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Uranium Resources as Co- and By-products of Polymetallic, Base, Rare Earth and Precious Metal Ore Deposits

IAEA
International Atomic Energy Agency

URANIUM RESOURCES AS CO- AND BY-PRODUCTS OF POLYMETALLIC, BASE, RARE EARTH AND PRECIOUS METAL ORE DEPOSITS

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FOREWORD

Uranium produced in conjunction with other metals has accounted for over 11% of historical world uranium production. Measured in tonnes of uranium (t U), gold, copper and uranium–vanadium deposits have provided the bulk of this production with 158 934 t U, 68 674 t U and 64 227 t U, respectively. Production of uranium as a co- or by-product of other metals has made an important contribution to overall uranium production in the past and is likely to continue to do so into the future.

Quartz pebble conglomerates of the Witwatersrand Basin in South Africa have been the primary source of uranium as a by-product of gold. Acid leaching of ore and slimes for uranium recovery prior to treatment for gold recovery was shown to enhance subsequent gold recovery. In addition, redeposition of slimes into modern, engineered and controlled tailings management facilities have the potential to reduce long term environmental contamination. In the suburbs of Johannesburg, this has made available new land for expanding urban construction.

Uranium and copper are closely associated in many deposits throughout the world, but particularly in Australia, South Africa, South America, Zambia, the Democratic Republic of Congo and the United States of America. A significant portion of historical US uranium production derived from uranium–vanadium deposits in the Northern Colorado Plateau. By-products of uranium production include molybdenum, gold, yttrium, scandium and rare earths. Recovery of these minor by-products of uranium production may enhance the economics of uranium recovery.

This publication will raise awareness of the presence of uranium in ore deposits that are not commonly thought of as containing uranium, as well as highlighting potential additional sources of uranium for uranium miners. It also provides examples of uranium deposits that host other metals of interest in order to maximize the commercial advantages. The relevance of this to the concept of comprehensive extraction, where miners are encouraged to ‘disturb’ the ground only once, and the additional benefits of reducing metals in mine wastes are also highlighted. Together, the commercial and environmental aspects combine to maximize the socioeconomic benefits of a broad range of metallic mining projects. The IAEA officer responsible for this publication was M. Fairclough of the Division of Nuclear Fuel Cycle and Waste Technology.

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1. INTRODUCTION

1.1. BACKGROUND

Production of uranium as by- or co-product of other metals has made an important contribution to overall uranium production in the past and is likely to continue to do so into the future. This report provides a summary of historical by- and co-product production of uranium in association with other metals. It is primarily intended to illustrate what is possible in this regard and to outline the means by which it has been done. Of particular importance is the potential for very low concentrations of uranium to be economically recovered and to enhance the economic recovery of associated metals. In addition, metal by-products of uranium production may enhance the economics of uranium recovery.

It should also be noted that by- and co-product uranium recovery provides for more complete utilization of a valuable resource, as well as reducing the potential for current and future environmental liability. This latter concern, that of environmental protection, was raised at a technical meeting of the IAEA during discussions on unconventional uranium resources by a European regulator who referred to uranium recovered as a by-product as ‘green’ uranium. This concept is part of a comprehensive extraction approach and may provide additional impetus to further research and development on low grade resources of uranium.

1.2. OBJECTIVE

The purpose of this publication is to raise awareness of the presence of uranium in ore deposits that are not commonly thought of as containing uranium, as well as highlighting potential additional sources of uranium for uranium miners. It also provides examples of uranium deposits that host other metals of interest in order to maximize the commercial advantages. The relevance of this to the concept of comprehensive extraction, where miners are encouraged to “disturb the ground only once”, and the additional benefits of reducing metals in mine wastes are also highlighted. Together, the commercial and environmental aspects combine to maximise the socio-economic benefits of a broad range of metallic mining projects.

1.3. SCOPE

The scope of the publication covers metalliferous deposits only, specifically excluding uranium from phosphates, coal, seawater and other unconventional uranium resources. For this publication, the IAEA 2013 geological classification of uranium deposits with 15 types of uranium deposits/resources [1–3] has been adopted. At the end of 2017, 2939 uranium deposits/resources were listed in the IAEA **U**ranium **DEPO**sits database (UDEPO) [4, 5]. Numerical data presented in various tables of this publication are based on the 2939 deposits/resources of the database.

1.4. STRUCTURE

The publication provides a brief outline of uranium deposit classification with the emphasis on deposit types that have known or potential co- or byproduct resources or production. This is followed by the various mining and processing approaches to these, supported by case studies of each deposit type. An Appendix showing the detailed deposit classification of uranium deposits and their global distribution, as well as for comparison the equivalent distribution of other deposit types not traditionally considered as uranium deposits (but nevertheless partly with uranium potential), is included.

2. GEOLOGICAL BACKGROUND

2.1. IAEA GEOLOGICAL CLASSIFICATION OF URANIUM DEPOSITS

A ***uranium deposit*** was defined in the 2016 Red Book [6] as “a mass of naturally occurring mineral assemblage from which uranium has been or could be exploited at present or in the future”. For the IAEA UDEPO database, which is primarily a geological database, the scope of the definition has been increased to include any identified geological concentration of uranium resource irrespective of tonnage or grade [5]. UDEPO also contains large to very large low grade resources termed ‘unconventional resources’. As of 31 December 2017, the database listed 2939 deposits/resources.

In the IAEA 2013 classification of 15 types of deposit, uranium deposits are listed in a geological order from primary magmatic deposits to sedimentary and surficial deposits (Fig. 1) [2, 3, 7]. The 15 types are as follows:

- Type 1: Intrusive
- Type 2: Granite-related
- Type 3: Polymetallic iron oxide breccia complex
- Type 4: Volcanic-related
- Type 5: Metasomatite
- Type 6: Metamorphite
- Type 7: Proterozoic unconformity
- Type 8: Collapse breccia pipe
- Type 9: Sandstone
- Type 10: Palaeo quartz-pebble conglomerate
- Type 11: Surficial
- Type 12: Lignite–coal
- Type 13: Carbonate
- Type 14: Phosphate
- Type 15: Black shale

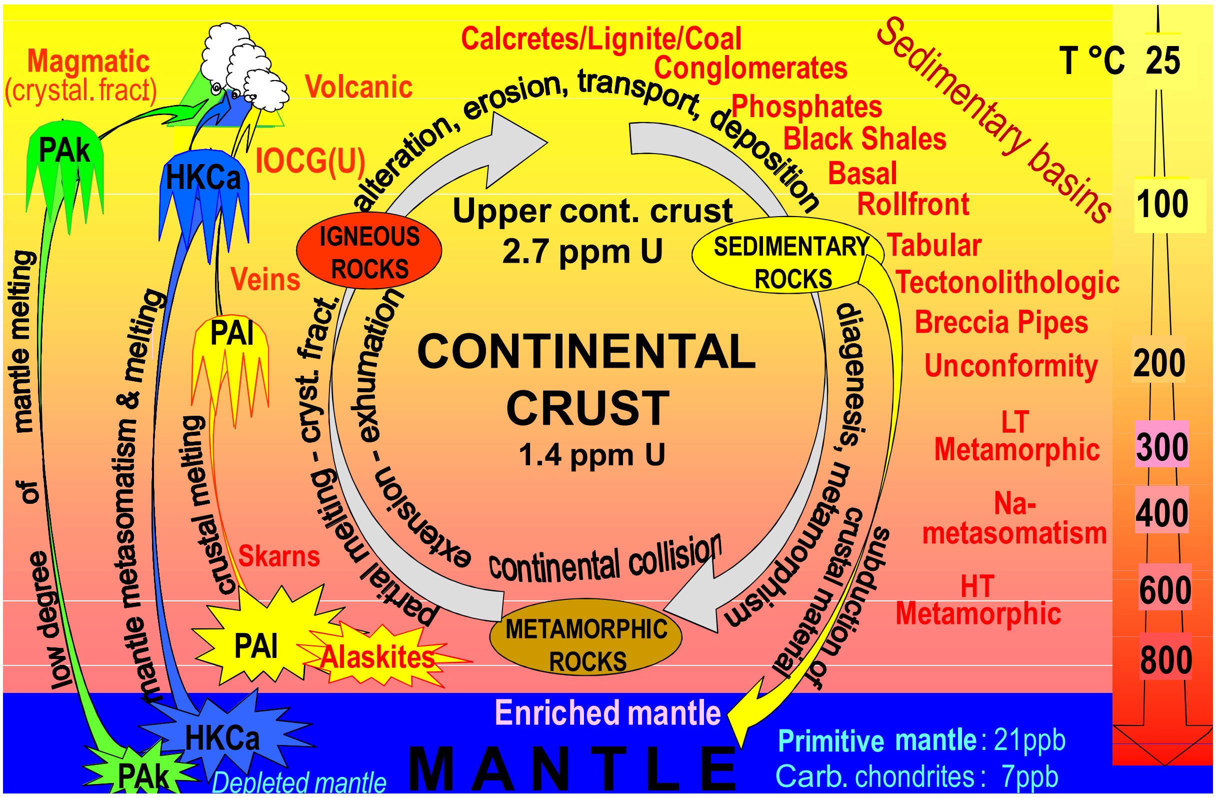

FIG. 1. Relative position of uranium deposit types with respect to the main fractionation processes within the geological cycle. The different types of U-rich magma are indicated as follows: Pak: peralkaline, HKCa: high-K calc-alkaline, Pal: peraluminous. Figure courtesy of M. Cuney and modified from [7].

Within the 15 types of uranium deposit, 37 subtypes and 14 classes have been defined [1–3]. For each type, subtype and class, typical deposits examples are listed below.

Type 1. Intrusive

1.1. Anatectic: pegmatite–alaskite (Rössing, Namibia; Bancroft district, Canada);
1.2. Plutonic with 3 classes: *Quartz monzonite* (Bingham Canyon, USA; Chuquicamata, Chile), *Peralkaline complex* (Kvanefjeld, Greenland; Poços de Caldas, Brazil) and *Carbonatite* (Phalabora, South Africa; Catalão, Brazil).

Type 2. Granite-related

2.1. Endogranitic (La Crouzille district, France; Xiazhuang district, China);
2.2. Perigranitic (Příbram region, Czech Republic; Niederschlema–Alberoda district, Germany).

Type 3. Polymetallic iron oxide breccia complex (Olympic Dam, Australia).

Type 4. Volcanic-related

4.1. Stratabound (Olovskoye, Russian Federation; Maureen, Australia);
4.2. Structurally-controlled (Streltsov–Antei, Russian Federation; Kurišková, Slovakia);
4.3. Volcano-sedimentary (Anderson Mine, USA; Sierra Pintada district, Argentina).

Type 5. Metasomatite

5.1. Na- metasomatite (Kirovograd District, Ukraine; Lagoa Real, Brazil);

5.2. K-metasomatite (Elkon district, Russian Federation);
5.3. Skarn (Mary Kathleen, Australia; Tranomaro, Madagascar).

Type 6. Metamorphite

6.1. Stratabound (Forstau, Austria; Malundwe, Zambia);
6.2. Structurally-controlled with 3 classes: *Monometallic veins* (Schwartzwalder, USA; Rozna, Czech Republic), *Polymetallic veins* (Shinkolobwe, Democratic Republic of Congo; Jaduguda, India) and *Marble-hosted phosphate* (Itataia, Brazil).

