulse column

(Merritt 1971). The most notable example of an Eluex circuit is the Rössing operation in Namibia (Griebel 2000). Basic IX and SX reactions are shown in Figure 12.

Three basic technologies are used for uranium ion exchange: fixed-bed columns, pinned-bed columns, and fluid-bed columns (NIMCIX [National Institute for Metallurgy Countercurrent Ion eXchange]). Fixed-bed columns have been applied in the Langer Heinrich operation, with pinned-bed predominant in the Kazakhstan ISL operations, and NIMCIX has been installed at Trekkopje and the Langer Heinrich operations in Namibia. NIMCIX offers the advantage of handling high suspended solids.

Anionic Exchange	$R^+X^- + A^- \Leftrightarrow R^+A^- + X^-$
Cationic Exchange	$R^-Y^+ + B^+ \Leftrightarrow R^-B^+ + Y^+$

Figure 12 Basic ion exchange and solvent extraction reactions

Precipitation

After the uranium solution has been purified and concentrated, the uranium can be precipitated. The process choice for final uranium precipitation is dependent upon solution chemistry, environmental requirements, reagent availability, and converter requirements. The most common reagents used are ammonia, hydrogen peroxide, sodium hydroxide, and magnesium oxide. However, most converters prefer oxide precipitates rather than an alkali precipitate because the latter results in additional environmental concerns.

Equipment choice most commonly includes a series of stirred tanks, but recent innovation using a fluid bed (Courtaud et al. 2011) produces an improved product with less dusting. One unusual case involves precipitating the uranium directly in SX organic (Kazakhstan).

The various chemistries of precipitation are as follows:

- Magnesia precipitation

$$UO_2SO_4 + 2Mg(OH)_2 \rightleftarrows MgUO_4\downarrow + MgSO_4 + 2H_2O$$

$$UO_2SO_4 + Mg(OH)_2 \rightleftarrows UO_2(OH)_2\downarrow + MgSO_4$$

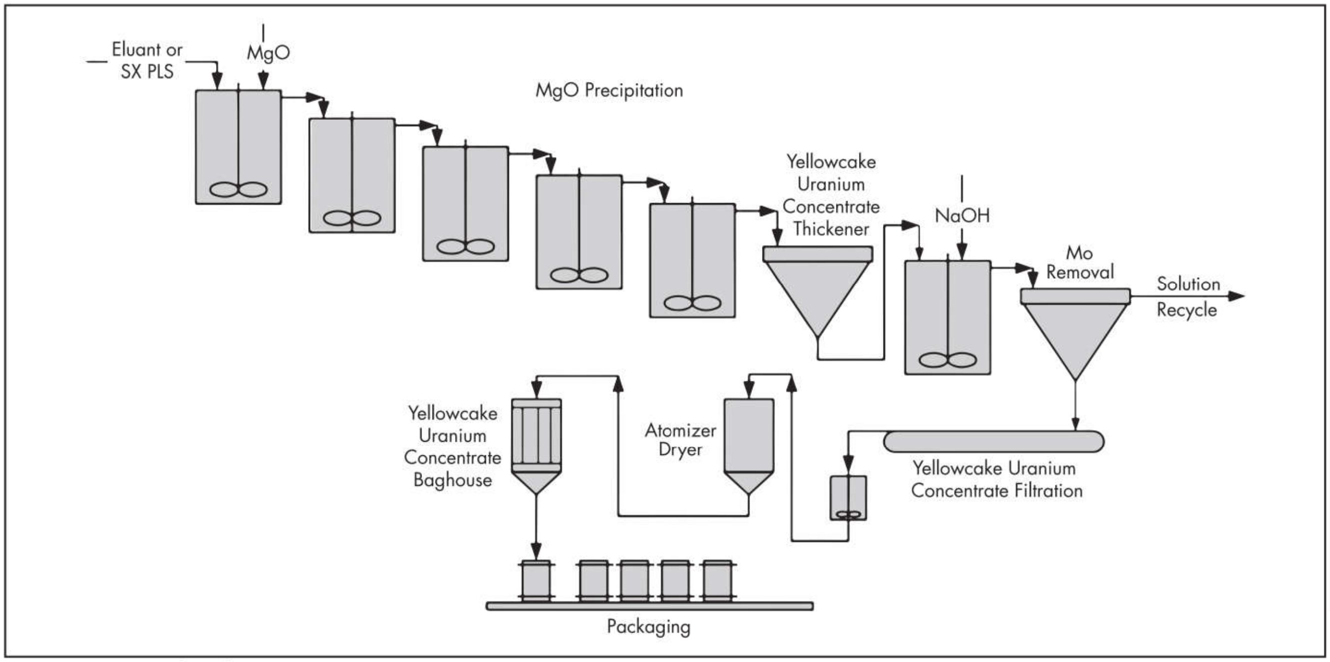


Figure 13 Typical tank precipitation circuit

- Sodium precipitation

$$2Na_4UO_2(CO_3)_3 + 6NaOH \rightleftarrows Na_2U_3O_7\downarrow + 6Na_2CO_3 + 3H_2O$$

$$Na_4UO_2(CO_3)_3 + NaOH \rightleftarrows UO_2(OH)_2\downarrow + 3Na_2CO_3$$

- Hydrogen peroxide precipitation

$$UO_2SO_4 + H_2O_2 + 2\ NaOH \rightleftarrows UO_4{\cdot}2H_2O\downarrow + Na_2SO_4$$

- Ammonia precipitation

$$2UO_2SO_4 + 6NH_4OH \rightleftarrows (NH_4)_2U_2O_7\downarrow +2(NH_4)_2SO_4 + 3H_2O$$

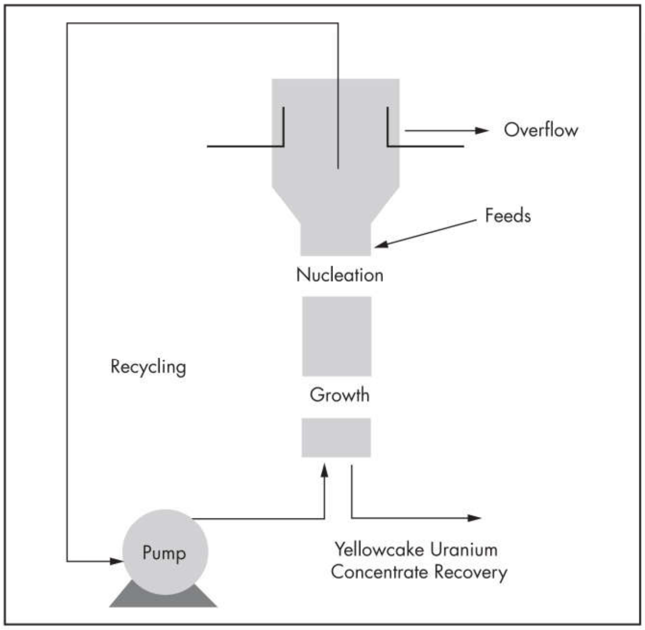
$$UO_2SO_4 + 2NH_4OH \rightleftarrows UO_2(OH)_2\downarrow +(NH_4)_2SO_4$$

Illustrated in Figure 13 is a typical tank precipitation circuit with thickener operation to recycle precipitate and feed a filter before final precipitate drying. Figure 14 shows a schematic of fluid-bed precipitation.

Drying and Packaging

The precipitate is generally dewatered before drying with use of a centrifuge or filter (horizontal filter or pressure filter). The final dewatered precipitate is then dried or calcined with the final product and is referred to as "yellowcake." Commercially it is then marketed as U_3O_8, although there is a general attempt to express everything as uranium rather than U_3O_8. The color of the precipitated uranium (from ammonia, magnesia, or peroxide precipitation) is dependent upon the final drying temperature—yellow at 100°C, orange at 250°C, and very dark green (black) at higher than 800°C, as illustrated in Figure 15 for ammonium diuranate. The final product has a uranium concentration above 75% U.

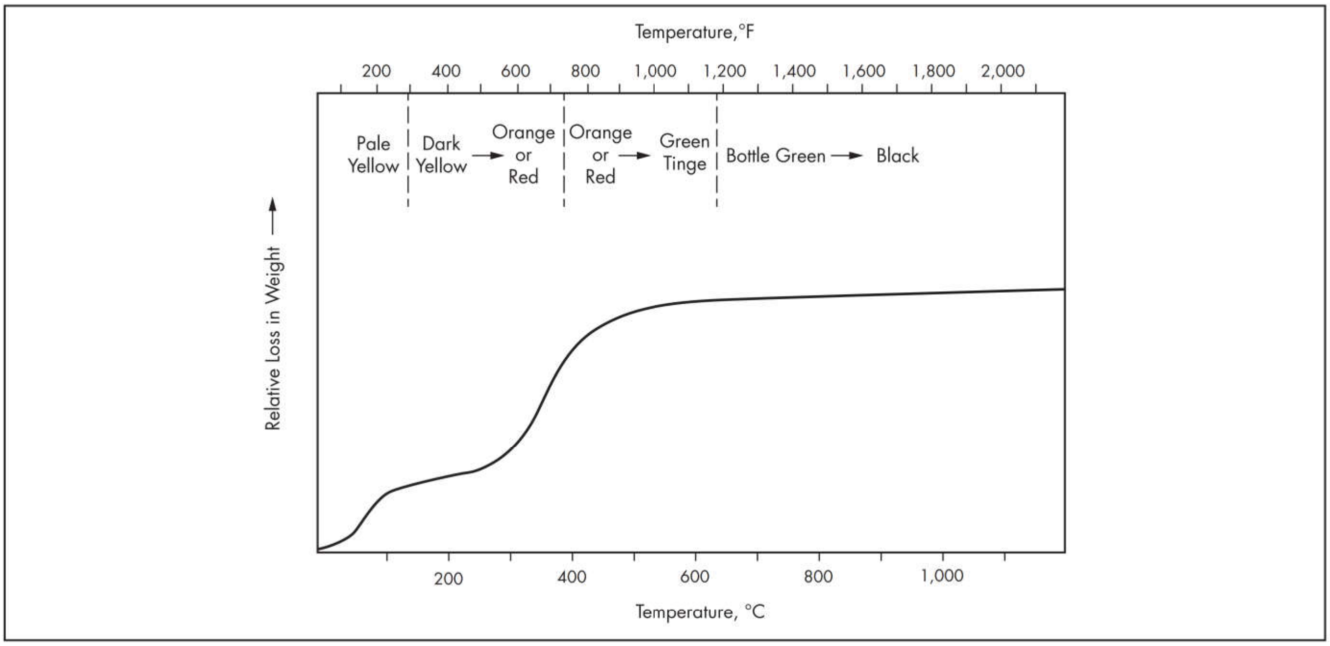
Product drying and/or calcining equipment ranges from multi-hearth furnaces (Key Lake and McClean Lake), to rotary tube dryers and calciners (Kazakhstan), to hollow Flyte dryers (Rabbit Lake and Langer Heinrich), to spray dryers (Cominak). The choice of calcining or drying to a lower temperature is dependent upon economics and the destination of final product.



Adapted from Courtaud et al. 2011

Figure 14 Fluid-bed precipitation

In all cases, the final precipitate is packed into standard 170-L steel drums for shipping. Packaging equipment and installations are being upgraded to reduce personnel exposures to yellowcake dust. With these upgrades, facilities are generally located in a separate enclosed space with remote operation and drum lids added with glove-box type



Source: Merritt 1971

Figure 15 Effect of temperature on the decomposition of ammonium diuranate

of equipment. Table 7 lists comparative converter yellowcake quality requirements.

Effluent Treatment

The plant product is yellowcake, but the wastes from the process must also be considered. There are basically two waste streams to consider: waste/process water and tailings from leaching. Both of these streams, and other less important streams, require environmentally acceptable treatment and disposal. Some of these waste streams also have value that must be considered. For example, all solutions in alkaline leach contain reagent values that can be recycled.

Liquid wastes must be treated to neutralize them, remove radioactive species such as radium, and eliminate other metals dissolved in the process. The tailings pulp needs similar treatment and neutralization. In Canada, waste solutions are treated and temporarily stored, then analyzed to ensure environmental compliance before the water is released into the environment. These streams are heavily monitored to confirm compliance and minimize environmental effects.

In warmer climates with high evaporation rates, evaporation ponds are typically used to treat liquid wastes. The residues in these ponds then need recovery and suitable long-term disposal. Ponds have a risk of leakage and are typically double lined with a leak detection system.

Tailings Solids

The tailings solids require final disposal after treatment. Traditionally, dammed structures have been used for tailings disposal and final dewatering. These structures have associated risks, and, more recently, an in-pit disposal or lined cell disposal has become a requirement. The schematic of an in-pit tailings disposal as practiced in the most recent operations (Rabbit, McClean, and Key Lakes) is shown in Figure 16.

URANIUM TESTING AND FLOW-SHEET DEVELOPMENT

Several key steps are required in establishing the processing route for a new uranium project. Before preliminary laboratory test work is undertaken to recover the uranium, a thorough understanding of the deposit's mineralogy is necessary, including both the uranium and gangue composition. The uranium grade, liberation characteristics, gangue associations, and grain size will all guide selection of the required leach conditions and leaching method (e.g., tank versus ISL).

It is important to also understand how the uranium and gangue mineralogy may vary through the deposit, as this will influence the sample selection for a test-work program. The leach test conditions should be optimized with consideration given to the variations in a deposit; therefore, testing should include samples representing this variability to determine the response of the leaching process to different feeds. Comminution studies play a crucial role in further understanding the physical characteristics of an ore body, including the potential to upgrade an ore through rejection of gangue minerals by scrubbing. It may also be possible to concentrate the uranium through methods such as radiometric sorting and gravity separation, or potentially problematic phases may be able to be rejected through processes such as flotation.

In most instances, the uranium and gangue mineralogy will define whether a sulfuric acid or carbonate leach is most suitable for uranium extraction; however, it is prudent to test and assess both systems in the first instance. Laboratory test work is then required to test the suitability of the chosen lixiviant, determine the maximum uranium extractable, and establish the preliminary leach parameters. These would include pH, temperature, particle size, leach duration, and oxidation–reduction potential. The composition of the expected site water and its impacts on leaching and reagent consumption, particularly for alkaline leaching, should also be investigated.

Table 7 Comparative converter yellowcake quality requirements

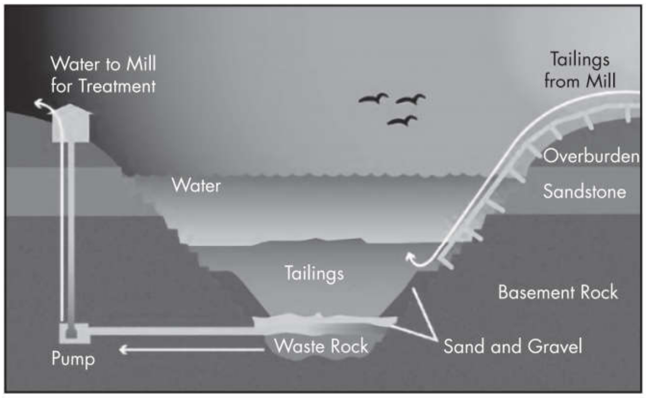
Element	Unit	ASTM Standard		Converter A		Converter B		Converter C	
U	% dry	>65%		>60%		>65%		>65%	
Isotope	%	0.7105–0.7115		0.7105–0.7115		0.7105–0.7115		0.7105–0.7115	
U234	ppm/U	56	62	56	62	58	62	62	
Particle size	mm	<6.35		<6		<6.35		—	
U insoluble	%	<0.10		<0.10		<0.10		—	
Limits	**Unit**	**Penalty**	**Reject**	**Penalty**	**Reject**	**Penalty**	**Reject**	**Penalty**	**Reject**
Moisture	%	2.00	5.00		10.00	5.00	10.00		5.00
As	%/U	0.05	0.10	1.00	2.50	0.05	0.15	0.01	0.40
B	%/U	0.005	0.10		0.20	0.01	0.15	0.01	0.10
Ca	%/U	0.05	1.00	1.00	5.00	3.00	4.00		1.00
CO_3	%/U	0.20	0.50	2.00	3.00	2.00	4.00	0.20	0.50
F	%/U	0.01	0.10	0.15	0.30	0.10	0.15	0.01	0.10
Halogen	%/U	0.05	0.10	0.15	0.25	0.10	0.20	0.05	0.10
Fe	%/U	0.15	1.00			1.00	2.00	0.15	1.00
Mg	%/U	0.02	0.50			3.00	4.00	0.02	0.50
Mo	%/U	0.10	0.30	0.10	0.30	0.10	0.30	0.10	0.30
P	%/U	0.10	0.70		0.32	0.50			0.20
PO_4	%/U	*~ 0.30*	*2.15*		1.00			0.10	1.00
K	%/U	0.20	3.00			+Na=1	+Na=2	0.20	3.00
SiO_2	%/U	0.50	2.50	0.50	2.50	0.50	2.00	0.50	2.50
Na	%/U	0.50	*7.50*	1.00	*7.50*	+K=1	+K=2	0.50	*7.50*
S	%/U	1.00	4.00	1.00	3.33	1.00	3.50		
SO_4	%/U			3.00	10.00			3.00	12.00
Th	%/U	1.00	2.50	1.00	4.00	0.50	2.00		
Ti	%/U	0.01	0.05			0.05	0.10	0.01	0.05
V	%/U	0.06	0.30		0.08	0.10	0.50		
V_2O_5	%/U	*~0.21*	*1.07*		0.30			0.10	0.75
Zr	%/U	0.01	0.10	0.20	2.00	0.10	0.50	0.01	0.50

Note: Numbers in italic represent preferred levels.

Typically, laboratory test work would commence as bench-scale tank leaches, which would be used to optimize the leach conditions and establish typical reagent consumptions. The results from these initial tests would also guide the anticipated leaching method. Although tank leaching is the most typical mode of leaching, heap leaching may be feasible for lower-grade ores with low acid consumption, with mineralogy that allows sufficiently porous heaps to be stacked. The amenability of a deposit to an ISL operation is dependent on the geology of the deposit, with smaller, porous sandstone deposits most suitable. If a heap leach or ISL process option was under consideration, a series of bottle roll, or preferably column leach tests, would be necessary. If more aggressive leach conditions are required, autoclaves may be used to investigate pressure leaching.

The fundamental purpose of laboratory-scale test work is to determine optimal leach conditions, maximum recoveries, and expected reagent consumptions. All this information is critical to understanding the viability of a uranium project. Depending on the results from preliminary leach test work, it may be necessary to investigate methods such as resin-in-pulp or resin-in-leach to counteract preg-robbing of uranium from liquors by ores containing clays or organic materials.

At this stage, it is necessary to consider the choice of a suitable oxidant, if one is mandatory. This will depend on the amount required to obtain an acceptable uranium extraction, as shown by the leach tests, the cost, and the proximity of the source of a supplier. For example, if a suitable manganese mine is close to the operation, then it is likely that the use of pyrolusite, MnO_2, may be feasible. If only a small amount of oxidant is required, then it may be reasonable to use a more-expensive, faster-reacting reagent. If initial leach tests show low iron dissolution, then it may also be necessary to investigate the addition of ferric iron to leach liquors to promote



Source: CNSC 2015

Figure 16 Example of in-pit tailings disposal

U^{4+} oxidation. The concentration of impurities in the leach liquors, and their potential effects in building up in concentration, need to be evaluated, with the subsequent requirement of bleed streams.

Another important factor to consider is the material handling characteristics of ores and slurries. Rheological studies at varying solids density should be performed on leach feeds and residues to identify potential issues with agitation and pumping of slurries, which can occur in ores with high clay content. In addition, the amenability of the slurry to solid–liquid separation, and the suitability of filtration or CCD, should be determined from filtration and settling tests.

Once the appropriate leaching and solid–liquid separation routes have been determined, concentration and subsequent precipitation of the uranium can then be tested using standard methods. The composition of the leach liquor will influence the selection of the concentration method. Ion exchange is typically favored for liquors with lower uranium tenor; however, it is sensitive to species that also load onto the resin, such as ferric iron. Liquors with high chloride concentrations are also generally not amenable to ion exchange. Solvent extraction is the most typical concentration method employed and is a robust process, although problems can arise for liquors with high silicon concentrations, with colloidal silica causing the formation of stable emulsions. Elements such as molybdenum, vanadium, and zirconium in leach liquors can also present challenges. Both ion exchange and solvent extraction are tested by generating adsorption isotherms to identify the impact of liquor composition on extraction and to identify the number of stages required. Equilibrium data generated for ion exchange can then be fed into appropriate models. Loading and elution kinetics must also be determined. The eluent or stripping reagent to recover the uranium from the chosen process must be selective and also not strip species that will be deleterious to the precipitated uranium product.

The final stage of processing—precipitation of the uranium as uranyl peroxide, sodium diuranate, or ammonium diuranate—is a standard process. If selection of the concentration method and stripping reagent is appropriate, then uranium can typically be effectively recovered without excessive levels of undesired impurities. Alternatively, it may be possible to selectively precipitate uranyl peroxide directly from the leach liquor without a concentration step. Uranium precipitates should be assayed to ensure that the product meets the specifications required for the converter to which the product would be sold.

Following evaluation of the preceding bench-scale batch work, it is then necessary in the next phase to move to mini-plant work to test and refine the process and identify any unforeseen challenges.

ACID LEACH OPERATIONS

Key Lake Mill for McArthur River Mine Ore

The Key Lake mill in Northern Saskatchewan's Athabasca Basin is operated by Cameco Corporation. Ownership is by Cameco Corporation with 83.33% and AREVA Resources Canada Inc. with 16.67%. The licensed annual production capacity is 10,000 t U. Since production start-up in 1983, Key Lake's lifetime production at the end of 2014 was 185,000 t U—about 210,000 t U by the end of 2017 (Edwards 1992; Cameco Corporation 2015).

The Key Lake mill process was designed to deal with the relatively high uranium ore grade and the presence of significant levels of arsenic and nickel sulfides (Neven and Gormley 1982; Neven et al. 1985). Particular attention was paid to effectively sequestering arsenic by ensuring a molar Fe/As ratio of 4 or more and maintaining a proper pH profile through contaminant precipitation operations (Krause and Ettel 1987).

Currently, McArthur River mine ore is processed in the Key Lake mill. A simplified flow sheet is shown in Figure 17. Run-of-mine (ROM) ore slurry is received by B-train truck from the mine. On arrival at Key Lake, the high-grade ore slurry is removed from the truck containers and mixed with a low-grade gravity concentrate from ground special waste rock. This dilutes the McArthur River ore slurry, grading on average 15% U_3O_8 down to 5% U_3O_8 for mill feed. This blended mill feed is transferred to storage pachucas in the mill and then pumped to leaching.

Key Lake mill benefited from the January 2002 replacement of the McArthur River grinding circuit classification screens with cyclones. With cyclones, the pitchblende in the ore is preferentially ground finer than 75 μm. In addition to improvements in ore slurry handling, shipping and unloading, and radiation safety, uranium recovery at Key Lake increased by approximately 1% (Edwards and Dyck 2004).

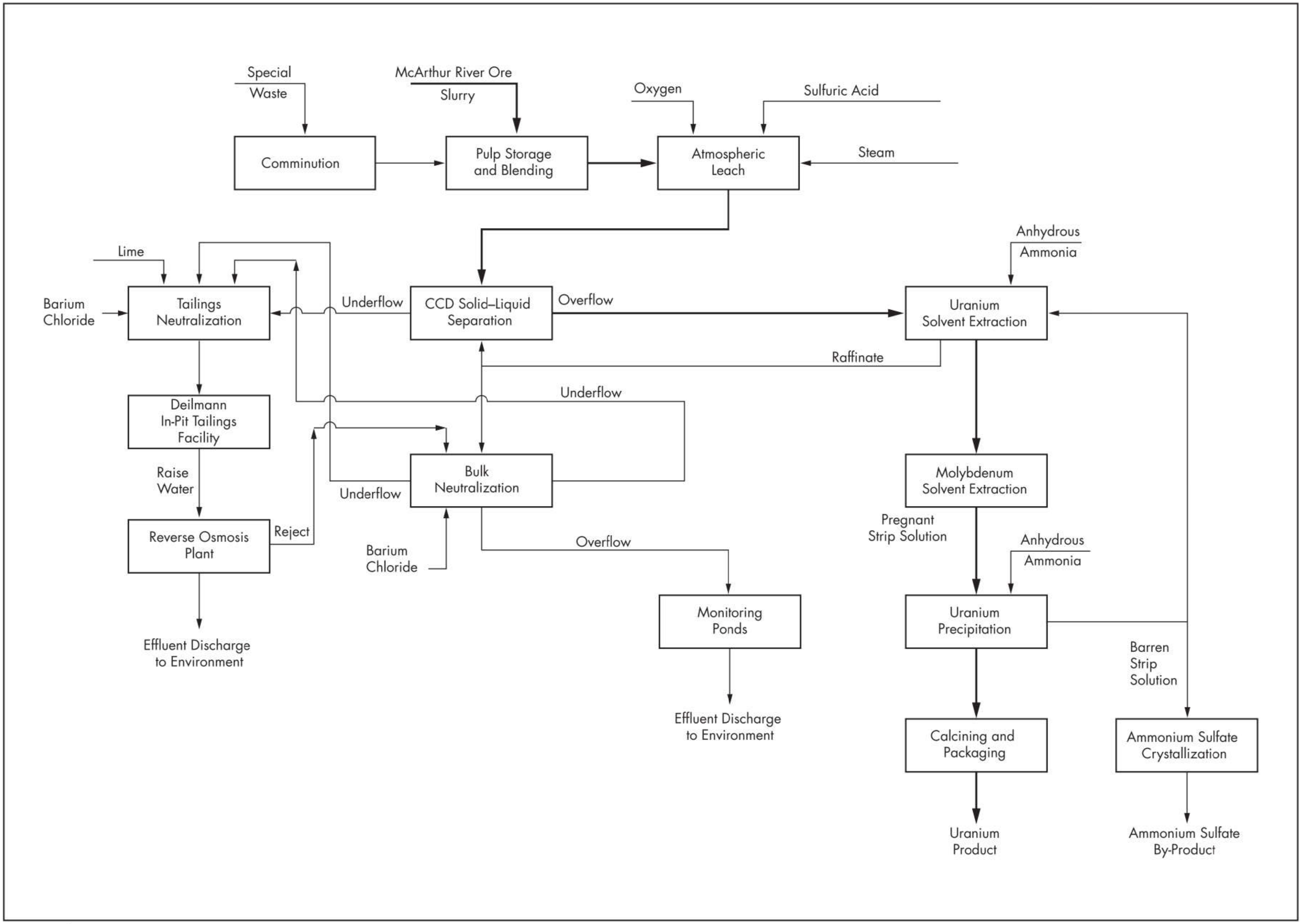
Sulfuric acid leaching is carried out in pachucas followed by continuous stirred-tank reactors, with oxygen as oxidant. Steam injection is required to maintain leach temperature. Two autoclaves, previously used for leaching, are on standby if extra leaching capacity is needed. Leach circuit discharge is pumped to the CCD circuit. Final CCD underflow slurry is pumped to tailings neutralization. Final CCD overflow is pumped through sand filters for clarification. The clarified solution is SX feed. The SX plant uses tertiary amine in a kerosene solvent with isodecanol modifier and ammonia stripping in a series of mixer-settlers. Pregnant strip solution from the uranium SX plant is fed to a smaller SX plant to remove molybdenum. This plant uses LIX 63 in a kerosene solvent to extract molybdenum (U.S. Patent No. 5,229,086; Lam et al. 1993). Uranium is precipitated from the cleaned pregnant strip solution by ammonia addition, forming ammonium diuranate. Washed and dewatered precipitate is calcined and packaged in standard 210-L steel drums.