Type 7. Proterozoic unconformity

7.1. Unconformity-contact (Cigar Lake, Key Lake and McArthur River, Canada);
7.2. Basement-hosted (Jabiluka and Ranger, Australia; Eagle Point and Arrow, Canada);
7.3. Stratiform structurally-controlled (Lambapur and Chitrial, India).

Type 8. Collapse breccia pipe (Arizona Strip, Canyon, USA)

Type 9. Sandstone

9.1. Basal channel (Dalmatovskoye, Russian Federation; Beverley, Australia);
9.2. Tabular (Arlit district, Niger; Ambrosia Lake district, USA);
9.3. Rollfront (Wyoming Province, USA; Chu-Sarysu Basin, Kazakhstan);
9.4. Tectonic-lithologic (Lodève Basin, France; Franceville Basin, Gabon);
9.5. Mafic dykes/sills in Proterozoic sandstone (Westmoreland district, Australia; Matoush, Canada).

Type 10. Palaeo quartz-pebble conglomerate

10.1. U-dominant (Elliot Lake district, Canada);
10.2. Au-dominant (Witwatersrand Basin, South Africa).

Type 11. Surficial

11.1. Peat bog (Kamushanovskoye, Kyrgyzstan; Flodelle Creek, USA);
11.2. Fluvial valley (Yeelirrie, Australia; Langer Heinrich, Namibia);
11.3. Lacustrine–playa (Lake Maitland, Australia; Aquelt Assfaya, Mauritania);
11.4. Pedogenic and fracture fill (Beslet and Senokos, Bulgaria);
11.5. Placers.

Type 12. Lignite and coal

12.1. Stratiform (Ambassador, Australia; Springbok Flats, South Africa);
12.2. Structurally-controlled (Freital–Gitterse, Germany; Badyelskoye, Russian Federation).

Type 13. Carbonate

13.1. Stratabound (Tumalappalle, India);
13.2. Structurally-controlled (Mailuu-Suu, Kyrgyzstan; Todilto district, USA);
13.3. Karst (Sanbaqi, China; Tyuya-Muyun, Kyrgyzstan).

Type 14. Phosphate

14.1. Organic phosphorite (Mangyshlak Peninsula, Kazakhstan; Ergeninsky region, Russian Federation);

14.2. Minerochemical phosphorite (Phosphoria Formation and Florida Land Pebble district, USA; Gantour, Morocco);
14.3. Continental phosphate (Bakouma district, Central African Republic).

Type 15. Black shale

15.1. Stratiform (Ranstad and MMS Vicken, Sweden; Chattanooga Shale Formation, USA);
15.2. Structurally-controlled (Ronneburg district, Germany; Dzhantuar, Uzbekistan).

2.2. THE IAEA UDEPO DATABASE

The IAEA UDEPO database is an information system on the geological and technical characteristics of uranium deposits worldwide [4, 5]. The UDEPO web application (http://www-nfcis.iaea.org/), which is publicly accessible, is part of the IAEA Integrated Nuclear Fuel Cycle Information System (INFCIS), which displays technical and statistical information about nuclear fuel cycle activities worldwide. At the present time, it is IAEA policy to publish only resource and grade ranges, even if the exact resource and grade numbers are publicly available for a large number of deposits.

As of 31 December 2017, the UDEPO database listed 2939 uranium deposits/resources grouped into 15 geological types and 37 subtypes (Table 1). Fifty parameters are described for each deposit.

Two types of resources are distinguished in UDEPO: conventional and unconventional [4, 5]. For the 2016 Red Book, conventional resources are those that have an established history of production and where uranium is a primary product, co-product or an important by-product. Very low grade resources or those from which uranium is only recoverable as a minor by-product are considered unconventional resources, as exemplified by uranium resources associated to phosphates and black shales [7].

For UDEPO, which is primarily a geological database, there are no economic connotations. Unconventional resources/deposits are those that are not or cannot be mined solely for uranium. They include:

- Very large (generally >1 million tU), low grade (50–200 ppm U) resources such as those in volcanic formations (Latium district, Italy), lignite–coal (Northern Great Plains, USA), phosphorites (Morocco districts) and black shale formations (Baltoscandia district);
- Large (10 000–100 000 tU), low grade (100–250 ppm U) resources hosted within plutonic intrusions (Kvanefjeld, Greenland), polymetallic iron oxide breccia complexes (Olympic Dam, Australia) and Au-dominant auriferous palaeo quartz-pebble conglomerates (Witwatersrand deposits, South Africa);
- Very low grade (‘background’) uranium levels (10–30 ppm U) in porphyry copper intrusions (Bingham Canyon, USA) and base metal deposits (Talvivaara mine, Finland);
- Placer deposits.

As of 31 December 2017, total UDEPO geological uranium resources amounted to 62716 680 tU, hosted within 2733 deposits with known (exact figures) or estimated (ranges) resources (Tables 2 and 3). These comprise:

- Conventional uranium resources which amount to 11899 632 tU, hosted within 2453 deposits with known/estimated resources;
- Unconventional resources which amount to 50 817 048 tU, hosted within 280 deposits with known/estimated resources (Table 4). As indicated before, deposits such as Olympic Dam (polymetallic iron oxide breccia complex) are included in the unconventional resource category, which explains some of the differences in the data when compared, for example, with 2016 Red Book data.

TABLE 1. NUMBER OF URANIUM DEPOSITS FOR EACH DEPOSIT TYPE AND SUBTYPE (UDEPO 2017)
(as of 31 December 2017)

	Type of deposit	Number of deposits	Subtype	Number of deposits
1	Intrusive	129	1.1. Anatectic	89
			1.2. Plutonic	40
2	Granite-related	586	2.1. Endogranitic	387
			2.2. Perigranitic	199
3	Polymetallic iron oxide breccia complex	21		21
4	Volcanic-related	204	4.1. Stratabound	20
			4.2. Structurally-controlled	156
			4.3. Volcano-sedimentary	28
5	Metasomatite	152	5.1. Na-metasomatite	124
			5.2. K-metasomatite	21
			5.3. Skarn	7
6	Metamorphite	225	6.1. Stratabound	8
			6.2. Structurally-controlled	204
			6.3. Marble-hosted	13
7	Proterozoic unconformity	114	7.1. Unconformity-contact	47
			7.2. Basement-hosted	63
			7.3. Stratiform structurally-controlled	4
8	Collapse breccia pipe	18		18
9	Sandstone	951	9.1. Basal channel	118
			9.2. Tabular	458
			9.3. Roll-front	334
			9.4. Tectonic-lithologic	33
			9.5. Mafic dykes/sills	8
10	Palaeo quartz-pebble conglomerate	144	10.1. U-dominant	28
			10.2. Au-dominant	116
11	Surficial	123	11.1. Peat bog	8
			11.2. Fluvial valley	47
			11.3. Lacustrine–playa	48
			11.4. Pedogenic/fracture-fill	7
			11.5. Placer	13
12	Lignite–coal	75	12.1. Stratiform	69
			12.2. Structurally-controlled	6
13	Carbonate	34	13.1. Stratabound	3
			13.2. Structurally-controlled	28
			13.3. Palaeokarst	3
14	Phosphate	73	14.1. Organic phosphorite	20
			14.2. Phosphorite	48
			14.3. Continental phosphate	5
15	Black shale	75	15.1. Stratiform	37
			15.2. Structurally-controlled	38
	Unknown	15		22
		2 939		**2 939**

TABLE 2. URANIUM RESOURCES FOR THE 15 TYPES OF URANIUM DEPOSIT ACCORDING TO ORDER OF IMPORTANCE (UDEPO 2017)
(as of 31 December 2017)

Deposit type	Number of deposits	Number of deposits with resources	Resources (tU)*
Black shale	75	64	21 748 873
Phosphate	73	73	14 326 175
Lignite–coal	75	74	7 406 507
Sandstone	951	897	4 827 107
Intrusive	129	128	2 846 966
Polymetallic iron oxide breccia complex	21	21	2 562 526
Palaeo quartz-pebble conglomerate	144	143	2 503 879
Volcanic-related	204	195	1 908 537
Proterozoic unconformity	114	113	1 547 500
Metasomatite	152	121	1 070 106
Metamorphite	225	209	663 188
Surficial	123	121	532 100
Granite-related	586	526	526 912
Carbonate	34	30	184 285
Collapse breccia pipe	18	18	19 476
Unknown	15		
	2 939	**2 733**	**62 716 680**

TABLE 3. URANIUM RESOURCES CONTAINED IN URANIUM DEPOSIT SUBTYPES (UDEPO 2017)
(as of 31 December 2017)

Deposit type		Deposit subtype	Deposits with resources	Resources (tU)
1	Intrusive	1.1. Anatectic	88	910 843
		1.2. Plutonic	40	1 936 123
2	Granite-related	2.1. Endogranitic	348	238 602
		2.2. Perigranitic	179	288 310
3	Polymetallic iron oxide breccia complex		21	2 562 526
4	Volcanic-related	4.1. Stratabound	20	1 232 853
		4.2 Structurally-controlled	147	615 903
		4.3. Volcano-sedimentary	28	59 781
5	Metasomatite	5.1. Na-metasomatite	96	662 933
		5.2. K-metasomatite	18	380 888
		5.3. Skarn	7	22 785
6	Metamorphite	6.1. Stratabound	7	16 413
		6.2. Structurally-controlled	189	498 531
		6.3. Marble-hosted	13	148 244
7	Proterozoic unconformity	7.1. Unconformity-contact	46	696 276
		7.2. Basement-hosted	63	879 073
		7.3. Stratiform structurally-controlled	4	18 072
8	Collapse breccia pipe		18	19 476
9	Sandstone	9.1. Basal channel	108	372 479
		9.2. Tabular	446	2 346 211
		9.3. Roll-front	302	1 989 382
		9.4. Tectonic-lithologic	33	81 259
		9.5. Mafic dykes/sills	8	37 897
10	Palaeo quartz-pebble conglomerate	10.1. U-dominant	27	468 122
		10.2. Au-dominant	116	2 035 757
11	Surficial	11.1. Peat-bog	8	4 405
		11.2. Fluvial valley	50	319 932
		11.3. Lacustrine–playa	43	139 577
		11.4. Pedogenic/fracture-fill	7	1 239
		11.5. Placer	13	66 948
12	Lignite-coal	12.1. Stratiform	69	7 393 080
		12.2. Structurally-controlled	5	13 427
13	Carbonate	13.1. Stratabound	3	139 701
		13.2. Structurally-controlled	24	43 364
		13.3. Palaeokarst	3	1 220
14	Phosphate	14.1. Organic phosphorite	20	141 750
		14.2. Minerochemical phosphorite	48	14 148 025
		14.3. Continental phosphate	5	36 400
15	Black shale	15.1. Stratiform	36	21 492 797
		15.2. Structurally-controlled	27	256 076
			2 733	**62 716 680**

In UDEPO 2017, unconventional resources are associated with several deposit types and subtypes (Table 4):

- Type 1: Intrusive, plutonic subtype, with 3 classes (quartz monzonite, peralkaline complex and carbonatite);
- Type 2: Volcanic-related, stratabound and volcano-sedimentary subtypes;
- Type 3: Polymetallic iron oxide breccia complex;
- Type 10: Palaeo quartz-pebble conglomerate, Au-rich subtype ;
- Type 11: Surficial, placer subtype;
- Type 12: Lignite–coal, stratiform subtype;
- Type 14: Phosphate, minerochemical phosphorite subtype;
- Type 15: Black shale, stratiform subtype.

Almost all types and subtypes of uranium deposit listed in UDEPO contain anomalous to economic concentrations of metallic elements besides uranium and can therefore be termed polymetallic (see Appendix). Strictly, monometallic uranium deposits are rare. Today, in conventional deposits types and subtypes, uranium is the main component and is the sole element extracted; any accompanying elements are generally not taken into account owing to their low grade, to the cost of extraction of specific elements or to the relative abundance of the element on the world market. Uranium is a co- and by-product in several deposit types and subtypes associated with numerous elements such as REE-Cu-Au-Ag-Mo-Zn-Pb-Sn-V-Ni-Co-Ta-Nb-Zr-Y-W-Th-Ti-Li-P-F-PGE.

TABLE 4. UNCONVENTIONAL URANIUM RESOURCES (UDEPO 2017)
(as of 31 December 2017)

Deposit type/subtype	Number of unconventional deposits with resource data	Total resources (tU)	Grade range (% U)
Type 1: Intrusive, plutonic	33	1 907 085	0.002–0.025
Type 3: Polymetallic iron oxide breccia complex	18	2 759 738	0.0040–0.035
Type 4: Volcanic-related, stratabound and volcano-sedimentary	2	1 204 250	0.002–0.005
Type 10: Palaeo quartz-pebble conglomerate, Au-dominant (including tailings)	116	2 035 757	0.01–0.04
Type 11: Surficial, placers	13	66 948	
Type 12: Lignite–coal, stratiform	21	7 222 500	0.004–0.040
Type 14: Phosphate, minerochemical phosphorite	48	14 148 025	0.006–0.030
Type 15: Black shale, stratiform	29	21 472 745	0.001–0.040
Total	280	50 817 048	

3. URANIUM PROCESSING

The production of uranium from ores is a hydro-metallurgical process which uses chemicals to extract and purify the uranium from the host ore matrix. This is required because the final uranium product, known as yellowcake, must be of high purity to meet requirements for the next step in the nuclear fuel production cycle: refining.

There are four major systems for uranium processing:

- Conventional agitation leach;
- Recovery as a by-product;
- Heap leach;
- In situ leach (ISL), also refered to as in situ recovery (ISR).

By-product uranium is generally derived from geological sources referred to as (1) unconventional resources or (2) other metal production, where gold, copper or nickel is the primary product. There may also be situations where production of the primary product may require uranium removal for reasons of product purity, as in the case of copper concentrates at Olympic Dam in Australia, or for environmental reasons.

As per the Red Book, unconventional resources are resources from which uranium is only recoverable as a minor by-product, such as uranium associated with phosphate rock, non-ferrous ores, carbonatite, black schist and lignite [6]. For Schnell [8], “unconventional resources are resources from which uranium is only recoverable as a minor by-product, as a co-product, occurring in low to very low grades or requiring unconventional mining or processing methods”. The ranges of concentrations for these unconventional sources are shown in Fig. 2 [9]. Table 5 presents the average concentration of ore related to these deposit types.

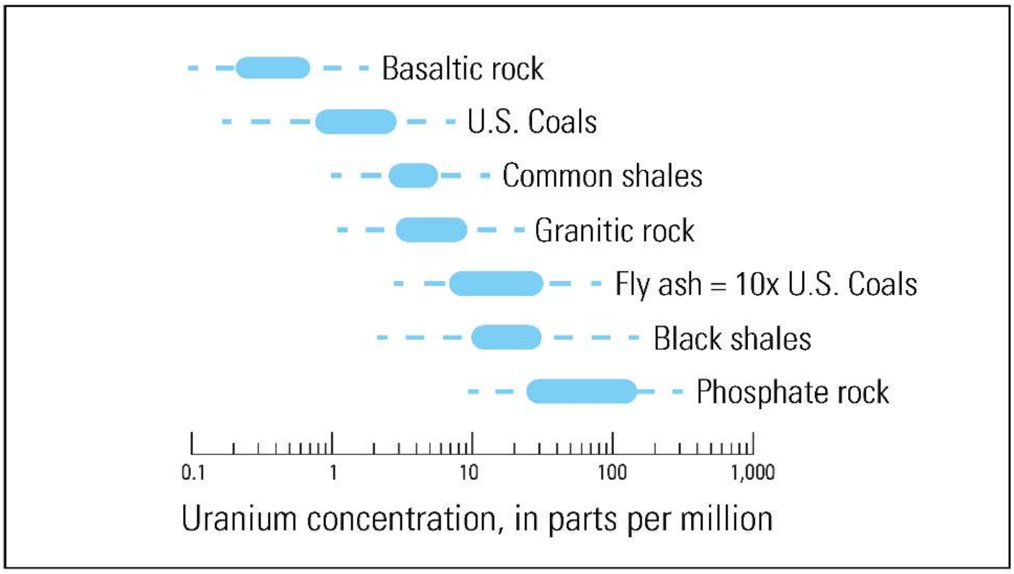

FIG. 2. Typical range of uranium concentration in coal, fly ash and a variety of common rock (adapted and redrawn from Ref. [9]).

As mentioned above, unconventional resources defined in the Red Book include phosphates, non-ferrous ores, carbonatite, black schist and lignite. However, there are other undefined sources for by-product production, such as tailings (ERGO, ChemWest), sea water, mineral sands, coal ash, coal associated (Springbock Flats) and metallurgical solutions. Many of these potential sources of uranium are not specifically discussed in this publication, which focuses on sulphides.

TABLE 5. TYPES OF ORE WITH URANIUM CONCENTRATION AND DEPOSIT TYPES

Type of ore	Range (% U)	Deposit type
Very high grade	5–20	Proterozoic unconformity
High grade	1–5	Proterozoic unconformity
Medium grade	0.1–0.5	Granite-related Volcanic-related Collapse breccia pipe Tabular sandstone Metamorphite Metasomatite
Low grade	0.01–0.05	Intrusive anatectic Intrusive plutonic IOCG-U Sandstone roll-front Palaeo quartz-pebble conglomerate Surficial Carbonate Black shale
Very low grade	0.001–0.01	Intrusive plutonic Phosphate Coal-lignite Black shale

3.1. BY-PRODUCT PRODUCTION CLASSIFICATION

There is potential that uranium will be recovered from a variety of unconventional resources in the future and there are some examples currently close to or in production. A possible alternative classification is to use the process options available for unconventional resources that is based on geological formations which will cause an overlap and confusion of process alternatives.

To avoid redundancy and provide a better understanding of processes for unconventional sources, a classification method based on production alternatives is presented below. These categories are based on general flowsheet priorities that also consider economic production parameters.

3.1.1. Category 1: By-production

Category 1 by-production is employed for resources where the primary product also contains a secondary and even tertiary product. Often, the secondary product is not in itself economic, but its extraction can be facilitated by the primary commodity production process (Fig. 3).

An example is the copper heap leach operation whereby uranium is co-leached during the copper extraction process. The leached uranium can then be recovered from the copper solution using solvent extraction or ion exchange.

Examples:

- Anglo Gold: Au and U (palaeo quartz-pebble conglomerate);
- Bingham Canyon: Cu and U (porphyry copper).

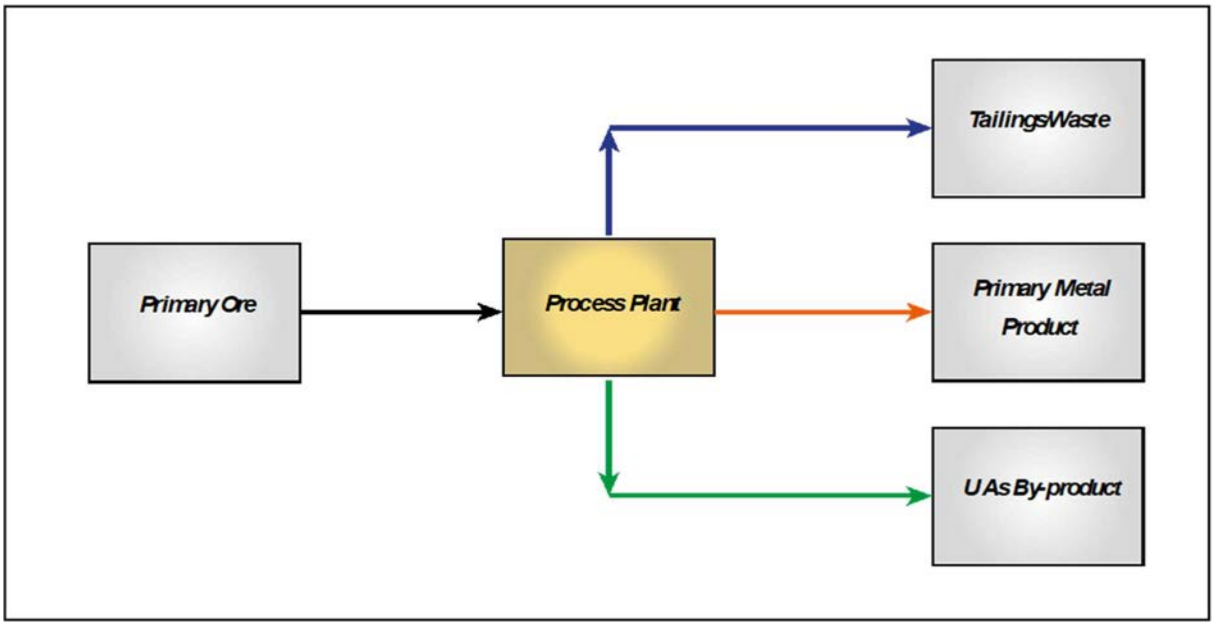

FIG. 3. Simplified flow diagram for by-product production of uranium (courtesy H. Schnell).

3.1.2. Category 2: Co-production

Category 2 co-production is where more than one metal is present in economic concentrations. This is common in base metal ores where typically copper and nickel or zinc, lead and silver are produced from a single ore source (Fig. 4).

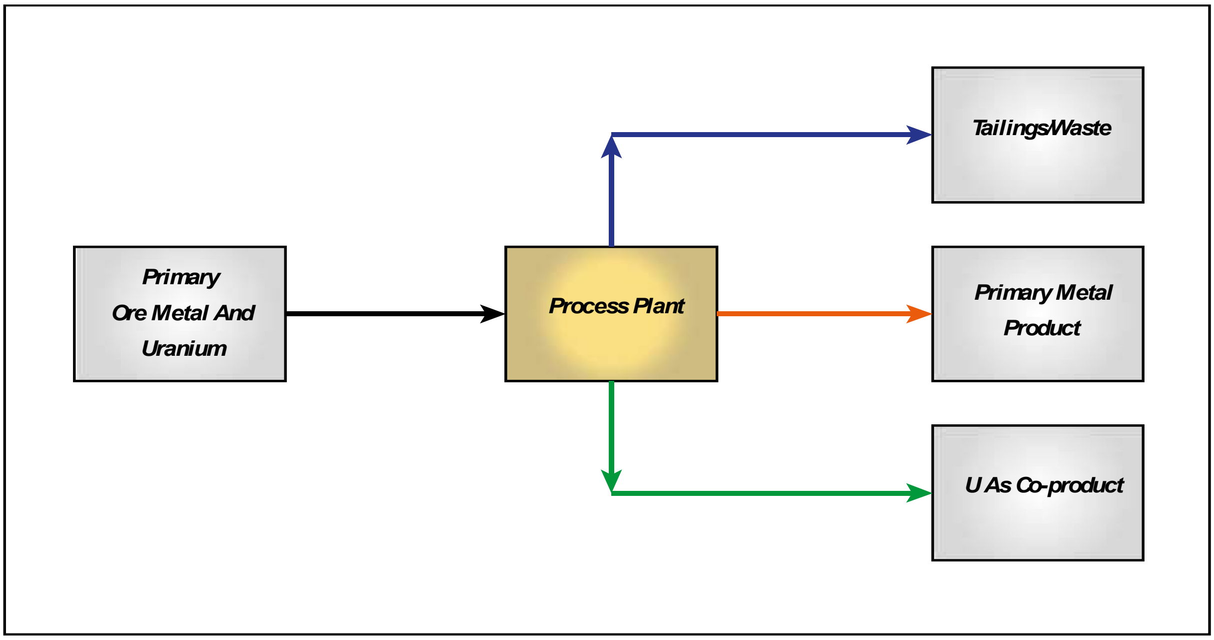

FIG. 4. Simplified flow diagram of co-production of uranium (courtesy H. Schnell).