To prevent ammonia contamination of mill effluent, a crystallization plant removes ammonium sulfate from the barren strip solution. The crystallization plant produces a quality product that meets the specifications for sale as ammonium sulfate fertilizer.

SX raffinate is neutralized with slaked lime. Overflow solution from the bulk neutralization thickener is treated for radium removal with barium chloride. Bulk neutralization and radium removal precipitates, collected as thickener underflow slurry, are disposed in the Deilmann in-pit tailings management facility via the tailings neutralization circuit. Overflow from the radium removal thickener flows to a series of monitoring ponds. These ponds hold effluent until assays on the contents of each pond confirm that effluent specifications are met, and only then are the ponds individually discharged (Neven et al. 1986; Rodgers and Sarion 1998; Rodgers 1996, 2000; Bharadwaj et al. 2010; Lien 2010).

McClean Lake Mill

The McClean Lake mill, referred to as JEB Mill in Northern Saskatchewan's Athabasca Basin, is operated by AREVA Resources Canada Inc. Ownership is by AREVA Resources



Courtesy of Cameco Corporation

Figure 17 Simplified flow sheet for Key Lake mill processing McArthur River ore

Canada Inc. with 70%, Denison Mines Inc. with 22.5%, and OURD (Canada) Co. Ltd. with 7.5%. The licensed annual production capacity is 5,000 t U. Since production start-up in 1999, the JEB Mill lifetime production at the end of 2016 was approximately 30,000 t U (Edwards 1992; Pacquet 2017; CNSC 2012; Denison Mines Corporation 2015) before the Cigar Lake mine commenced shipment of ore to McClean Lake.

Although the McClean Lake mill is based on the same chemistry as the Key Lake mill, McClean Lake's design is not a clone of Key Lake. One major difference is that the radiation protection installations at Key Lake limit maximum mill feed grade to 5% U_3O_8, yet the McClean Lake mill was designed for feed grades as high as 35% U_3O_8 (30% U), in anticipation of processing Cigar Lake ore (Badea and Schwartz 2000; Carino 2001; McKinnon 2000; St-Pierre and Huffman 2000).

Cigar Lake mine ore is currently processed in the McClean Lake mill (Figure 18). ROM ore slurry is received by B-train truck from the mine. The average ore grade is 18% U_3O_8. Sulfuric acid leaching is carried out in continuous stirred-tank reactors, with hydrogen peroxide as oxidant. With higher uranium and arsenic contaminant grades in the feed, cooling is required to maintain leach temperature. Leach circuit discharge is pumped to the CCD circuit. Final CCD underflow slurry is pumped to tailings neutralization. Final CCD overflow is pumped through sand filters for clarification. The clarified solution is SX feed. The SX plant uses tertiary amine in a kerosene solvent with isodecanol modifier and ammonia stripping in a series of mixer-settlers. Impurity molybdenum is removed from pregnant strip solution using activated carbon columns. Uranium precipitation is by ammonia, forming ammonium diuranate. Washed and dewatered precipitate is calcined and packaged in standard 210-L steel drums.

To prevent ammonia contamination of mill effluent, a crystallization plant removes ammonium sulfate from barren strip solution. The crystallization plant produces a quality product that meets the specifications for sale as ammonium sulfate fertilizer.

SX raffinate is pumped to tailings neutralization. Neutralization is with slaked lime. Tailings neutralization precipitates are disposed in the JEB in-pit tailings management facility. Water recovered from tailings slurry densification is pumped to the water treatment circuit. Ferric sulfate is added to remove dissolved heavy metals. Dissolved radium is precipitated with barium chloride. Neutralization is with slaked lime. Initially, treated water was pumped through sand filters prior to entering the monitoring ponds, a practice first used at Rabbit Lake (Edwards 1987, 2002b; Milde 1989). Currently, the effluent sand filters are bypassed. The monitoring ponds hold effluent until assays on the contents of each pond confirm effluent specifications are met, and only then are the ponds individually discharged (Edwards 1997, 2002a; Edwards and Schnell 2000; Remple and Schnell 2000; Stewart 2010).

Kayelekera Mine—Crush/Grind/Leach/RIP/Double-Precipitation Operations

The Kayelekera mine is located in Malawi about 52 km west of Karonga on the northern tip of Lake Malawi. Uranium mineralization was first identified in the North Rukuru River valley in Malawi in 1957, but it was only following an airborne radiometric survey in 1977 that Agip located several radiometric anomalies in the area. In 1983, the United Kingdom's Central Electricity Generating Board (CEGB) was granted a reconnaissance license, and a year later, when they realized that there could be a significant deposit of uranium at Kayelekera, they applied for a three-year Exclusive Prospecting Licence.

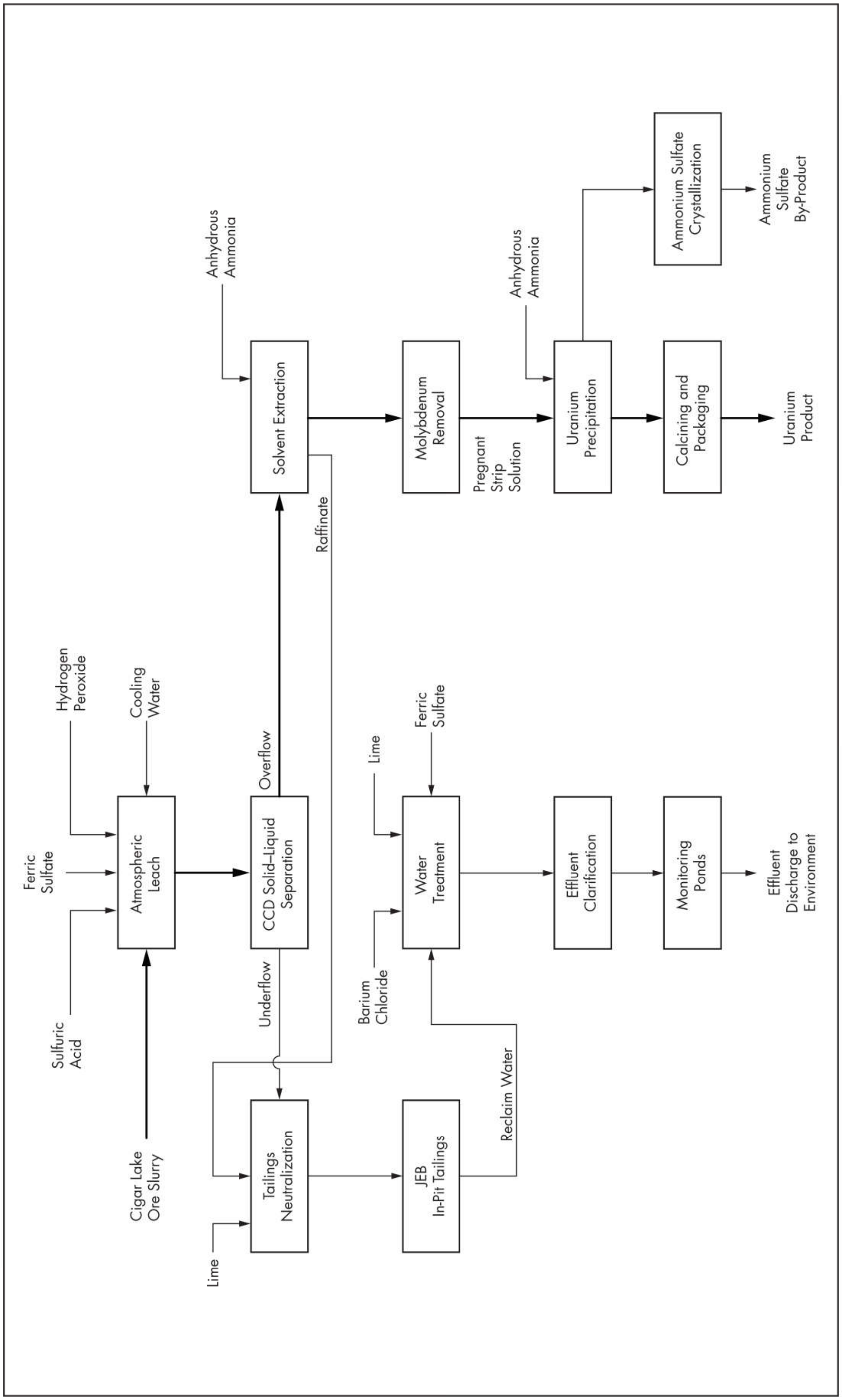
The Karoo rocks of the North Rukuru Basin consist of a succession of clastic sediments, overlain by a series of thick arkosic sandstones and mudstones. The succession is indicative of cyclic sedimentation within a broad, shallow, intermittently subsiding basin. Each cyclotherm passes upward from coarse reduced-facies arkose, through oxide-facies mudstone, to carbonaceous silty mudstone with coal-rich horizons. The arkose units, on average, are 8 m thick. They consist of coarse-grained quartz and fresh feldspar (albite) together with biotite and muscovite. The carbonaceous mudstone units contain discrete quartz grains with calcite veining and laminated carbonaceous pyritic black shale.

Most of the uranium mineralization is contained in the arkose units, but some secondary mineralization occurs in the mudstones. Coffinite is the main primary uranium mineral in the reduced zone; it is often associated with organic debris and pyrite and it is finely intergrown with chlorite/clay that fills the interstices. Extremely fine-grained uranium oxide and a uranium-titanium mineral have been identified in minor quantities in the transitional zone. In the oxidized zone, several secondary uranium minerals, including meta-autunite, boltwoodite, and uranophane, are the result of oxidative weathering of the primary uranium minerals.

A prefeasibility study, including preliminary laboratory testing, confirmed the potential of the deposit, and a full feasibility study was completed by Wright Engineers in 1991 for the CEGB. However, the low uranium price and the privatization of the CEGB at that time led to the project being placed on a "minimum cost" basis. Paladin Resources purchased a controlling interest in the deposit in 1998. It updated the earlier feasibility study and undertook a bankable feasibility study that led to a development agreement with the government of Malawi. The environmental impact assessment was approved and a mining license was granted in 2007.

The study was based on a conventional acid leach, CCD, SX, and ammonium diuranate flow sheet. However, following test work at ANSTO (Australian Nuclear Science and Technology Organisation) and an optimization review, it was found that a process using resin-in-pulp (RIP) and hydrogen peroxide precipitation offered lower capital and operating costs; better plant operability; and reduced safety, health, and environmental concerns. This innovative flow sheet represented the first RIP plant to be built in more than 50 years and the first ever in the Western world to treat high-density slurry (over 30% solids). It also included the use of hydrogen peroxide as the leach oxidant with multistage addition of acid and oxidant.

In 2009, a plant with a capacity of 1.3 t/yr U was commissioned. It was designed to treat 1.5 Mt/yr of ore at a grade of ~1,100 ppm U_3O_8. After open pit mining, the ore is delivered to dedicated stockpiles classified by ore type and grade. The ore is then blended to control the plant feed grade and to maintain the mudstone component to between 10% and 20% of the ROM feed. A front-end loader transports the ore from each stockpile to a dump hopper fitted with a grizzly. Ore is drawn from the hopper using a variable-speed apron feeder that discharges into a rotary tooth mineral sizer. The mineral sizer was subsequently supplemented with a jaw crusher to provide greater crushing flexibility. The crushed ore, cyclone



Courtesy of Cameco Corporation

Figure 18 Simplified flow sheet for McClean Lake mill processing Cigar Lake ore underflow, and water are fed to a ball mill with grinding media added to the mill feed conveyor as required.

The mill discharge, after removal of scats and grinding media on a trommel, is pumped to a hydrocyclone cluster operating in closed circuit with the mill. The target grind size is P_{100} of 300 µm (P_{80} of 180 µm). Water is added to the mill discharge to control the cyclone feed density. The cyclone overflow is directed to a trash screen and then gravitates to the pre-leach thickener where flocculant is added to the feed launder. The thickener overflow gravitates to a process water tank for reuse in the mill, and the underflow is pumped to the leaching circuit.

Slurry feed from the pre-leach area, liquor from the tailings thickener overflow, and leach spillage all flow into the leach feed dilution box. From this box, slurry is fed to a series of six mechanically agitated leach tanks (with bypass launders). Sulfuric acid can be added to the first four leach tanks and hydrogen peroxide (H_2O_2) to tanks 2, 3, and 4 to control both the pH and oxidation–reduction potential. Slurry leaving the leach gravitates onto a linear screen to protect the RIP plant by removing any particles coarser than 400 µm. Screened slurry is pumped to the first RIP contactor.