The most notable example of co-production of uranium is the Olympic Dam operation, which records economic uranium production from what is, primarily, a copper producer. Within these categories, the Olympic Dam operation could also be placed in the next category (Category 3), as the primary source of the uranium is copper tailings. Considering the high value of the uranium, this operation would normally be considered a co-product relationship.

Talvivaara in Finland is an interesting case of multiple products. The Talvivaara operation produces copper, zinc, nickel and cobalt as products using a heap leach flowsheet in a cold climate. It also intends to recover uranium. This operation is a good example of 'comprehensive extraction'.

Examples:

- Olympic Dam: Cu, Au, Ag and U (polymetallic iron oxide breccia complex);
- Talvivaara: Cu, Zn, Ni, Co, (U);
- Colorado Plateau: U and V (sandstone, tabular).

3.1.3. Category 3: Tailings reprocessing

Category 3: tailings reprocessing concerns the treatment of tailings or mine waste. In general, this involves the reprocessing of previously disposed tailings (Fig. 5).

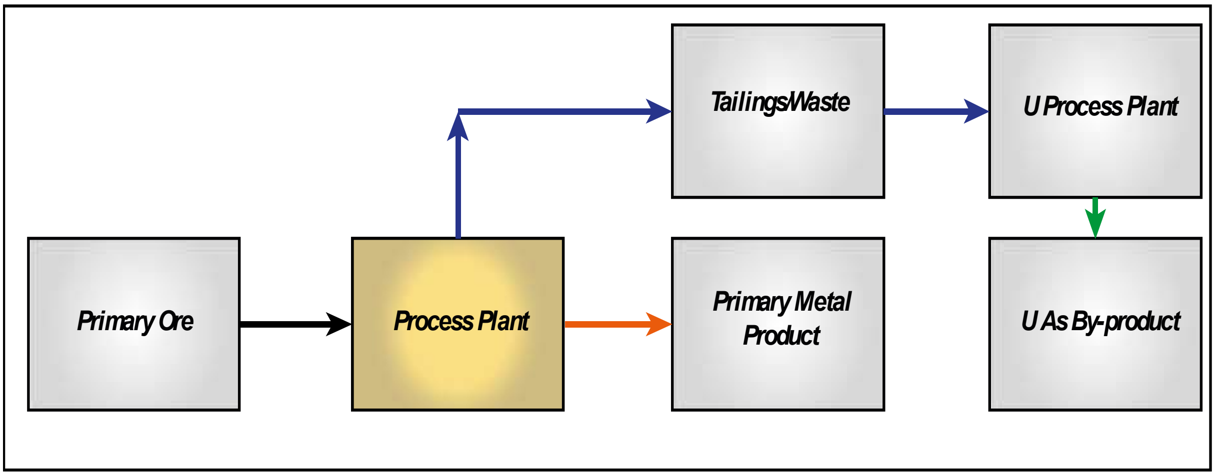

FIG. 5. Simplified flow diagram of uranium production from tailings (courtesy H. Schnell).

The best examples of tailings reprocessing are the recovery operations for gold in South Africa. The tailings are hydraulically mined with high pressure water and then fed to a leach plant for gold recovery. Some of these tailings also contain economic quantities of uranium values whereby this uranium may be recovered (Mine Waste Solutions plant of First Uranium Corporation). Similarly, a recent project at the SOMAIR operation in Niger employed a heap leach process on mine waste with previously uneconomic uranium values to produce 400–800 tU/year. As metal prices fluctuate according to market considerations, reprocessing of old tailings will continue. The optimal solution of course would be to extract the metals of economic interest during the first stage of hydrometallurgical processing.

Examples:

- Cluff Lake: Uranium tailings reprocessing (Fig. 6);
- South Africa: Gold tailings reprocessing (Mine Waste Solutions);
- SOMAIR: Mineralized waste reprocessing.

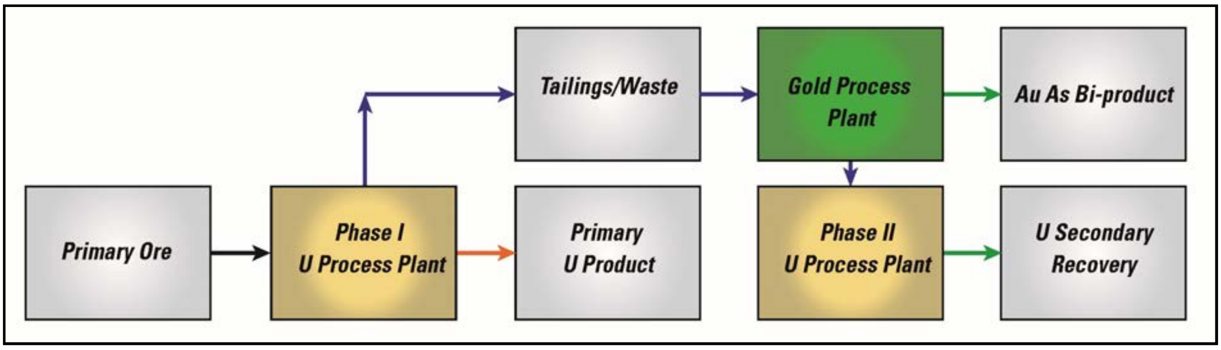

FIG. 6. Simplified flow diagram of tailings reprocessing for uranium production (courtesy H. Schnell).

3.1.4. Category 4: Unusual treatment

Category 4: unusual treatment involves the application of new innovative processes on previously uneconomic or unconventional resources (Fig. 7).

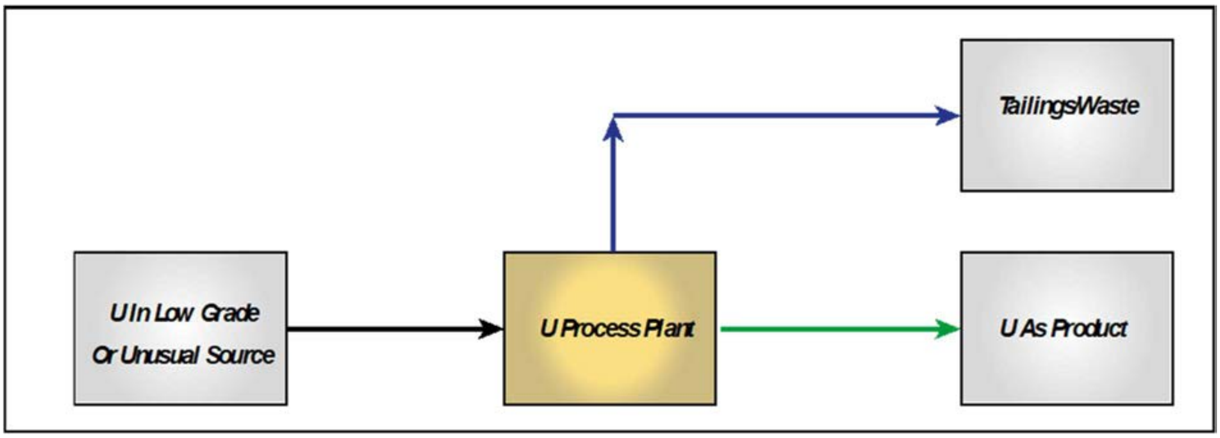

FIG. 7. Simplified flow diagram of unusual uranium production (courtesy H. Schnell).

An example is the Trekkopje deposit in Namibia, which has extremely low grades (0.012% U) that were formerly considered too low for conventional processing. However, the application of economies of scale, combined with heap leach technology and the new application of alkaline chemistry to heap leaching, has endowed this deposit with the potential for significant uranium production in the future.

Another unusual resource are the Bakouma district deposits in the Central African Republic. The major difficulty with these deposits is their high apatite content, which requires very high levels of acid consumption and subsequent neutralization. This characteristic alone can render a moderately good grade (0.2–0.3% U) uranium deposit uneconomic.

Examples:

- Trekkopje (Surficial, fluvial valley);
- Sea water;
- Bakouma (continental phosphate);
- Chile (U extracted from copper heap leach solutions).

3.2. PROCESS SELECTION

Initial metal production from mining operations generally produces a metal concentrate that is then further refined to a final metal product. This first step in producing a concentrate employs two basic processes: mineral processing (using a mineral's physical characteristics) and hydrometallurgical processing (using chemical dissolution and recovery). The vast majority of base metal production uses mineral processing (Fig. 8).

Most base metal extraction employs mineral processing methods such as flotation, gravity concentration, electrostatic and magnetic separation, and similar methods dependent upon mineral property characteristics (Fig. 8). In addition, recovery of metals of interest often involves hydrometallurgical methods to produce a concentrate. Gold and uranium production are the most common metals recovered through hydrometallurgy. The final gold product, a 'Doré' gold bar, also relies on smelting to produce a concentrate that will still include substantial impurities.

Uranium concentrate or yellowcake has much more stringent impurity requirements (minimum ASTM defined levels) and requires a more complex flowsheet. All yellowcake production depends on hydrometallurgical operations.

Uranium as a by-product of mineral processing or in 'milling' plants must be separated from the other metallic minerals by exploiting some mineral characteristics. There are a mulititude of uranium minerals and they generally occur as admixtures, making efficient recovery by mineral processing extremely difficult. Such a separation will then require addition of a hydrometallurgical flowsheet to the mineral processing plant, as is the case for the Olympic Dam operation in Australia.

Conversely, production of by-product from uranium is more obvious, but the limited capacity of uranium plants and the potential for contamination of a base metal product with radionuclides pose competitive and technical challenges.

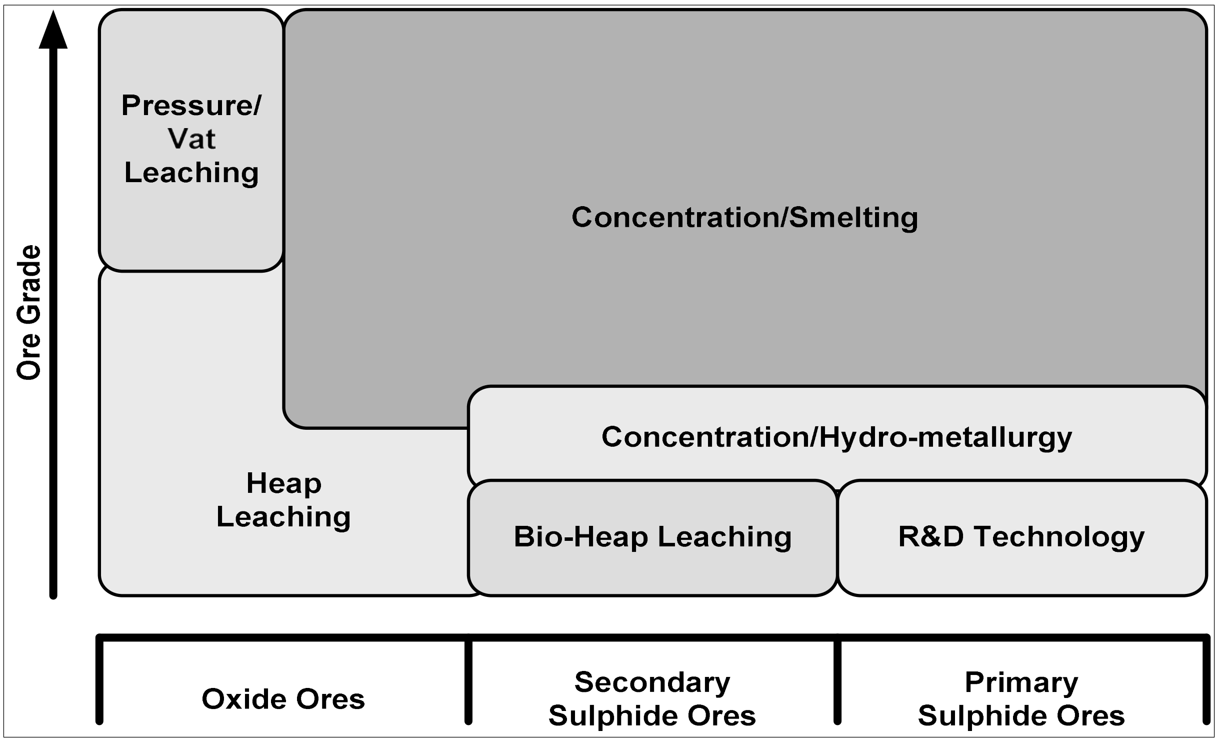

FIG. 8. Basic sulphide mineral production choice (courtesy H. Schnell).

3.2.1. Uranium processing unit operations

3.2.1.1. Process selection

There are basically three basic process alternatives: (i) conventional, (ii) heap leach and (iii) in situ leach (ISL or ISR). Although the term ‘conventional’ is used, these plants comprise a wide variety of processes and equipment such that no two processing plants are the same. In reviewing the three sets of unit operations, as shown in Fig. 9, heap leach has the advantage of not necessarily requiring a grinding stage and of eliminating solid–liquid separation. In ISL, the mine infrastructure is eliminated but the orebody itself now forms the leach section of the plant and accordingly offers much less control. ISL has the least opportunity for by-product production because the leach solution is generally quite specific to uranium and is, therefore, not considered in this publication.

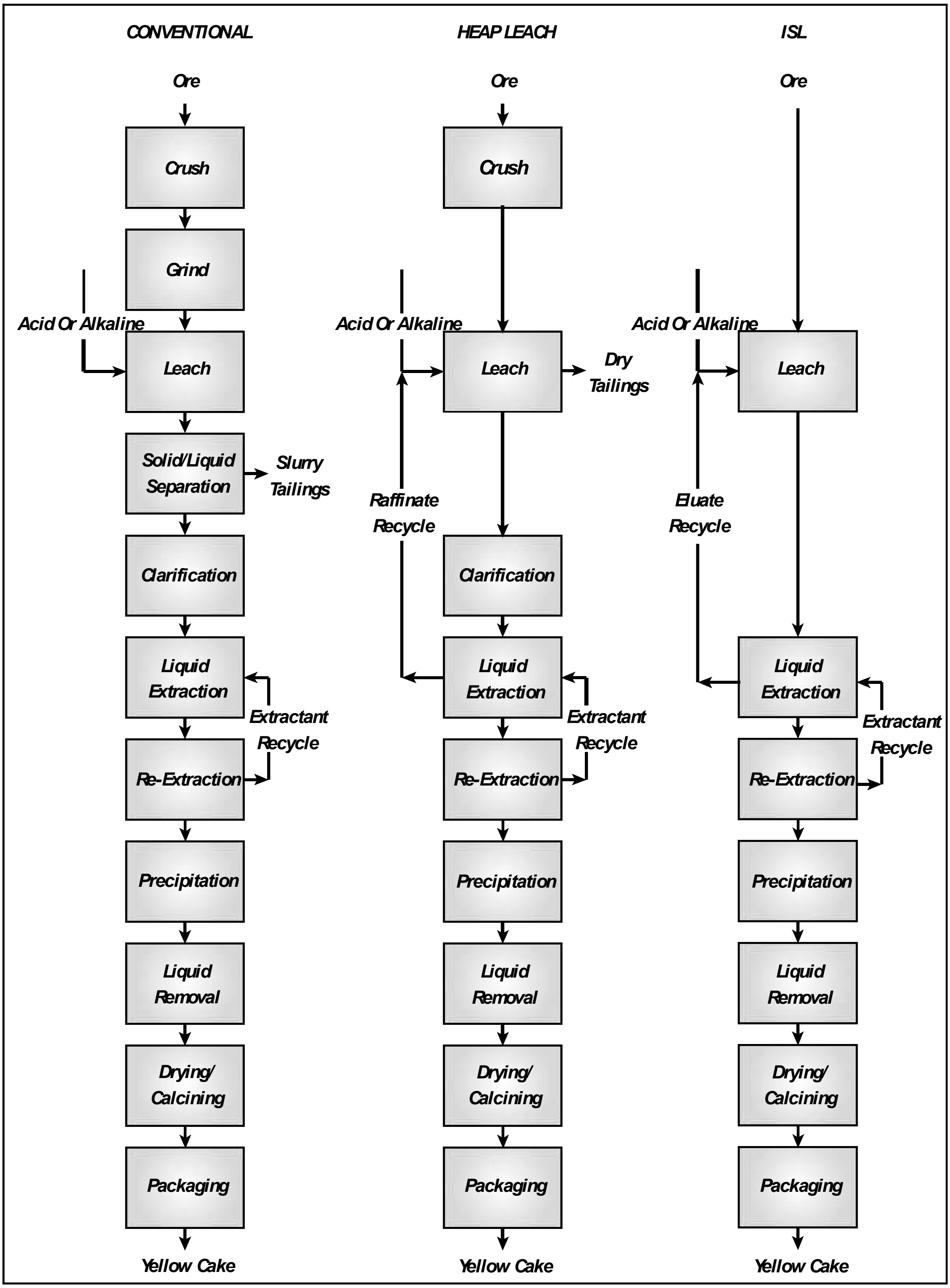

FIG. 9. Principal uranium process flow diagrams (courtesy H. Schnell).

3.2.1.2. Crushing

Uranium most commonly occurs along the fissures or cracks within the ore or gangue matrix and extraction at such fissures and cracks preferentially exposes the uranium minerals. The most common means of crushing are the gyratory and jaw crushers, but there is a current trend towards the use of sizers for low grade operations.

Crushing is generally limited down to the 10–15 mm size, which is usually not sufficient to expose all the uranium minerals, which is required for effective leaching. In addition, other base metal mineral species are generally not exposed at ththis particle size.

3.2.1.3. Pre-Concentration

Historically, radiometric sorting has been used to reduce the tonnage to the main uranium processing plant and this option is still studied with several pilot ore sorters installed.

The pre-concentration step at the Langer Heinrich operation in Namibia, where uranium in the form of carnotite is hosted in calcrete ore [10], uses an attrition grind followed by sizing of the various ore fractions to reduce ore tonnage, because most of the uranium is contained in the finer fractions.

Gravity separation has been applied to uranium ores in the past [11] with some success, but is associated with serious radiation issues. This method 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 carry over coming from McArthur River underground mine backfill [12].

3.2.1.4. Grinding

Ore particle sizes in the 10–15 mm range are further reduced by the use of grinding mills, but the final grind size is generally coarse (100–500 µm) compared with other extractive operations, because, in the case of uranium, only mineral exposure is required, rather than mineral liberation.

Most grinding operations are wet and use either coarse ore to grind the fine ore (semi-autogenous grinding (SAG)) or steel rods or balls in more conventional grinding circuits. SAG milling is most applicable to softer ores.

An exception is dry grinding, as exemplified 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 uranium minerals from an expanding clay matrix.

Grinding employed for high grade deposits generally uses small autogenous mills, as in the case of McClean Lake in Canada. McArthur River and Cigar Lake uranium mining operations (also in Canada) have small autogenous mills sited underground to avoid hoisting high grade ore. In these cases, ore is transported to surface as ground slurry by means of positive displacement pumps. Both operations then transport the ground slurry by truck in special dual containment tanks about 80 km to the mills.

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

3.2.1.5. Conventional leach circuits

Following comminution (crushing and grinding), uranium minerals in the ore are exposed and available for dissolution of uranium in a hydrometallurgical process. There are two basic uranium leaching alternatives — acid and alkaline — with U^{VI} dissolving and forming either a sulphate or a carbonate.

Gangue minerals in the ore will determine whether acid or alkaline leaching is preferable, primarily depending upon reagent consumption costs. Conventional acid leaching is most commonly conducted using sulphuric acid (25–200 kg acid/t), typically at about 55°C. Higher temperatures, such those used in pressure leach, result in silica dissolution, making the subsequent unit operations more difficult.

Leaching can be carried out using a wide variety of equipment types — heap, vat, agitated, autoclave, etc., — all depend upon numerous variables, determined by laboratory and pilot-scale testing. The interdependence of time/temperature/pressure variables results in an increase in one variable while decreasing the other variables. Other minor variables have a similar characteristic, but their effects are perhaps not as dramatic.

Acid leach chemistry dissolves U^{VI} directly, but the U^{IV} species must first be oxidized. Through the electrochemical series iron in solution oxidizes first from ferrous iron to ferric iron which then oxidizes tetravalent uranium to hexavalant uranium. Leach solutions that are low in iron may exihibit reduced leach kinetics and therefore require longer retention times. Typical oxidants are sodium chlorate, manganese dioxide, hydrogen peroxide and oxygen. Oxygen has become most desirable due to low cost oxygen production using vacuum swing adsorption technology.

There is a general move away from the use of chlorates owing to environmental and corrosion issues. Use of pyrolucite (MnO_2) is still common where it is economically available. The simplified chemistry of alkaline leaching is oxidation followed by formation of a uranyl carbonate complex.

Acid leach reactions

- $UO_2^{2+} + SO_4^{2-} \leftrightarrow UO_2SO_4$
- $UO_2SO_4 + SO_4^{2-} \leftrightarrow [UO_2(SO_4)_2]^{2-}$
- $[UO_2(SO_4)_2]^{2-} + SO_4^{2-} \leftrightarrow [UO_2(SO_4)_3]^{4-}$
- $6Fe^{2+} + NaClO_3 + 6H^+ \leftrightarrow 6Fe^{3+} + NaCl + 3H_2O$
- $UO_2 + 2Fe^{3+} \leftrightarrow UO_2^{2+} + 2Fe^{2+}$

Alkaline leach reactions

- $2UO_2 + O_2 \leftrightarrow 2UO_3$
- $UO_3 + Na_2CO_3 + 2NaHCO_3 \leftrightarrow Na_4UO_2(CO_3)_3 + H_2O$

Continuous tank leach typically uses an agitated system of no less than 5 or 6 reactors arrayed in series to eliminate short circuiting. Typical acid leach times are 12-24 h for a 35–50% solids slurry at 50–60°C (Table 6). 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 by providing a pre-leach function.