The RIP tanks are laid out for carousel operation so that any tank can be used as the feed or tail contactor and any tank can be taken off-line and drained for recovery of the resin. The RIP tanks are each equipped with a pump-cell mechanism. This device consists of a down-pumping agitator to keep the pulp and resin in suspension, a wedge-wire screen to retain the resin beads in the contactor, and an up-pumping impellor inside the screen basket to develop sufficient head for the pulp to flow to the next contactor via an open volute. The RIP plant comprises 10 such contactors with 8 online at any one time extracting uranium from the leached slurry. The other two contactors are off-line, one of which transfers loaded resin to the resin recovery circuit while the second is on standby or receiving stripped resin from the elution circuit.

Slurry that has passed through the series of adsorption contactors flows from the final RIP tank to another linear screen to ensure that any resin that may have escaped the interstage screens is recaptured. The RIP tail slurry is then pumped, via a sampler, to the tailings thickener feed box. The overflow from this thickener is returned to the leach feed box, and the underflow is pumped to two mechanically agitated tanks, where it is neutralized with slaked lime before being pumped to the tailings dam.

On a fixed cycle or when the resin in the lead RIP contactor is sufficiently loaded, the lead RIP contactor is taken off-line, whereby the second contactor in the sequence becomes the lead contactor and the tank containing freshly stripped resin is added to the end of the adsorption train. The tank containing the loaded resin and pulp is drained in its entirety and pumped over a linear screen with water sprays to clean and recover the resin. The screen undersize is returned to the RIP feed sump, and the recovered resin is subjected to additional cleaning using spirals and a secondary screen to remove near-size grit.

The clean loaded resin is fed to the last contactor in the elution circuit, which operates in a similar countercurrent carousel fashion to the adsorption circuit. Fresh eluent (1.1 M sulfuric acid) is pumped to the lead elution contactor and then passes by gravity flow through the other contactors in sequence before the concentrated eluate is discharged from the last elution vessel. The elution vessels are air-agitated conical bottom tanks with wedge-wire "spiders" to retain the resin. The design provided for resin scrubbing and regeneration; however, this part of the process has not been required.

In 2012, another pioneering innovation was introduced at Kayelekera (patent pending). To recover sulfuric acid from the eluate and reduce the magnesia demand for neutralization of the concentrated eluate, the concentrated eluate is first fed to a membrane plant where ultrafiltration followed by nanofiltration recovers a significant fraction of the acid in the eluate as permeate (which is recycled to the leach, offsetting the new acid demand for leaching) and simultaneously increases the uranium concentration in the retentate. The retentate is then advanced to the neutralization section, instead of untreated concentrate eluate as was done in the original circuit.

The nanofiltration retentate is pumped through a series of neutralization tanks, and magnesia slurry is added to raise the pH to 2.5–3.0. Some of the iron in the eluate precipitates, and this precipitate, along with impurities introduced by the magnesia, is captured on a filter press. The filter cake is recycled to the leach or RIP feed to recover any co-precipitated uranium, and the filtrate is pumped to the batch uranium peroxide precipitation tanks (one being filled, one being used for precipitation, and one being drained).

The uranium precipitation tanks are mechanically agitated, and the solution is recirculated after the addition of hydrogen peroxide and sodium hydroxide (to control the pH) as a batch process before being transferred in batches to the uranium precipitation thickener. The underflow from this thickener is washed in two countercurrent wash thickeners and the overflow returned to the pre-leach thickener. The underflow is dewatered in a centrifuge before reporting to the drying and packaging plant.

Solids from the centrifuge are blended with a recycle stream of dried yellowcake (recycled from the dryer) and fed to the dryer with a screw feeder. The drying is done indirectly with hot oil that flows through the screws. Dried product gravitates into a product hopper and is then fed into 210-L drums using a rotary feeder. A schematic flow diagram and the reagent consumptions are presented in Figure 19 and Table 8, respectively.

In April 2014, the mine and plant was placed on care and maintenance because the uranium price was insufficient to justify continuing operation. However, from a technical perspective, the plant was performing close to design expectations with uranium extraction in the leach of around 88% and more than 98% uranium recovery by RIP. The ore is variable in its acid and oxidant demand, and, in the future, this aspect could be improved by better management of stockpile blending and the application of radiometric sorting.

ALKALINE LEACH OPERATIONS

Langer Heinrich—Attrition/Tank Leach/CCD/IX/H_2O_2 Precipitation

The Langer Heinrich mine (LHM) is located in Namibia about 80 km inland from a deep-water port at Walvis Bay. When it was commissioned in early 2007, it was the first conventional mining and processing operation to be brought into production for more than a decade and the first one to use alkaline leaching since 1981. Remarkably, the head grade is less than one-third that of previous alkaline leach plants.

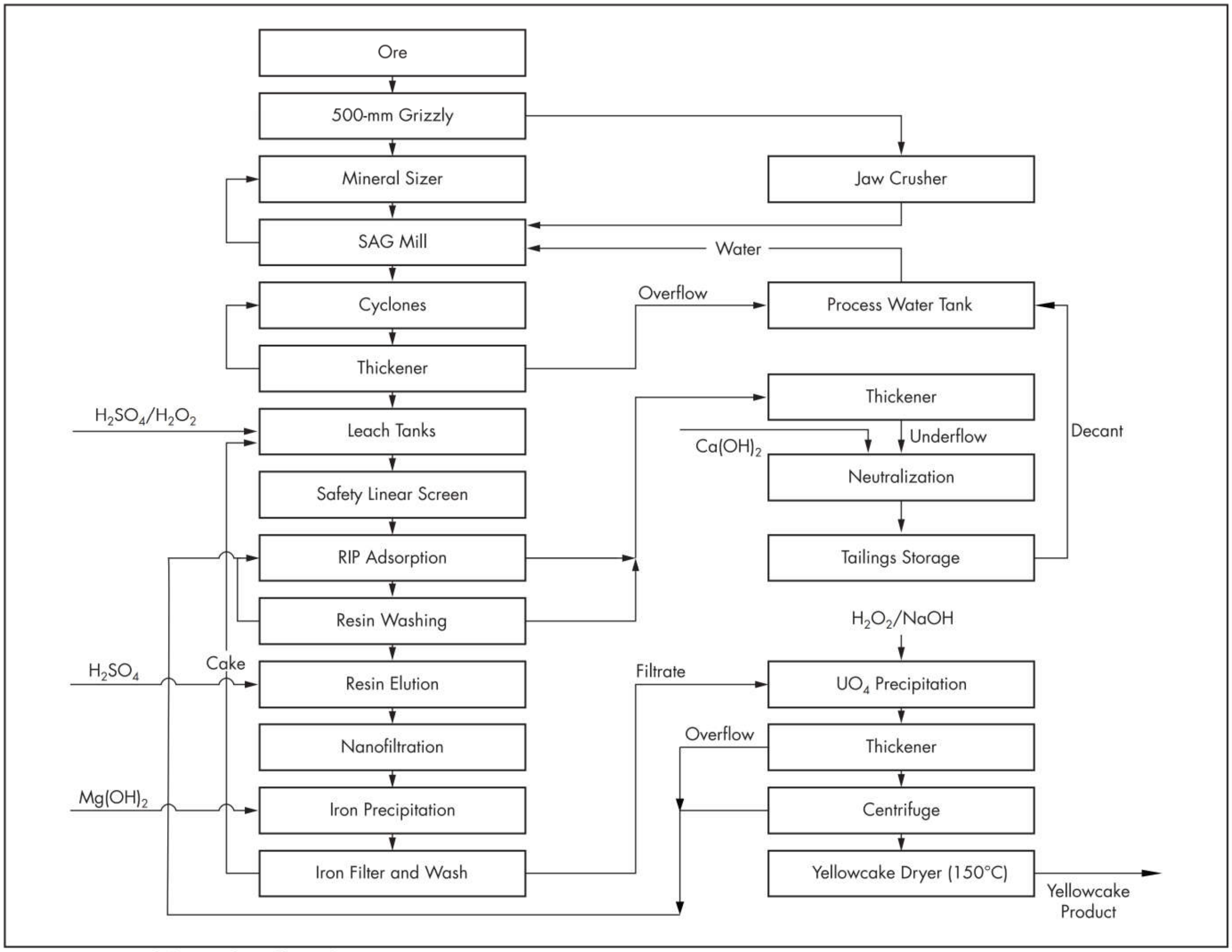


Figure 19 Kayelekera plant flow sheet

Gencor (now BHP) discovered the deposit in the 1970s and conducted an extensive project evaluation over a period of eight years. In 1998, the project was sold to Acclaim Uranium and then in 2002, to Paladin Energy. Paladin completed a bankable feasibility study in April 2005 and, based on a mill throughput of 1.5 Mt/yr at a feed grade of 740 g/t U, concluded that a production rate of 1 million kg U/yr could be sustained for more than 11 years. Site works began in September 2005, and the mine was officially opened in March 2007. Nameplate production was achieved in December 2007.

Uranium mineralization at LHM is associated with the calcretization of fluvial sediments in an extensive paleodrainage system that extends more than 15 km. It is between 50 and 1,100 m wide and occurs at the depth of between 1 and 30 m below the surface. The uranium occurs as carnotite (an oxidized secondary uranium/vanadium mineral) as thin films lining cavities and fracture planes and as grain coatings.

In May 2008, work on the Stage 2 expansion of production to 2.6 Mt/yr of ore and 1.4 million kg U/yr was started. This extension was commissioned in early 2009, and, before the end of the year, the design and engineering of Stage 3 expansion to 2 million kg U/yr was initiated, and nameplate production was achieved in early 2012.

Table 8 Kayelekera key reagent consumptions

Reagent	Units	Value
Fresh water	kg/t ore	540
Grinding media	kg/t ore	1.5
Flocculant	kg/t ore	0.2
Sulfuric acid	kg/t ore	55
Hydrogen peroxide—leach	kg/t ore	2.5
Lime	kg/t ore	10
Resin	L/t ore	0.25
Magnesia	kg/kg U_3O_8	3.0
Hydrogen peroxide—precipitation	kg/kg U_3O_8	0.2
Sodium hydroxide	kg/kg U_3O_8	0.4
Sulfur	kg/kg acid	0.33
Electricity	kW·h/kg U_3O_8	36
Diesel fuel for power	L/kW·h	0.26

The LHM process is possibly unique in that after crushing to a nominal top size of 100 mm, the ore is fed to an autogenous rotary scrubber where the agglomerates are broken down into individual grains and the uranium minerals are scrubbed from the grain surfaces. Size classification is used to separate the barren sands from the rich fines. This size classification is done with a combination of cyclones and screens, permitting around 40% of the mass to be rejected (with less than 6% of the uranium) and the leach feed to be upgraded to 1,000 ppm U.

Leaching is done using a mixture of sodium carbonate and sodium bicarbonate in a series of mechanically agitated tanks at elevated temperature. Two sets of spiral heat exchangers are used. The first set recovers heat from the leached pulp and the second set uses hot water (recycled through electric and diesel boilers) to raise the temperature in the leach feed to 75°C. In addition, inter-tank hot-water-immersion heating bundles are installed to compensate for the heat losses during the leach.

The leached solids are then washed in a six-stage CCD circuit and the tailings are pumped to a tailings storage facility. The decant liquor is recycled to the plant to conserve water and reagents. Part of the overflow from CCD2 is recirculated to the leach conditioning tank to control the leach density (30% solids), and the overflow from CCD1 is clarified in preparation for uranium recovery by ion exchange. The pH of the PLS is increased to about 11.2 using lime and sodium hydroxide to improve the selectivity of the resin for uranium over vanadium.

In the very few instances where ion exchange has been used for uranium recovery from an alkaline PLS, sodium chloride has been used as the eluent. However, in the LHM context, situated as it is in a national park in the Namib Desert, with no ability to bleed process water, the chlorides would build up in the process water to levels that would severely depress the uranium loading on the resin. Sodium bicarbonate was found to be a suitable alternative eluent and although it is not as efficient an eluent as sodium chloride, it avoids adding chloride to the system and indirectly satisfies the leach reagent demand (with no other sodium carbonate or sodium bicarbonate being required).

The concentrated eluent is fed to the continuous SDU (sodium diuranate) precipitation plant where sufficient sodium hydroxide is added to react with the remaining sodium bicarbonate and uranium as well as leaving 10 g/L residual NaOH (sodium hydroxide). The final liquor is thickened and the underflow is repulped with fresh water to wash out sodium carbonate and caustic soda.

The thickened SDU slurry is then pumped to one of three batch tanks, where, after decanting excess liquor, sulfuric acid is added to dissolve the SDU, decompose any carbonates, and adjust the pH to 3.5. The next step uses hydrogen peroxide to precipitate a pure uranyl peroxide, while the pH is controlled using sodium hydroxide. The UO_4 precipitate is thickened, repulped with fresh water, and dewatered with a centrifuge. The centrifuge product is then sent to a dryer where the remaining moisture is removed at low temperatures (115°C). The dried product discharges to a bin and is packaged in tightly sealed 210-L drums.

Plant expansions, as noted previously, were the inclusion of recycle crushing of coarse scrubbed rocks and a second larger pre-leach thickener. The next expansion added a second primary crushing circuit and a tertiary crushing plant, plus a third larger scrubber and classification equipment. The expansions also added a second leaching train, with the associated heat exchangers. The original six 25-m-diameter CCD thickeners were rearranged into two trains of three and supplemented by three additional 35-m-diameter thickeners, and then the original six thickeners were reconfigured as three trains of two thickeners and, with two additional 35-m-diameter thickeners, created a seven-stage CCD circuit.