TABLE 6. 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	12-24 h	48–96 h
Pressure	Atmospheric	Pachuca or autoclave
Oxidant	H_2O_2, O_2, MnO_2	O_2
Temperature	50–60°C	85–90°C
Recovery	93–99%	85–90%

An alternative to the typical agitated tank leach are the SOMAIR and Cominak operations that use a pug leach process where the dry ore is combined with concentrated sulphuric and nitric acids in a drum, after which the moist mixture is cured on a slow moving conveyor for two hours before entering agitating tanks and diluted prior to solid–liquid separation [13]. The Niger ores contain a mineral structure of siliceous clay that requires a concentrated acid curing step to achieve good metal recovery.

Carbonate leaching has re-emerged with the startup of the Langer Heinrich operation in Namibia and Trekkopje will be the first application using carbonate in a heap leach operation [14].

Leaching equipment ranges from pressure leach (Key Lake and McClean Lake) to agitated tank leach (Langer Heinrich, Rössing, etc.) and a recent return to Pachuca leach [12].

3.2.1.6. Heap leach

Although some heap leach work was utilized in the 1970s and 1980s, uranium heap and dump leach did not develop as rapidly or as completely as it did in gold and copper extraction primarily due to high relative production costs, poor uranium recovery and the availability of higher grade deposits. Recent increased interest in lower grade deposits and the efforts of countries to secure internal uranium production have resulted in expanded heap leach application in uranium production. Heap leaching based on the fine crushed agglomerated technology of the copper operations in Chile which use sulphuric acid has been applied for many years at Caetite operation in Brazil [15] and, more recently, at SOMAIR in Niger for the processing of mineralized waste [16]. In addition, the large Imouraren project in Niger will also employ crushed agglomerated heap leach [17]. Many junior mining companies have been assessing acid heap leach as a means of exploiting low grade deposits and it is anticipated that more heap leach uranium operations will arise in the future as the application of this technology to uranium extraction matures. The Trekkopje project in Namibia is of significant interest in that it will apply crushed agglomerated heap leach with carbonate (alkaline) leach chemistry.

The objectives of an optimum heap leach operation are:

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

Most difficult is the consistent and thorough leaching of all material, irrespective of location within a heap. This challenge requires that leach conditions are the same everywhere in the heap and that the ore is also equally responsive.

When considering heap leach (Fig. 10) it is essential to have:

- Conducted months (years) of careful column test work and to have confirmed favourable results in all areas of the orebody;
- An orebody that has been carefully sampled to assess its variability;
- Conducted initial test work not only on a blended sample, but also to assess variability to better predict actual performance.

The general operating conditions for uranium heap leach are presented in Table 7.

TABLE 7. GENERAL URANIUM HEAP LEACH OPERATING CONDITIONS

Parameter	Acid heap leach	Alkali heap leach
Typical U grade	200–500 ppm	150–500 ppm
Crush size	10–25 mm	10–25 mm
Clay content	<35%	<35%
Agglomeration time	45–90 s	45–90 s
Agglomeration moisture	7–10%	7–10%
Heap height	6–9 m	7–9 m
Irrigation rate	4–10 $L\cdot min^{-1}\cdot m^{-2}$	5–10 $L\cdot min^{-1}\cdot m^{-2}$
Leach reagent	10 g/L H_2SO_4	40 g/L CO_3^{2-} and 10 g/L HCO_3^-
Leach cycle	1 or 2 counter-current	3 to 4 counter-current
Leach time	30–90 d	120–180 d
Recovery	80–85%	75–80%

Heap leach seems simple, but there are some pitfalls, such as poor heap permeability due to compaction or jarosite ($KFe_3(OH)_6(SO_4)_2$) formation. Compaction can cause possible increased phreatic head resulting in heap failures and heap instability, and extraordinary measures are required to correct such problems. Solution flow considerations, including rain events, require design that considers the inherent solution energy and avoids solution discharges into the environment.

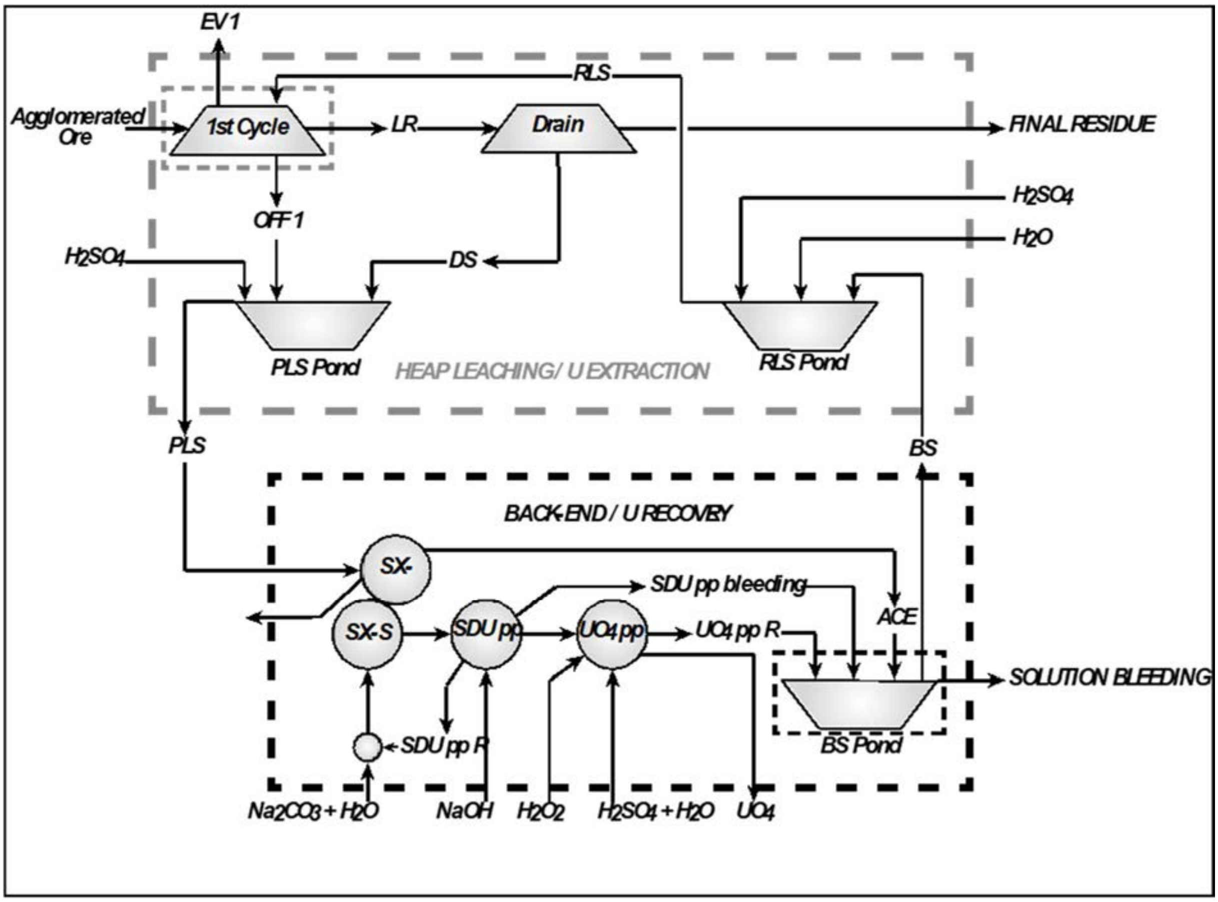

FIG. 10. Acid heap leach flowsheet [17].

3.2.1.7. Solid–liquid separation

After leaching the uranium into a soluble complex, the leach solution must be separated from the residue solids by physical separation using either decantation or filtration. The solid–liquid separation choices for uranium depend on the characteristics of the leached pulp — essentially how the pulp filters or decants. The options used are numerous and include: filtration (belt, drum, pressure), counter-current decantation (CCD), counter-current cyclones, counter-current classifiers or resin-in-pulp.

The resin-in-pulp techniques has recently been applied, as exemplified by 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 (Fig. 11).

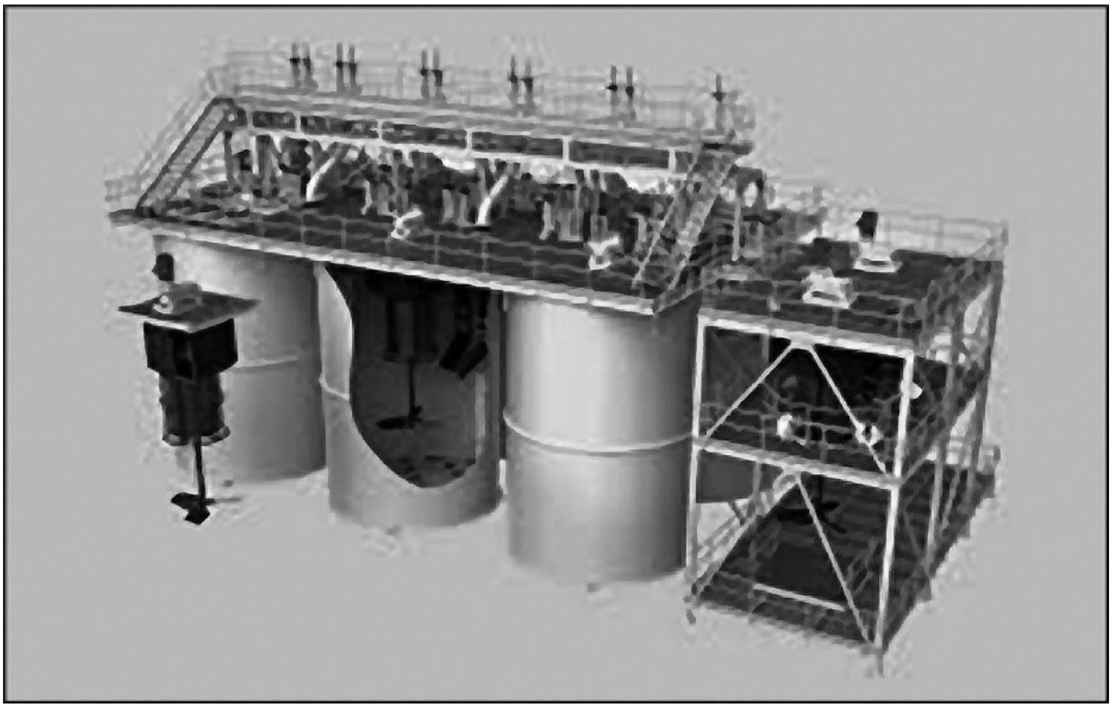

FIG. 11. Resin-in-pulp example [18].

3.2.1.8. Solution clarification

Uranium operations generally have lower volumetric throughput relative to base metal operations and subsequently clarification is a viable option in most conventional and heap leach operations. The pregnant leach solution will commonly contain fine solids that need to be clarified prior to solution purification. Conventional clarifiers and sand filtration are the norm. Some continuous sand filtration types have shown poor availability, but the use of pinned bed clarifiers has been applied more recently to deal with high pregnant leach solution flows [17].

3.2.1.9. Thickening

Settling of solids from solution typically uses a thickener or decanter or settler (different terms are often used for same item of equipment) with the solids being raked into the centre and with the uranium-bearing solution overflowing. The decantation circuits operate as counter-current decantation units typically using 5–8 thickeners arranged 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.

3.2.1.10. Filtration

Filtration is another option for separating the uranium solution from the pulp and washing the pulp to reduce uranium losses. There is a wide variety of filtration equipment 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 onto a conveyor belt which carries the washed pulp to the tailings treatment process.

3.2.1.11. Solvent extraction

Various other metals are often dissolved during uranium leaching, a situation that may require purification of uranium-bearing solutions. The purity requirements of the final uranium precipitate are becoming stricter and most uranium conversion plants require a precipitate quality better than the ASTM standard. The most common impurities associated with uranium are molybdenum and vanadium, but consideration should be given to other metal contaminants, most notably zirconium, arsenic, copper and nickel.

Solvent extraction (SX) is the norm for uranium, but little progress has been made for extractants and modifiers and the tertiary amines predominate with some use of DEPHA. Equipment is usually conventional mixer–settlers or Krebs mixer–settlers as used in Canada (McClean Lake and Key Lake) (Figs 12 and 13). Bateman columns have been used [19] at Olympic Dam in Australia. Vertical smooth flow technology is being considered for future projects [20].

Uranium stripping in the solvent extraction process is achieved with ammonium sulphate (Key Lake, McClean Lake), sodium carbonate (SOMAIR), or strong acid (Rabbit Lake). In one interesting alternative, uranium is precipitated directly in the organic solvent. Extractants are generally tertiary amines with typical concentrations of 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.

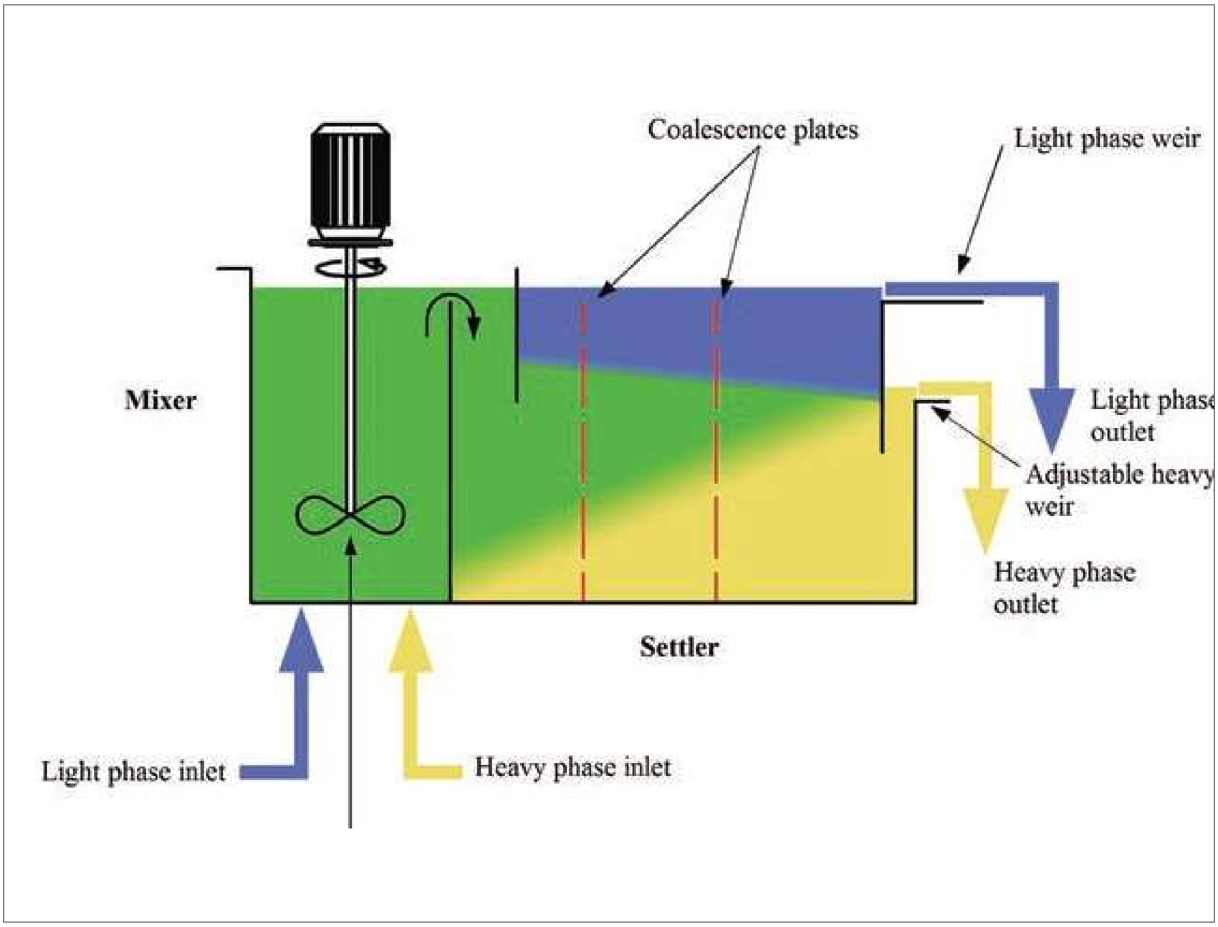

FIG. 12. Typical schematic of an SX mixer–settler. Figure permission through Creative Commons [21].

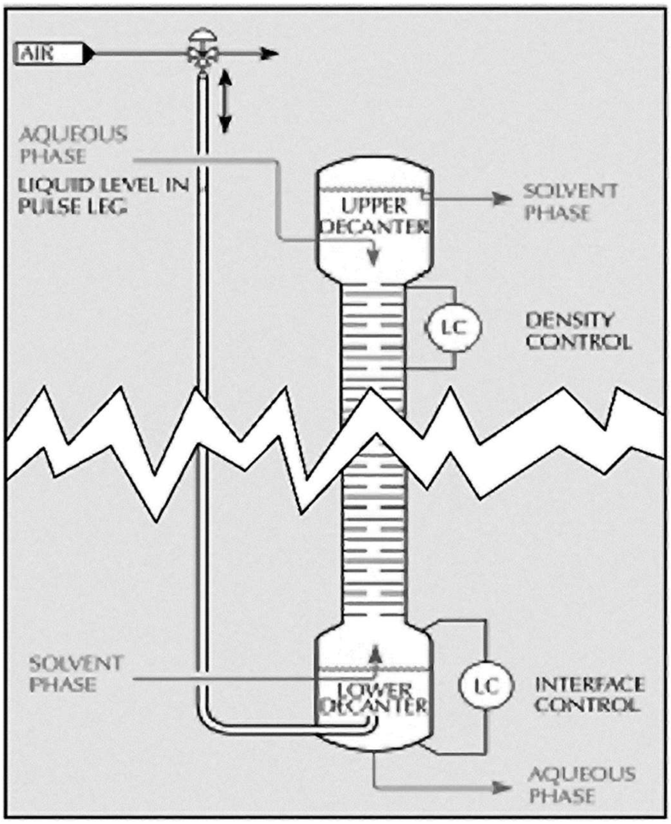

FIG. 13. Schematic of an SX pulse column. Modified from [19].

3.2.1.12. Ion exchange

Ion exchange (IX) has progressed in both resin and equipment design and recovery of uranium from lower grade deposits and in situ leach operations with lower grade solutions makes IX the preferred choice (Fig. 14).

Normally, IX is chosen for uranium solution grades in the <1.0 g/L range, although slightly lower solution grades may use the SX option. In the case of IX, there may also be a need for additional solution purification, which usually adds a small SX circuit, a process known as the Eluex process [22]. The most notable example of an Eluex circuit is at the Rössing operation in Namibia [23].

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

FIG. 14. Basic IX and SX reactions.

There are three basic technologies used for uranium IX: (i) fixed bed columns; (ii) pinned bed columns, and (iii) fluid bed columns (NIMCIX). Fixed bed columns have been applied in the Langer Heinrich operation, with pinned bed predominant in the in-situ leach operations in Kazakhstan. NIMCIX has been installed at the Trekkopje. NIMCIX offers the advantage of handling high levels of suspended solids.

3.2.1.13. Precipitation

After the uranium-bearing solution has been purified and concentrated, the uranium can then be precipitated. The process selected for final uranium precipitation depends on several factors, including solution chemistry, environmental constraints, reagent availability and converter requirements. The most common reagents used are ammonia gas, hydrogen peroxide, sodium hydroxide and magnesium oxide. However, most converters favour oxide precipitates, rather than an alkali precipitate, because the latter results in additional environmental concerns.

Equipment choice is most commonly expressed in a series of stirred tanks, but recent innovation using a fluid bed [24] produces an improved product with less dusting (Fig. 15). One unusual case (Kazakhstan) precipitates the uranium directly in organic solvent.

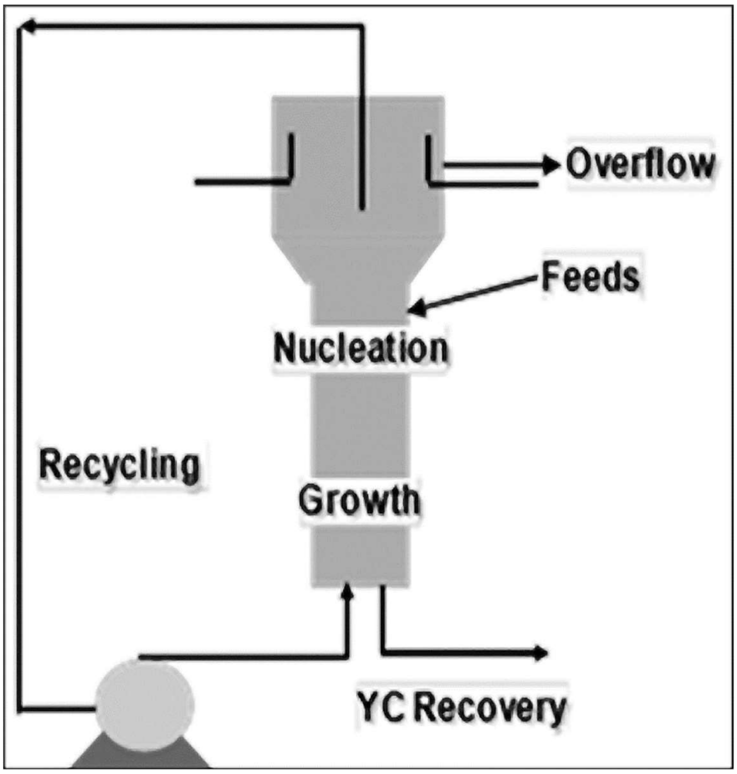

FIG. 15. Schematic of fluid bed precipitation. Modified from [24].

The various chemical reactions of precipitation are:

Magnesia precipitation

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

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

Sodium precipitation

$$2Na_4UO_2(CO_3)_3 + 6NaOH \rightleftarrows Na_2U_2O_7\downarrow + 6Na_2CO_3 + 3H_2O$$

$$Na_4UO_2(CO_3)_3 + 2NaOH \rightleftarrows UO_2(OH)_2\downarrow + 3Na_2CO_3$$

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 Fig. 16 is a typical tank precipitation circuit with thickener operation to recycle precipitate and feed a filter before final precipitate drying.