The original IX plant consisted of 16 downflow fixed-bed columns (3.1 m-diameter), each containing 22.4 m^3 of strong base anion resin in lead, lag, or elution mode. The number of columns was initially increased to 24. However, the resin bed clogged up with suspended solids and frequent backwashing was required, introducing inefficiencies to the system. Finally it was decided to use NIMCIX multistage fluidized-bed contactors and two parallel modules of adsorption, and elution columns were installed with a nominal capacity of 600 m^3/h PLS. A schematic flow diagram and the reagent consumptions are presented in Figure 20 and Table 9, respectively.

Trekkopje, Namibia—Alkaline Heap Leach

Trekkopje is a shallow, high-tonnage, low-grade, calcrete surface deposit. The principal mineralization extends over an area approximately 14 km long and 3 km wide. It will be exploited with an open pit mine. Owing to the very low grade of the ore (~120 ppm U) and the method used to process it, exploiting this deposit presents a technical challenge.

The Trekkopje Project uranium mineralization was first drilled in the 1970s, with some confirmatory work performed during the 1990s, and was reestablished as a uranium development project by UraMin in 2005. The project was then taken over by AREVA in 2007. The project was advanced through to the feasibility study, incorporating a MINI pilot plant of 30,000 t and then to a demonstration MIDI-phase plant with the first uranium concentrate (SDU) from the Trekkopje deposit produced in January 2011. During this demonstration-plant phase, ending in mid-2013, almost 435 t of SDU was produced, confirming the technical aspects of this large-scale alkaline leaching operation. The installations for the full-scale production phase, called MAXI, were mothballed in 2013 waiting for improved uranium market conditions.

Heap leaching was selected as the preferred process for the Trekkopje Project. Given a total leach cycle time of 120 days (plus washing and rinsing cycles), an irrigation rate of 5–10 L/h/m^2, a crush size of 38 mm, and a carbonate concentration of ~20 g/L carbonate and ~10 g/L bicarbonate, an extraction of +75% of the uranium was achieved.

The performance of a multistage countercurrent fluidized bed IX system was modeled based on the batch equilibrium loading and loading rate data. To optimize uranium recovery, the IX plant needs a total resin inventory (for a PLS of 2,300 m^3/h containing 300 mg/L U) of between 1,000 and 1,300 m^3.

The crushing plant was designed to crush ore at a rate of 7,000 t/h to a product size of 100% passing 38 mm. The crushing plant consists of two primary movable mineral sizer crushers running in parallel and two secondary crushers, also running in parallel, each with a throughput of 3,500 t/h. All ore is transported from the primary crusher by overland conveyors to the stationary secondary and tertiary crusher installation.

Both sets of secondary crushing circuits deliver crushed ore into one of two storage silos. Material is reclaimed from these silos by means of apron feeders feeding onto the agglomeration drum feed conveyors that discharge material directly into the agglomeration drums.

Crushed ore is fed to the agglomeration drum at 100,000 t/d, with water added to bring the moisture to ~8%. The agglomerated ore is then stacked with a RAHCO

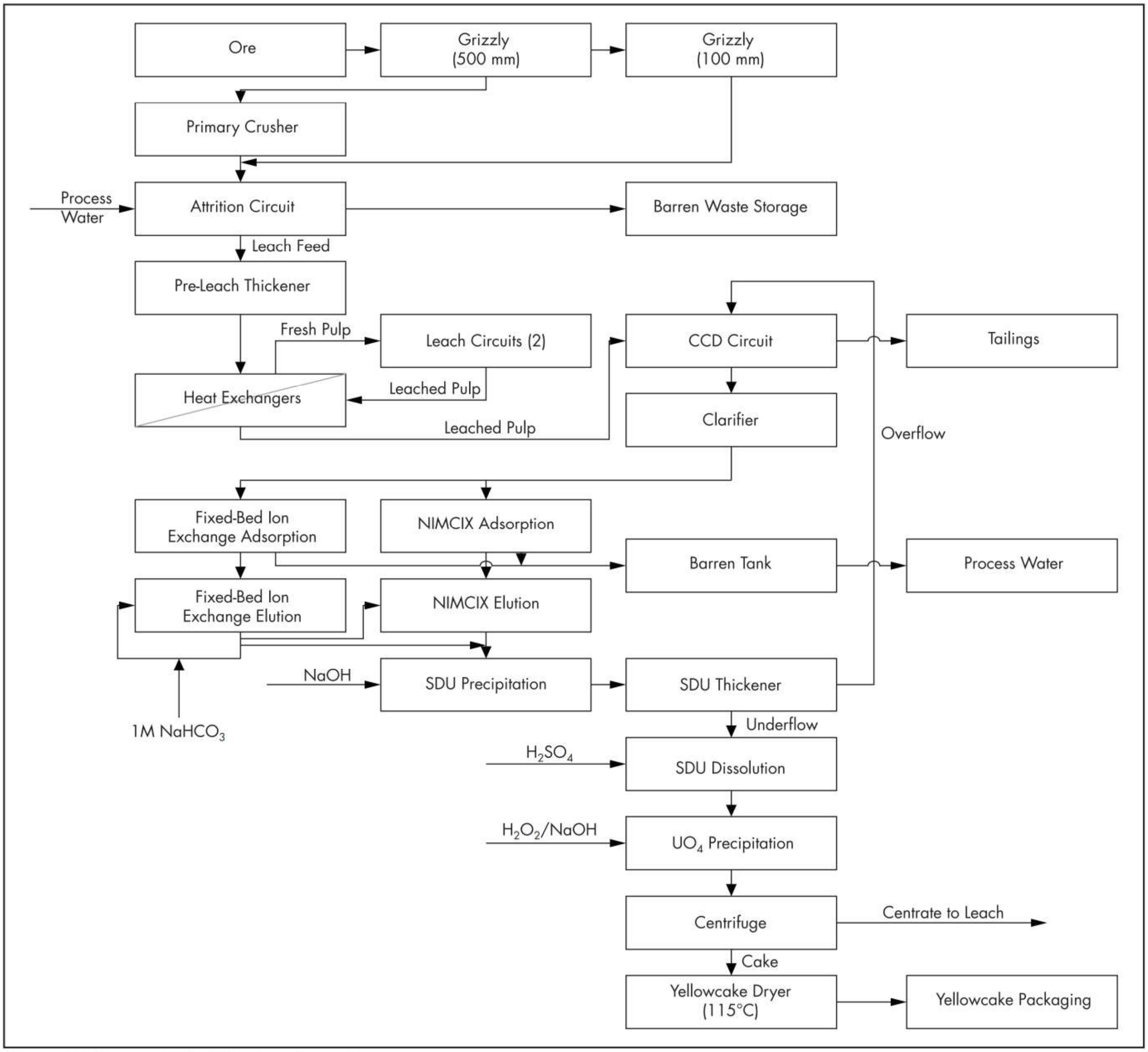


Figure 20 Langer Heinrich plant flow sheet

Table 9 Langer Heinrich key consumption data

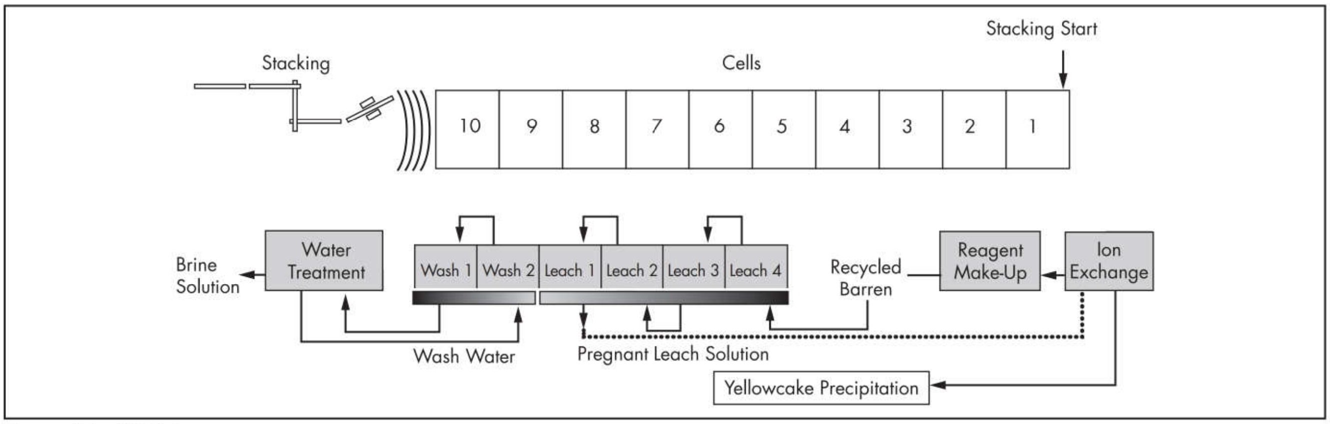
Reagent	Units	Value
Fresh water	kg/t leach feed	830
Steam	kg/t leach feed	90
Flocculant	kg/t leach feed	0.21
Sodium bicarbonate	kg/kg U_3O_8	30
Sodium hydroxide	kg/kg U_3O_8	13
Hydrogen peroxide	kg/kg U_3O_8	0.25
Hydrated lime	kg/kg U_3O_8	2
Sulfuric acid	kg/kg U_3O_8	0.80
Electricity	kW·h/kg U_3O_8	30

conveyor/stacker in 1-million-t cells (~900 m long × ~100 m wide × 9 m high).

The Trekkopje ore deposit contain chlorides that must undergo washing with fresh water to remove the chlorides. The chlorides compete with uranium for loading on the resin, and if the concentration in the PLS is in appreciable amounts (>1.6 g/L), it can lead to poor uranium loading on the resin, affecting the performance of the entire circuit downstream.

Dripper layout design is such that the pad can be irrigated at 5–8 L/m^2/h, with the cell area at 75,000 m^2, giving a total flow rate to each cell around 600 m^3/h.

The wash is comprised of two cells operating in series, with the discharge of the first cell feeding the next. The



Source: Schnell 2011

Figure 21 Alkaline heap leach basic flow sheet

discharge of the second cell reports to the high-chloride water treatment plant where reverse-osmosis technology is used to bring the chloride down to about 600 ppm.

Leaching is carried out with a sodium carbonate and sodium bicarbonate solution at 40 g/L and 8–10 g/L, respectively. The irrigation rate during the leach cycle is at 5–8 L/m^2/h with a 40-day leach cycle and a three-cycle countercurrent leach for ~120 days of total leach time.

The average uranium grade of the stacked ore is expected to be around 106 g/t, and the sulfate, 0.233%. Sodium carbonate consumption due to the ore is about 14 kg/t/%SO_4, which is equivalent to a consumption of 3.2 kg/t. The sodium bicarbonate consumption for leach due to the ore is ~1 kg/t. The PLS grade of three cells in series reaches a peak of 150 ppm U.

The PLS is fed through a series of ponds to the NIMCIX IX columns with solution characteristics as shown in Figure 21, which provides a basic flow sheet for an alkaline heap leach facility. These columns use resin to adsorb the uranium in a multistage fluidized column. The uranium is eluted from the resin using 1 M sodium bicarbonate solution. The uranium grade of the solution leaving the elution columns is <4 g/L U for effective precipitation.

SDU is produced by direct precipitation with caustic soda added to the eluate. Before the precipitate can form, the sodium bicarbonate in the eluate must first be oxidized by the caustic according to the following reaction:

$$NaOH + NaHCO_3 \rightarrow Na_2CO_3 + H_2O$$

Excess caustic in the region of 5–7 g/L of eluate is required for effective precipitation, and once the required amount of excess caustic is achieved, SDU precipitate is formed with a precipitation efficiency of 98%, with eluate at 4.6 g/L. Precipitated SDU is sent to a thickener for dewatering to increase the slurry density from 1.07 to around 1.2. The dewatered SDU is washed with water to remove excess caustic.

Sulfuric acid is then added to the SDU slurry to dissolve the SDU. Caustic and hydrogen peroxide are then introduced to increase the pH of the solution, and thereby precipitate UO_4. The precipitate is filtered, dried, and packaged into standard 170-L drums ready for shipment.

BY-PRODUCTS AND CO-PRODUCTS FROM URANIUM OPERATIONS

Uranium is commonly found in many geological and background settings, and there are also many examples of uranium associated with other metals or in unconventional geological settings. The term *unconventional* has been used in the IAEA *Red Book* (NEA and IAEA 2016: IAEA/OECD biannual Uranium Resources, Production and Demand Review) for many years to suggest geological settings that have not been generally associated with uranium production.

In this section various options are considered where uranium can be produced from unconventional resources or as a by-product with other production. The *Red Book* defines a series of unconventional resources—phosphates, nonferrous ores, carbonatite, black schists, and lignite.

Significant quantities of uranium are found in many unconventional resources (NEA and IAEA 2016). These are resources from which uranium is only recoverable as a minor by-product, such as phosphate rocks, nonferrous ores, peralkaline intrusions and carbonatite, and black shale and coal-lignite. Quantities shown in Table 10 are largely incomplete, as many potential resources are not assessed properly and reported. Apart from these deposit types, reprocessing of previous tailings, wastewater, and residues (such as coal ash) can also be a source of unconventional uranium. Historically, significant quantities of uranium have been produced from phosphates as a by-product of fertilizer production (IAEA 2013b). About 17,150 t U were recovered in the past from phosphate rocks in Florida (United States; IAEA 1989). Gold tailing projects in South Africa are another source that contributed uranium production considerably in the past, which continues to date, though at reduced levels (IAEA 1985).

By-Product Production Classification

Uranium will be produced from a variety of unconventional resources in the future, and there are some examples already close to or in production. A possible alternative unconventional classification is to use the process options available for these resources:

- **Category 1—By-production.** By-production is used for resources when a primary product also contains a

Table 10 Unconventional resources of uranium (metric tons of uranium)

Country	Phosphate Rocks	Nonferrous Ores	Carbonatite	Black Schist/Shales, Lignite
Brazil	84,500	2,000	13,000	–
Chile	400	800	–	–
Columbia	20,000–60,000	–	–	–
Egypt	35,000–100,000	–	–	–
Finland	1,000	–	2,500	35,000
Greece	500	–	–	–
India	1,700–2,500	6,600–22,900	–	4,000
Iraq	19,000–42,800	–	–	–
Jordan	60,000	–	–	–
Kazakhstan	29,000	–	–	–
Mexico	240,000	1,000	–	–
Morocco	6,526,000	–	–	–
Peru	41,600	140–1,410	–	–
South Africa	180,000	–	–	70,700
Sweden	42,300	–	–	1,012,000
Syria	60,000–80,000	–	–	–
Thailand	500–1,500	–	–	–
United States	14,000–33,000	1,800	–	–
Venezuela	42,000	–	–	500

Adapted from NEA and IAEA 2016

secondary product or more than one other product. Often, the secondary product is not economic by itself, but perhaps as an add-on it can take advantage of the primary commodity production process to be available for recovery.

An example is uranium from phosphoric acid. The uranium is already soluble in the phosphoric acid and could be extracted, thereby reducing environmental concerns of uranium in fertilizers and producing a higher-quality phosphoric acid. Considerable work is underway to make the uranium economically attractive. Another example would be uranium in copper heap leach operations with uranium leached during the copper process.

- **Category 2—Co-production.** When more than one metal has economic values, co-production is employed. This is common in base metals when, typically, copper and nickel or zinc and lead are produced from a single ore source.

  The most notable example for uranium is the Olympic Dam operation in Australia, with economic uranium production from a copper producer. This places the Olympic Dam operation in the co-production category given that the primary source of uranium is from the copper tailings. Considering the high value of the uranium, this is normally considered a co-product relationship. Talvivaara in Finland is an interesting case of multiple products: the Talvivaara operation has copper, zinc, nickel/cobalt, and uranium as products using a heap leach flow sheet in a cold climate. This operation is a good example of "comprehensive extraction."
- **Category 3—Tailings reprocessing.** This option is used for tailings or mine waste reprocessing. Generally, this category would be for reprocessing previously placed tailings.

  The best examples are the tailings reprocessing operations for gold in South Africa. The tailings are hydraulically mined with high-pressure water, then fed to a leach plant for gold recovery. Some of these tailings also contain significant uranium values, and in some cases, this uranium is also recovered (Mine Waste Solutions plant of First Uranium Corporation). Similarly, a recent project at the Somair operation in Niger is using heap leach on previous mine waste with previous uneconomic uranium values to produce 400–800 t U/yr. As metal prices change with environmental considerations, reprocessing of old tailings will continue. The better option, of course, is to extract the most value initially.
- **Category 4—Unusual treatment.** This option would be used for new, innovative processes on previously uneconomic or unconventional resources.

  For example, the Trekkopje deposit in Namibia has extremely low grades (120 ppm U) that have been considered too low, but applying economics of scale combined with heap leach technology and new application of alkaline chemistry to heap leach, this deposit has potential for significant uranium production in the future. Another unusual resource is the Bakouma deposit in Central African Republic. The difficulty of this deposit is the high apatite content that requires very high-acid consumptions and subsequent neutralization that makes even a good-grade uranium deposit uneconomic. Also, this deposit has an unfavorable geographic location. It will be interesting to see how eventually these difficulties will be overcome for these projects.

Uranium has been produced as a by- or co-product of other metals and minerals, most notably: gold, copper, vanadium, and phosphate. Historically, by- and co-product uranium has accounted for approximately 12% of world production.

By-Product Operation Examples

Gold

Gold/uranium ores of South Africa's Witwatersrand Basin consist of a quartz-pebble conglomerate of Precambrian age containing low to moderate concentrations (0.02–0.06% U) of uraninite- and brannerite-type minerals. Uranium production from these ores has been ongoing since 1952, from both primary ore and historical tailings (slimes). Uranium extraction is by sulfuric acid leach of slurry either pre- or post-gold recovery. Pre-gold recovery leaching has been employed to enhance subsequent gold recovery. Uranium is recovered from the leach solution by ion exchange, concentrated by solvent extraction, and precipitated with ammonia. Overall uranium recovery is quite low at about 30%–40%.

Copper

Uranium has been recovered as a by- or co-product of copper in several ways, including (1) direct sulfuric acid leaching of initial copper ore slurry (Twin Buttes, United States); (2) IX extraction from dump leach solutions (Bingham Canyon, United States); (3) as a gravity concentrate (Palabora in South Africa; and Rakha, Surda, and Mosaboni in India); and (4) sulfuric acid leaching of copper flotation concentrates and tailings (Olympic Dam, Australia). More specific details for some operations are given here:

- **India (1970s–2000).** Small quantities of uranium were produced as a by-product of copper in India at three uranium recovery plants: Rakha, Surda, and Mosaboni. These plants treated flotation tails (~0.015% U) from copper concentrators by means of Wilfley tables to produce uranium-bearing heavy mineral concentrates (0.13% U). Concentrates were shipped by truck to the Jaduguda uranium mill for final recovery.
- **Palabora (1971–2002).** Uranium at Palabora occurs as uranothorianite at an average concentration of 35 ppm U in close association with copper sulfide mineralization in a carbonatite. Uranium recovery was from copper flotation tailings through a complex process of magnetic separation, gravity concentration, nitric acid leaching, and solvent extraction. The average recovered uranium grade was about 6 ppm.
- **Bingham Canyon (1978–1985).** The Bingham Canyon uranium plant ran four Higgins Loop IX columns operating in parallel to process approximately 1,800 m^3/h of copper–barren dump leach solution. Uranium concentration in the feed solution was typically in the range of 4 to 6 ppm U. Uranium was eluted with sulfuric acid, concentrated by solvent extraction, and precipitated with ammonia.
- **Twin Buttes (1980–1985).** At Twin Buttes, copper oxide ore containing about 30 ppm U was crushed and wet-ground to liberate both copper and uranium. Ore pulp was acidified with sulfuric acid to dissolve copper and uranium. Uranium was extracted by ion exchange, concentrated by solvent extraction, and precipitated with ammonia.
- **Olympic Dam (1988–present).** Sulfide mineralization at Olympic Dam averages 0.9% Cu and 0.04% U with minor amounts of gold and silver. Copper concentrates are recovered by flotation, and both concentrates and tailings are then leached with sulfuric acid to dissolve uranium. Uranium is recovered by solvent extraction, stripped with ammonium sulfate, and precipitated with ammonia.

Vanadium

Uranium/vanadium deposits of the Colorado Plateau in the western United States were first mined for radium in the late 1800s and then for vanadium beginning in the 1920s. Uranium for the Manhattan Project (the first atom bombs) was recovered from these tailings during World War II. Following the war, uranium and vanadium were produced as co-products, a practice that continues sporadically to the present.

Ores with a vanadium-oxide-to-uranium-oxide ratio of about 6:1 occur in sedimentary rocks of the Colorado Plateau district. Uranium and vanadium grades average 0.19% U and 1.23% V_2O_5 (vanadium oxide), respectively. Current milling operations incorporate a sulfuric acid leach with an SX process containing separate recovery circuits for uranium and vanadium extraction. Uranium is extracted first, stripped with sodium chloride, and precipitated with ammonia. Vanadium is stripped with soda ash, batch precipitated, dried, and fused to V_2O_5.

Phosphate

Uranium occurs in low concentrations (typically 50–150 ppm U) in most marine phosphate deposits. Phosphate rock is beneficiated, concentrated by flotation, and acidified with sulfuric acid to produce phosphoric acid, a basic building block for phosphate fertilizers. Phosphoric acid typically contains about 0.5 kg U/t of acid. During the period 1977–1999, several uranium by-products from phosphoric acid facilities operated in the United States, Canada, and Belgium. All used similar SX processes and all have now been decommissioned.

Typically, a uranium recovery plant consisted of two SX circuits. The primary circuit was designed to remove uranium from the large volume of feed acid and concentrate it (by a factor of 50). Concentrated uranium was then sent to the secondary circuit where uranium was further concentrated, stripped with ammonium carbonate, and then steam stripped to remove ammonia and carbon dioxide, which caused uranium to precipitate.

Future Technologies and Challenges

Efforts are underway on several fronts to develop new technologies for by- and co-product uranium recovery and to recover uranium from polymetallic and multielement deposits. Uranium from phosphate deposits are advancing with various levels of research and some pilot-plant operations. The Black Shale deposits, predominantly found in Sweden, have one new operation underway and research is continuing for these deposits.

Rare earth deposits typically contain some uranium that may be recoverable as a by-product if economics permit. The proposed Round Top project in Texas (United States), for example, exhibits concentrations of rare earths on the order of 600 ppm and potentially recoverable quantities of niobium, hafnium, tantalum, and tin, along with 45 ppm U. Extraction would utilize a sulfuric acid heap leach.

Good ore deposits of high grade and without high impurities are rare and exceptional. As a result, future challenges for uranium production are to produce uranium as a by-product and to produce by-products from uranium ore deposits. Many other challenges also exist: to keep costs low, reduce environmental problems, provide safe working conditions, reduce worker exposure, and consider sustainable development.

BEST PRACTICES FOR OPERATIONS AND DESIGN

Recognition and adoption of best practice principles are considered fundamental cornerstones of sustainable development for the uranium industry. This covers the social, environmental, and economic aspects of an operation and includes the active search, documentation, and implementation of those practices and principles that are most effective in improving the social, environmental, and economic performance of an operation (IAEA 2010).

The guiding principle for the peaceful use of nuclear energy defines that "any use of nuclear energy should be beneficial, responsible and sustainable, with due regard to the protection of people and the environment, non-proliferation, and security" (IAEA 2008). Based on this principle, the criteria for uranium are well defined under (1) beneficial, (2) responsible, and (3) sustainable uses (IAEA 2013a, 2013b). Sustainability of uranium production depends on improved recovery techniques and continued exploration and development of new ore bodies as older ones are depleted. At the end of mine life, decommissioning and remediation of facilities are also required (IAEA 2009).

The International Council on Mining and Metals (ICMM) has developed 10 principles for sustainable development performance (ICMM 2003). The World Nuclear Association (WNA) has published *Sustaining Global Best Practices in Uranium Mining and Processing*, which sets down a corresponding set of principles applicable to the worldwide uranium production industry (WNA 2008).

The application of best practice principles for a project starts at the conceptual phase and continues throughout all of the stages of a project, including conceptual design and/or exploration, feasibility studies, construction, operation, remediation, and closure and post-closure stewardship. Best practice principles are continually developed and improved upon for any project as it moves through the stages of the cycle and as more information is collected and a better understanding gained.

Generally, the first step is baseline data collection. Baseline information is required to characterize both the physical and social environment prior to project development. Typically, baseline studies are required to understand the predevelopment conditions and integrate information into project supporting documents. Public and stakeholder consultation processes should commence and be managed in parallel with the baseline data collection program during the exploration or conceptual design stage. Recognition and response to stakeholder concerns and expectations can minimize the potential for conflict and be of mutual benefit to the communities and the operators.

The impact assessment process identifies potential adverse impacts of the project. Impact assessment is a process of identification, communication, prediction, and interpretation of information to identify potential (both adverse and beneficial) impacts through the life of a project and determine measures to manage these impacts. Impacts are predicted based on the comparison of baseline information and anticipated future conditions, both with and without the project occurring.

Undertaking a formal risk analysis is a fundamental component of the decision-making process for the sustainable operation of projects. Risk assessment can be used to determine the existing level of risk to the social, environmental, and economic aspects of a project. Risk assessment can also be used to evaluate the relative risk reduction achieved by various risk management options and is site or project specific. Sensitivity analysis is used to examine how robust an alternative is to changes in the information or assumptions used in the original analysis.

A great challenge to operational design is that standards continue to evolve over time. Any design and subsequent reclamation strategy that does not anticipate and allow for changes in legislative and regulatory requirements, as well as evolving community expectations, carries a high risk of failure.

Uranium projects need to incorporate an environmental management system (EMS) into the operation. Several EMS standards have been developed for operations to ensure they are designed and operated to meet the objectives and specific needs of the project. Two series of International Organization for Standardization (ISO) standards that are particularly relevant in this regard are the ISO 9000 and 14000 series. The ISO 9000 series focuses on quality, while the ISO 14000 series defines an EMS based on a commitment to continuous improvement. Another operational management tool is the use of key performance indicators (KPIs). KPIs are targets that may be either quantitative or qualitative, and, in contrast to ISO 14000, are used to measure performance against specific objectives or set values.

Management systems required for waste products associated with a mine or processing facility are site specific and in some rare instances region specific. Typically, regional optimization of waste management is not feasible, and each site must manage their own waste streams, which generally include water, waste rock, process residues, and radiologically and chemically contaminated equipment.

Continued stewardship of an operation post-closure is required to meet the best practice of sustainable development. This may consist of, but not be limited to, ongoing monitoring, collection and treatment of contaminated water, management and storage of water treatment sludges, and maintenance of facilities such as water diversion structures.

ENVIRONMENTAL IMPACTS

Documented environmental impacts from uranium mining and processing include elevated concentrations of trace metals, arsenic, and uranium in water; localized reduction of groundwater levels; and exposures of populations of aquatic and terrestrial biota to elevated levels of radionuclides and other hazardous substances. Such impacts have mostly been observed at mining facilities that operated at standards of practice that are not acceptable today. Designing, constructing, and operating uranium mining, processing, and reclamation activities based on modern international best practices has the potential to substantially reduce near- to moderate-term environmental effects. The exact nature of any adverse impacts from uranium mining and processing would depend on site-specific conditions and on the nature of efforts made to mitigate and control these effects.

Uranium tailings present a significant potential source of radioactive contamination for thousands of years into the future and, therefore, must be controlled and stored carefully. Over the past few decades, improvements have been made to tailings management systems to isolate tailings from the environment, and below-grade disposal practices have been developed specifically to address concerns regarding tailings dam failures. Modern tailings management sites are designed so that the tailings remain segregated from the water cycle to control mobility of metals and radioactive contaminants for at least 200 years, and possibly up to 1,000 years. However, because monitoring of tailings management sites has only been carried out for a short period, monitoring data are insufficient to assess the long-term effectiveness of tailings management facilities designed and constructed according to modern best practices (NRC 2012).

The precise impacts of any uranium mining and processing operation would depend on a range of specific factors for a particular site. Therefore, a thorough site characterization, supplemented by air quality and hydrological modeling, is essential for estimating any potential environmental impacts and for designing facilities to mitigate potential impacts. Additionally, until comprehensive site-specific risk assessments are conducted, including accident and failure analyses, the short-term risks associated with natural disasters, accidents, and spills remain poorly defined.

A well-designed and executed monitoring plan is essential for gauging the performance of best practices to limit environmental impacts, determining and demonstrating compliance with regulations, and triggering corrective actions if needed. Making the monitoring plan available to the public helps foster transparency and public participation. Regular updates to the monitoring plan, along with independent reviews, allows the incorporation of new knowledge and insights gained from analysis of monitoring data. In addition, best practice is to undertake an assessment of the appropriate mitigation and remediation options that are required to minimize predicted environmental impacts, such as acid mine drainage control, and tailings and waste management.

HEALTH AND SAFETY

Any mining operation or other industrial activity involving a mineral or raw material has the potential to increase the effective dose received by individuals from natural sources, resulting from exposure to radionuclides of natural origin contained in or released from such material. Where this increase in dose is significant, radiation protection measures may be needed to protect workers or members of the public.

For many of its aspects, the potential adverse health effects associated with uranium mining are no different than the risks identified in other types of non-radiation-related mining activities (Laurence 2011). Uranium mining, however, adds another dimension of risk because of the potential for exposure to elevated concentrations of radionuclides. Internal exposure to radioactive materials during uranium mining and processing can take place through inhalation, ingestion, or absorption through an open cut or wound. External radiation exposure from beta particles or gamma rays can also present a health risk.

All minerals and raw materials contain radionuclides of natural, terrestrial origin—these are commonly referred to as primordial radionuclides. The ^{238}U and ^{232}Th decay series and ^{40}K are the main radionuclides of interest. The activity concentrations of these radionuclides in normal rocks and soil are variable but generally low. However, certain minerals, including some that are commercially exploited, contain uranium and/or thorium series radionuclides at significantly elevated activity concentrations. Furthermore, during the extraction of minerals from the earth's crust and subsequent physical and/or chemical processing, the radionuclides may become unevenly distributed between the various materials arising from the process, and selective mobilization of radionuclides can disrupt the original decay-chain equilibrium. Consequently, radionuclide concentrations in materials arising from a process may exceed those in the original mineral or raw material, sometimes by orders of magnitude.

External exposure from ionizing radiation (beta and gamma) is usually not a major problem in uranium mining. Where this is a potential problem, such as the high-grade operations in Canada, extra precautions are put in place. The high-grade Canadian operations use remote ore handling methods, add shielding and additional worker controls. Lower-grade uranium (<1% U) ores, process solutions, and concentrates do not emit large quantities of penetrating radiation.

The most common radiation hazard encountered in the milling of uranium ore is airborne radioactive material. This airborne material may consist of uranium together with all its radioactive decay products in the case of ore dust, or uranium dust alone in the case of yellowcake. The other airborne hazard is radon gas and the associated radon daughter decay products, particularly during crushing, grinding, and leaching unit processes. Ventilation of process equipment and buildings is essential. Many precautions are taken at a uranium mine to protect the health of workers:

- Dust is controlled, to minimize inhalation of gamma- or alpha-emitting minerals. In practice, dust is the main source of radiation exposure in an open-cut uranium mine and in the mill area.
- Radiation exposure of workers in the mine, plant, and tailings areas is limited. In practice, radiation levels from the ore and tailings are usually very low.
- Radon daughter exposure is minimal in an open-cut mine because there is sufficient natural ventilation to remove the radon gas. In an underground mine, a good forced ventilation system is required. For example, at Olympic Dam in Australia, radiation doses in the mine from radon daughters are kept very low, with an average of less than about 1 mSv/yr. Canadian doses (in mines with high-grade ore) average about 3 mSv/yr.
- Strict hygiene standards are imposed on workers handling the uranium oxide concentrate. If it is ingested, it has a chemical toxicity similar to lead oxide. (Both lead and uranium are toxic and affect the kidney. The body progressively eliminates most lead or uranium via the urine.) In effect, the same precautions are taken as in a lead smelter, with use of respiratory protection in particular areas identified by air monitoring.

Health, Safety, and Environmental Management

International good practice for sound health, safety, and environmental (HSE) management of a uranium mining and processing site has been accumulated through long-term development of mining and processing of uranium and other minerals.

An integrated approach is one good and important practice in tackling HSE management of a uranium mining and processing site. This method considers radiological and non-radiological issues of HSE in a single, comprehensive system, which means that observing one aspect of HSE good practice will not compromise any other. Many HSE elements share a common engineering approach of sound control of radiological and non-radiological risk in HSE management, such as control of worker safety, surface run-off, and stability of the tailings dam. IAEA standards and guidelines are implemented around the world for the control of radiological risk related to uranium mining and processing sites and for protection against ionizing radiation. In particular, the IAEA safety and security standards provide a framework of international standards (including safety fundamentals, safety requirements, safety guides, safety reports, and technical documents) that address radiation, environmental, waste, and transport safety and security for uranium mining and processing sites and their related activities.

While IAEA safety standards and security series are more focused on radiological concern, the importance of an integrated approach in HSE management is addressed and encouraged in the planning and operation of radiological-related facilities and activities. The WNA policy document *Sustaining Global Best Practices in Uranium Mining and Processing: Principles for Managing Radiation, Health and Safety, Waste and the Environment* (WNA 2008) is another good reference that proportionally positions radiation risk among the diverse HSE risks that need to be addressed in uranium mining and processing.

In practice, some non-radiological HSE elements associated with any mining project may result in acute consequences, including injuries and fatalities, which rank high in the HSE inventory. Issues such as handling explosives, ground support for open pit and underground mines, operating of large vehicles and equipment, and so on, each bear significant safety risks that require careful design, practice, and management.

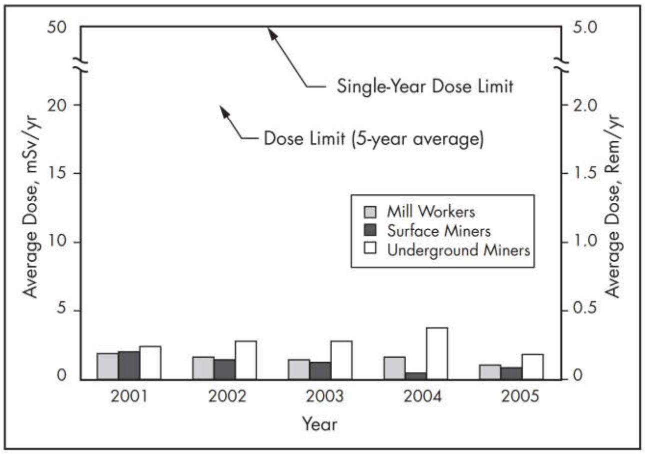
Integration of HSE aspects as part of design and operations tends to lead to greater efficiency and performance. This is an important evolution that took place over the last decade or so. The management of HSE aspects in isolation from each other and from design and operations is of a past era. Moreover, for greater efficiency, HSE issues are managed as part of an integrated management system such as ISO 9000, 14000, and 18000 series or an Occupational Safety and Health Administration equivalent.

As part of integrated management, HSE issues are addressed to an increasing level of depth in successive project development phases from prefeasibility, feasibility, detailed feasibility, and preparation for commissioning and operations.

There is a wealth of relevant international experience for a sound occupational health and safety program, based on many decades of experience in mining, which is also directly applicable to uranium mining. Among others, this experience pays particular attention to the risk of incidents (near misses), accidents, injuries, and fatalities due to "conventional" accidents (i.e., those with physical causes). For uranium mining, experience also includes specific radiation protection measures for the control and monitoring of radiation levels and of exposures to external gamma radiation, radon gas, and long-lived radioactive dust. In addition to the main items already mentioned earlier, extra radiation protection measures include the control and monitoring of possible incidental/accidental intakes of uranium by site personnel (e.g., uranium in urine program) as well as the use of personal protective equipment for both conventional and radiation health risks.

Radiation Safety Basis

The following basis of radiation protection standards is from the WNA (2014):



Source: Chambers 2011

Figure 22 Evolution of radiation dose in Canadian mining operations

In practice, radiation protection is based on the understanding that small increases over natural levels of exposure are not likely to be harmful but should be kept to a minimum [Figure 22]. To put this into practice the International Commission [on] Radiological Protection (ICRP) has established recommended standards [ICRP 1991] of protection (both for members of the public and radiation workers) based on three basic principles:

- Justification. No practice involving exposure to radiation should be adopted unless it produces a net benefit to those exposed or to society generally.
- Optimisation. Radiation doses and risks should be kept as low as reasonably achievable (ALARA), economic and social factors being taken into account.
- Limitation. The exposure of individuals should be subject to dose or risk limits above which the radiation risk would be deemed unacceptable.

These principles apply to the potential for accidental exposures as well as predictable normal exposures.

Underlying these is the application of the "linear hypothesis" based on the idea that any level of radiation dose, no matter how low, involves the possibility of risk to human health. This assumption enables "risk factors" derived from studies of high radiation dose to populations (eg from Japanese bomb survivors) to be used in determining the risk to an individual from low doses (ICRP Publication 60 [ICRP 1991]). However the weight of scientific evidence does not indicate any cancer risk or immediate effects at doses below about 50 millisievert (mSv) per year.

Based on these conservative principles, ICRP recommends that the additional dose above natural background and excluding medical exposure should be limited to prescribed levels. These are: one millisievert per year for members of the public, and 20 mSv per year averaged over 5 years for radiation workers who are required to work under closely-monitored conditions.

The frameworks of radiation safety in countries where most uranium is mined are based on the full adoption of international recommendations. This is not the case in all parts of the world. Even the 1977 Recommendation of the ICRP has not been universally adopted.

The safety record of the uranium mining industry is good. Radiation dose records compiled by mining companies under the scrutiny of regulatory authorities have shown consistently that mining company employees are not exposed to radiation doses in excess of the limits. The maximum dose received is about half of the 20 mSv/yr limit and the average is about one tenth of it. (This compares with natural doses of up to 50 mSv/yr for some places in India and Europe, without any adverse effects being evident, and mean exposures of 750 mSv/yr in some East German mines from 1946 to 1954, resulting in thousands of cases of lung cancer.)

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REFERENCES

Badea, A.L., and Schwartz, L. 2000. Uranium milling operations at McClean Lake. In *Uranium 2000: Proceedings of the International Symposium on the Process Metallurgy of Uranium*. Montreal, QC: Canadian Institute of Mining, Metallurgy and Petroleum.

Bateman Litwin. 2012. *From Concept to Turnkey: Solvent Extraction*. Corporate brochure. Amsterdam: Bateman Litwin.

Bharadwaj, B., Moldovan, B., Yesnik, L., Grant, S., and Piche, J. 2010. Cameco Corporation—The Key Lake uranium mill: Current status and vision for the future. In *Uranium 2010: Proceedings of the 3rd Annual International Conference on Uranium*. Montreal, QC: Canadian Institute of Mining, Metallurgy and Petroleum.

Cameco Corporation. 2009. *Cameco 2009 Annual Report*. Saskatoon, SK: Cameco.

Cameco Corporation. 2012. Revitalization Plan, Corporate Quarterly report 2012. Saskatoon, SK: Cameco.

Cameco Corporation. 2015. *Cameco 2015 Annual Report*. Saskatoon, SK: Cameco.

Carino, A. 2001. The McClean Lake operation: Jeb Mill process description and commissioning. In *33rd Annual Operator's Conference of the Canadian Mineral Processors*. Montreal, QC: Canadian Institute of Mining, Metallurgy and Petroleum.

Chambers, D. 2011. Overview of conventional uranium mining and milling technologies. Presented to Institute for Advanced Learning and Research, Virginia Tech, Blacksburg, VA.

CNSC (Canadian Nuclear Safety Commission). 2012. *Record of Proceedings, Including Reasons for Decision in the Matter of AREVA Resources Canada Inc*. Ottawa, ON: CNSC.

CNSC (Canadian Nuclear Safety Commission). 2015. Radioactive waste: Uranium mines and mills waste. Ottawa, ON: CNSC.

Courtaud, B., Auger, F., and Thiry, J. 2011. Method for preparing uranium concentrates by fluidized bed precipitation, and preparation of UO_3 and U_3O_8 by Drying/Calcining Said Concentrates. U.S. Patent Application 2011/0212005 A1.

Cuney, M. 2009. The extreme diversity of uranium deposits. *Miner. Dep.* 44:3–9.

Cuney, M. 2011. Uranium and thorium: The extreme diversity of the resources of the world's energy minerals. In *Non-Renewable Resource Issues: Geoscientific and Societal Challenges*. Edited by R. Sinding-Larsen and F.-W. Wellmer. Dordrecht: Springer. pp. 91–129.

Cyclone Uranium Corporation. 2012. Idealized cross-section of a sandstone-hosted roll front uranium deposit.

Dahlkamp, F.J. 2009. *Uranium Deposits of the World–Asia*. Berlin: Springer-Verlag.

Denison Mines Corporation. 2015. Exploration and Development: McClean Lake and the McClean Lake Mill. Toronto, ON: Denison Mines Corporation.

Durupt, N., Blanvillain, J.J., Jourde, J., and Sanoussi M. 2013. Heap leaching treatment of clayey ore at Somair. Presented at Heap Leach Solutions 2013 Conference, Vancouver, BC, Canada.

Edwards, C.R. 1987. Hydrogen peroxide precipitation of uranium in the Rabbit Lake mill. In *Crystallization and Precipitation: Proceedings of the International Symposium*. New York: Pergamon Press. pp. 149–157.

Edwards, C.R. 1992. Uranium extraction process alternatives. *CIM Bull.* 85:112–136.

Edwards, C.R. 1997. Ore handling for the McArthur River and Cigar Lake projects. Presented at Canadian Mineral Processors Division Conference of Canadian Institute of Mining, Metallurgy and Petroleum, Ottawa, ON.

Edwards, C.R. 2002a. The Cigar Lake project—Mining, ore handling and milling. Presented at Canadian Mineral Processors Division Conference of Canadian Institute of Mining, Metallurgy and Petroleum, Ottawa, ON.

Edwards, C.R. 2002b. Methods of evaluating ore processing and effluent treatment for Cigar Lake ore at the Rabbit Lake Mill. In *Technologies for the Treatment of Effluents from Uranium Mines, Mills and Tailings*. IAEA-TECDOC-1296. Vienna: International Atomic Energy Agency. pp. 5–14.

Edwards, C.R., and Dyck, K. 2004. Replacement of classification screens with cyclones in the McArthur River uranium mine grinding circuit. Presented at Canadian Mineral Processors Division Conference of Canadian Institute of Mining, Metallurgy and Petroleum, Ottawa, ON.

Edwards, C.R., and Schnell, H. 2000. Process design for handling, transporting and milling Cigar Lake ore. In *Uranium 2000: Proceedings of the International Symposium on the Process Metallurgy of Uranium.* Montreal, QC: Canadian Institute of Mining, Metallurgy and Petroleum.

El-Ansary, Z., and Lund, D. 2008. Tailoring process selection to uranium mineralogy and ore type. Presented at AusIMM International Uranium Conference, Adelaide, South Australia, June 18–19.

ERA (Energy Resources of Australia). 2008. *Energy Resources of Australia–Rio Tinto, Sustainable Development Report*. Darwin, Northern Territory: ERA.

Gomiero, L.A., Lima, H.A., and Morais, C.A. 2010. Evaluation of a new milling process for the Caetite. In *Uranium 2010: Proceedings of the 3rd Annual International Conference on Uranium.* Montreal, QC: Canadian Institute of Mining, Metallurgy and Petroleum.

Griebel, E.G. 2000. Rössing uranium update. In *Uranium 2000: Proceedings of the International Symposium on the Process Metallurgy of Uranium.* Montreal, QC: Canadian Institute of Mining, Metallurgy and Petroleum.

IAEA (International Atomic Energy Agency). 1985. *Advances in Uranium Ore Processing and Recovery from Non-Conventional Resources.* Panel Proceedings Series. STI/PUB/689. Vienna: IAEA.

IAEA (International Atomic Energy Agency). 1989. *The Recovery of Uranium from Phosphoric Acid.* IAEA-TECDOC-533. Vienna: IAEA.

IAEA (International Atomic Energy Agency). 2008. *Nuclear Energy Basic Principles.* IAEA Nuclear Energy Series No. NE-BP. Vienna: IAEA.

IAEA (International Atomic Energy Agency). 2009. *Establishment of Uranium Mining and Processing Operations in the Context of Sustainable Development.* IAEA NES NF-T-1.1. Vienna: IAEA.

IAEA (International Atomic Energy Agency). 2010. *Best Practice in Environmental Management of Uranium Mining.* IAEA NES NF-T-1.2. Vienna: IAEA.

IAEA (International Atomic Energy Agency). 2013a. *Nuclear Fuel Cycle Objectives.* IAEA Nuclear Energy Series No. NF-O. Vienna: IAEA.

IAEA (International Atomic Energy Agency). 2013b. *Radiation Protection and Management of NORM Residues in the Phosphate Industry.* Safety Reports Series No. 78. Vienna: IAEA.

ICMM (International Council on Mining and Metals). 2003. *Sustainable Development Framework: 10 Principles.* London: ICMM.

ICRP (International Commission on Radiological Protection). 1977. *Annals of the ICRP: Recommendations of the International Commission on Radiological Protection*. Vol. 1, No. 3. ICRP Publication 26 (superseded by Publication 103). New York: Pergamon Press.

ICRP (International Commission on Radiological Protection). 1991. *Annals of the ICRP: 1990 Recommendations of the International Commission on Radiological Protection*. ICRP Publication 60 (superseded by Publication 103). New York: Pergamon Press.

Kemix Pty Ltd. 2015. *AAC Pumpcell.* Corporate brochure. Midrand, Gauteng Province, South Africa: Kemix.

Krause, E., and Ettel, V.A. 1987. Solubilities and stabilities of ferric arsenates. In *Proceedings of the International Symposium on Crystallization and Precipitation.* Toronto, ON: Pergamon Press. pp. 195–210.

Lam, E.K., Neven, M.G., Steane, R.A., and Ko, K.F. 1993. Removal of molybdenum from uranium-bearing solutions. U.S. Patent No. 5,229,086.

Langmuir, D. 1978. Uranium solution-mineral equilibria at low temperatures with applications to sedimentary ore deposits. *Geochim. Cosmochim. Acta* 42:547–569.

Laurence, D. 2011. Mine safety. In *SME Mining Engineering Handbook*, 3rd ed. Edited by P. Darling. Englewood, CO: SME.

Lien, L. 2010. The use of engineered membrane separation technology "EMS" for radionuclide separations. In *Uranium 2010*: *Proceedings of the 3rd International Conference on Uranium, 40th Hydrometallurgical Meeting*, Saskatoon, SK.

Lunt, D., Boshoff, P., Boylett, M., and El-Ansary, Z. 2007. Uranium extraction: The key process drivers. *J. S. Afr. Inst. Min. Metall.* 107:419–426.

McKinnon, C. 2000. Water treatment at McClean Lake. In *Uranium 2000: Proceedings of the International Symposium on the Process Metallurgy of Uranium.* Montreal, QC: Canadian Institute of Mining, Metallurgy and Petroleum.

Merritt, R.C. 1971. *The Extractive Metallurgy of Uranium.* Golden, CO: Colorado School of Mines Research Institute.

Milde, W.W. 1989. Rabbit Lake project: Milling and metallurgy. *CIM Bull.* 82(932):69–75.

NEA and IAEA (Organisation for Economic Co-Operation and Development/Nuclear Energy Agency and International Atomic Energy Agency). 2016. *Uranium 2016: Resources, Production and Demand ("Red Book").* NEA No. 7301. Paris: OECD/NEA.

Neven, M., and Gormley, L. 1982. Design of the Key Lake leaching process. Presented at Uranium '82: 12th Annual Canadian Hydrometallurgical Conference, Toronto, ON.

Neven, M., Steane, R., and Becker, J. 1985. Arsenic management and control at Key Lake. In *Impurity Control and Disposal, Proceedings of the 15th Annual Meeting of the Hydrometallurgy Section of the Metallurgical Society.* Montreal, QC: Canadian Institute of Mining, Metallurgy and Petroleum.

Neven, M., Steane, R.A., and Lam, E. 1986. Design and operation of the Key Lake uranium mill. In *Proceedings of the Canadian Mineral Processors Division Conference of CIM.* Montreal, QC: Canadian Institute of Mining, Metallurgy and Petroleum. pp. 352–376.

NRC (National Research Council). 2012. *Uranium Mining in Virginia: Scientific, Technical, Environmental, Human Health and Safety, and Regulatory Aspects of Uranium Mining and Processing in Virginia.* Washington, DC: National Academies Press.

Paatero, E., et al. 2010. Recovery of molybdenum and uranium as by-products with solvent extraction. Metals and Materials Recycling Workshop, Santiago, Chile.

Pacquet, E. 2017. Vice President AREVA Resources Canada—Operations and Projects Update. Presentation to Saskatchewan Mining Supply Chain Forum, Saskatchewan Mining Association, April 5.

Perdula. 2013. Mixer-settler schema. https://commons.wikimedia.org/wiki/File%3AMixer-settler_schema.jpg. CC BY-SA 3.0. https://creativecommons.org/licenses/by-sa/3.0, via Wikimedia Commons.

Pool, T. 2013. Uranium market update. Presented at IAEA Technical Meeting on Uranium Production Cycle Pre-Feasibility and Feasibility Assessment, Vienna, October 7–10.

Remple, G., and Schnell, H. 2000. Processing of Cigar Lake Ore in the McLean Lake Mill. In *Uranium 2000: Proceedings of the International Symposium on the Process Metallurgy of Uranium.* Montreal, QC: Canadian Institute of Mining, Metallurgy and Petroleum.

Rodgers, C. 1996. Molybdenum extraction from uranium rich strip solution. In *Proceedings of the Canadian Mineral Processors Division Conference of CIM.* Montreal, QC: Canadian Institute of Mining, Metallurgy and Petroleum. pp. 3–22.

Rodgers, C. 2000. Converting the Key Lake Mill to Process McArthur River Ore. In *Uranium 2000: Proceedings of the International Symposium on the Process Metallurgy of Uranium.* Montreal, QC: Canadian Institute of Mining, Metallurgy and Petroleum.

Rodgers, C., and Sarion, A. 1998. Modifications of the CCD Circuit at Cameco's Key Lake Uranium Circuit. In *Proceedings of the Canadian Mineral Processors Division Conference of CIM.* Montreal, QC: Canadian Institute of Mining, Metallurgy and Petroleum. pp. 309–321.

Rössing Uranium Ltd. 2007. *Socio-Economic Component of the Social and Environmental Impact Assessment Report for the Rössing Uranium Mine Expansion Project: Socio-Economic Impact Assessment.* Public Information Document.

Schnell, H. 2009. Uranium from unconventional sources. Presented at International Atomic Energy Agency Technical Meeting on Uranium from Unconventional Resources, Vienna, Austria, November 4–6.

Schnell, H. 2010. Trekkopje. Presented at International Atomic Energy Agency Technical Meeting on Low Grade Uranium Ore, Vienna, Austria, March 29–31.

Schnell, H. 2011. Uranium processing, extremes of ore grade. Presented at SME Annual Conference and Exposition.

Stewart, K. 2010. Unique radiation protection design for processing high-grade uranium ores at the McClean Lake mill. *Uranium 2010: Proceedings of the 3rd Annual International Conference on Uranium.* Montreal, QC: Canadian Institute of Mining, Metallurgy and Petroleum.

St-Pierre, S., and Huffman, D. 2000. Unique radiation protection design for processing high grade uranium ores at the McClean Lake mill. In *Uranium 2000: Proceedings of the International Symposium on the Process Metallurgy of Uranium.* Montreal, QC: Canadian Institute of Mining, Metallurgy and Petroleum.

Thiry, J., and Bustos, S. 2014. Heap leaching technology—Moving the frontier for treatment: Applications in Niger and Namibia. Presented at IAEA-URAM Conference, Vienna, Austria, June 28.

Thiry, J., and Roche, M. 2000. Recent developments at COMINAK. In *Uranium 2000: Proceedings of the International Symposium on the Process Metallurgy of Uranium.* Montreal, QC: Canadian Institute of Mining, Metallurgy and Petroleum.

Vancas, M.F. 2003. Pulsed column and mixer-settler applications in solvent extraction. *JOM* 55(7):43–45.

WNA (World Nuclear Association). 2008. *Sustaining Global Best Practices in Uranium Mining and Processing: Principles for Managing Radiation, Health and Safety, Waste and the Environment.* WNA Policy Document. London: WNA.

WNA (World Nuclear Association). 2014. Occupational safety in uranium mining. London: WNA. www.world-nuclear.org/information-library/safety-and-security/radiation-and-health/occupational-safety-in-uranium-mining.aspx. Accessed March 2018.

WNA (World Nuclear Association). 2015. World uranium mining production. London: WNA. www.world-nuclear.org/info/Nuclear-Fuel-Cycle/Mining-of-Uranium/World-Uranium-Mining-Production. Accessed March 2017.

WNA (World Nuclear Association). 2016. Supply of uranium. August. London: WNA. www.world-nuclear.org/information-library/nuclear-fuel-cycle/uranium-resources/supply-of-uranium.aspx. Accessed February 2018.

WNA (World Nuclear Association). 2017. World uranium mining production. London: WNA. www.world-nuclear.org/information-library/nuclear-fuel-cycle/mining-of-uranium/world-uranium-mining-production.aspx. Accessed March 2018.