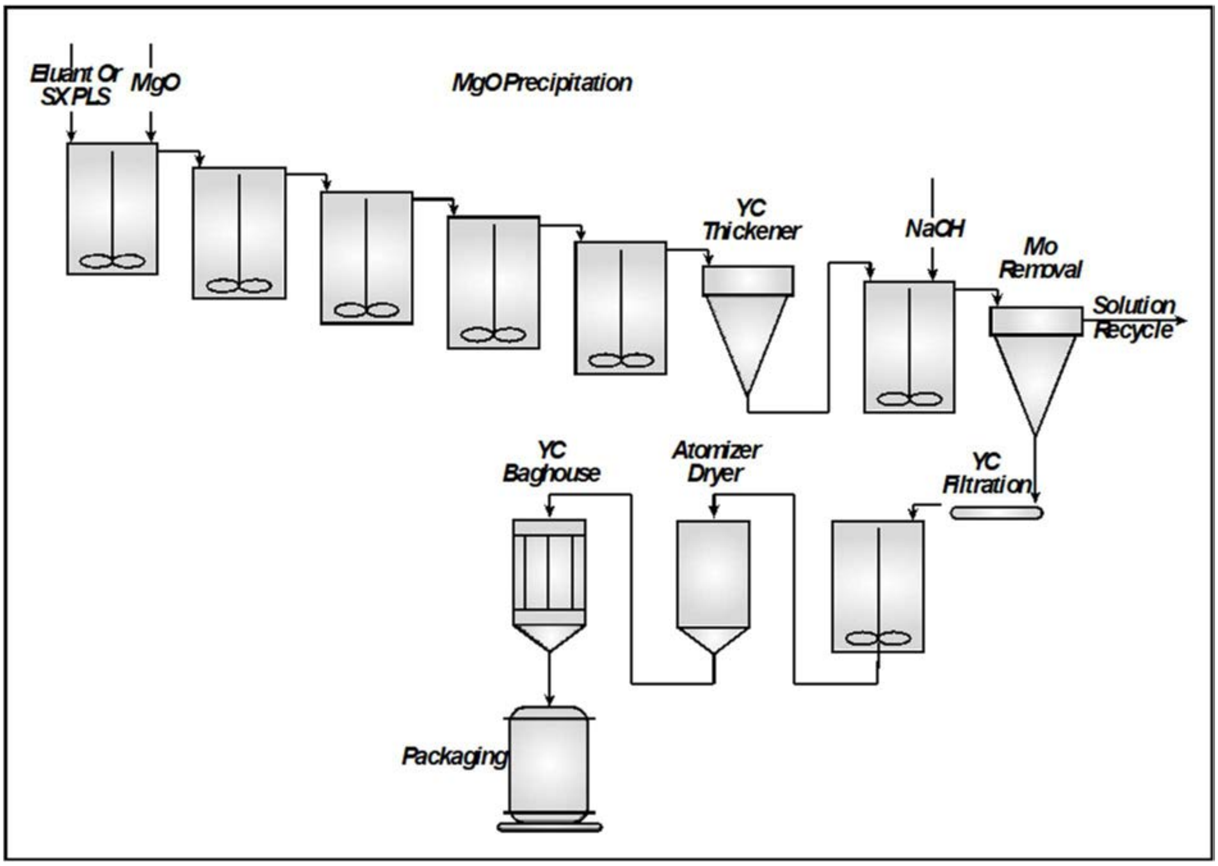

FIG. 16. Typical tank precipitation circuit (courtesy H. Schnell).

3.2.1.1.4. Drying and packaging

The precipitate is generally dewatered before drying by employing a centrifuge or filter (horizontal filter or pressure filter). The final dewatered precipitate is then dried or calcined, with the final product referred to as yellowcake. Commercially, it is then marketed as U_3O_8, although there is a general attempt to express everything as U, rather than U_3O_8. The colour is dependent upon the final drying temperature – yellow at 100°C, orange at 205°C and very dark green (black) at more than 800°C. The final product has a U concentration above 75%.

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 driers (Rabbit lake and Langer

Heinrich) to spray driers (Cominak). The choice of calcining or drying to a lower temperature is dependent upon economics and destination of the final product.

In all cases, the final precipitate is packed into standard 210 L steel drums for shipping. Packaging equipment and installations are continually being upgraded to reduce personnel exposures to yellowcake dust within these facilities, which are generally located in a separate enclosed space with remote operation and drum lids added with glovebox type equipment.

3.2.2. 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 these streams, and other less important streams, require environmentally acceptable treatment and disposal. Some of these waste streams also have values that must be taken into account, such as the solutions in alkaline leaching, which contain reagent values that can be recovered and recycled.

Liquid wastes must be treated to neutralize acidity or alkalinity, remove radioactive species such as radium, and remove other metals dissolved in the process. The tailings pulp needs similar treatment and neutralization. In Canada, waste solutions are treated and then temporarily stored and analysed to ensure environmental compliance prior to the water being released into the environment. These streams are heavily monitored to ensure compliance and minimize any environmental effects.

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

3.2.3. Tailings solids

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 in-pit disposal or lined cell disposal has become a requirement. The schematic of in-pit tailings disposal as practiced in the most recent operations (Rabbit Lake, McClean and Key Lake) is shown in Fig. 17.

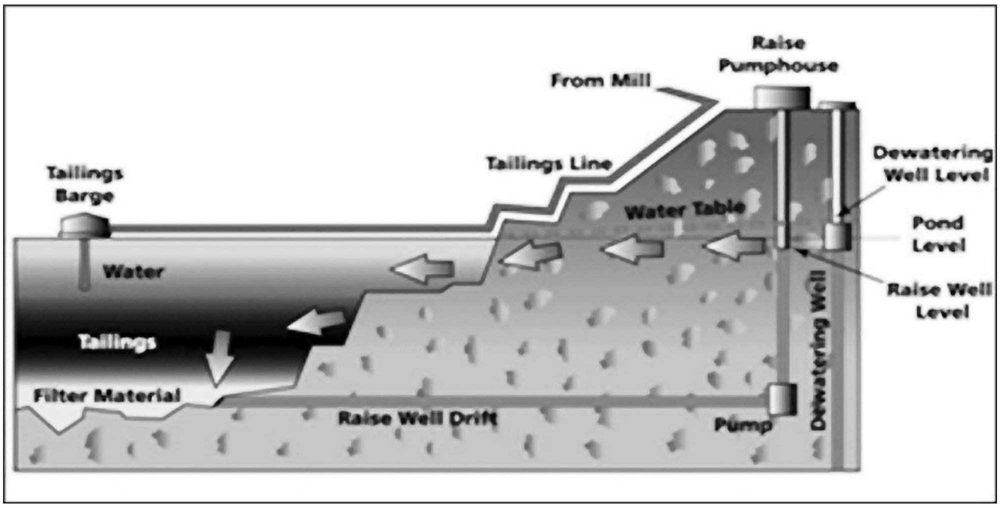

FIG. 17. Example of in-pit tailings disposal, modified from [25].

4. CASE STUDIES

Uranium forms many associations with other metals. This section summarizes these associations using the 2013 IAEA geological classification of uranium deposits [1–4] and the IAEA's UDEPO database. Thus, deposits/resources are presented in the order shown in Tables 1 and 3.

This report is focused on the production of uranium as a by- or co-product of the extraction of other metals such as copper or gold. It is also notable that other metals have been produced as by-products of uranium production.

A significant portion of historical US uranium production has been derived from uranium/vanadium deposits in the northern Colorado Plateau. By-product production has included gold at Cluff Lake (Canada), molybdenum at Akouta (Niger), Streltsovka (Russian Federation) and Poços de Caldas (Brazil), yttrium at Elliot Lake (Canada) and copper at Mexican Hat and Moab (USA). In addition, the Aktau uranium mine in Kazakhstan produced by-product scandium, rare earths and phosphate.

Copper and uranium are closely associated in by- or co-product relationships in several different geological environments, including intrusive magmatic deposits, polymetallic iron oxide breccia complexes and tabular sandstone deposits. Olympic Dam in Australia is the premier example of a polymetallic iron oxide breccia complex which contains significant uranium. Small quantities of uranium have been recovered from three copper mines in India's Singhbhum thrust belt which also hosts several uranium mines that have produced small quantities of by-product copper.

Co-existence of uranium and copper has also been reported in the Lumwana copper deposits in Zambia's copper belt and in uraniferous siltstones in central Sinai in Egypt.

In 2016, about 75 uranium deposits were mined worldwide with a production of 62 012 tU. Most uranium deposits are conventional deposits and uranium is generally the only element recovered, even if other elements are sometimes present in anomalous concentrations. Olympic Dam (Australia), the mines of the Singhbhum Belt (India) and those in the Witwatersrand Basin (South Africa) are the only ones producing uranium as a by-product of copper and gold mining. Molybdenum is produced as a by-product in Niger (Arlit district) and was produced in the Russian Federation (Streltsovska caldera) (Table 8).

Total uranium production to 2016 is 65 235 tU for Olympic Dam, 160 500 tU for the Witwatersrand Basin and about 12 000 tU for the Singhbhum Belt, representing 8.5% of the historical world uranium production which stands at 2 864 200 tU in 2016 (Table 9).

Table 10 lists deposits/districts where uranium was the main element produced. Table 11 lists the projects with planned production of uranium as a co- or by-product in the near future.

TABLE 8. DEPOSITS PRODUCING URANIUM AS PRIMARY, CO- AND BY-PRODUCTS (2013–2016)

Deposit/district	Country	Production (tU)				Deposit type	Elements recovered
		2013	2014	2015	2016		
Olympic Dam	Australia	3399	3952	3161	3235	Iron oxide breccia complex	Cu–(Au–Ag–U)
Arlit district	Niger	2730	2331	2509	2164	Sandstone tabular	U–(Mo)
Priargunsky	Russian Federation	2133	1970	1977	1873	Volcanic-related	U–(Mo)
Akouta	Niger	1508	1501	1607	1313	Sandstone tabular	U–(Mo)
Vaal Rivers	South Africa	531	504	346	423	Palaeo quartz-pebble conglomerate	Au–(U)
Singhbhum Belt (7 deposits)	India	400	385	385	?	Metamorphite	Cu–(U)
Ezulwini–Cooke	South Africa	0	69	47	67	Palaeo quartz-pebble conglomerate	Au–(U)
World uranium production		59 637	56 217	60 496	62 012		

TABLE 9. PRODUCTION OF URANIUM AS A BY- OR CO-PRODUCT (to 2016)

Deposit type	Deposit name	Historical production (tU)	Geological Resources (tU)	Contained metals	Status
Polymetallic iron-oxide breccia complex	Olympic Dam (Australia)	65 235	2 100 000	Cu–(Au–Ag–U)	Operating (Cu–Au–Ag–U)
Metamorphite	Singhbhum Belt (India)	12 230	60 000	Cu–(U)	Operating (Cu–U)
Palaeo quartz-pebble conglomerate	Witwatersrand Basin (South Africa)	160 500	2 000 000	Au–(U)	Operating (Au–U)
Intrusive plutonic	Bingham Canyon (USA)	420	25–50 000	Cu–(Au–Ag–Mo–Pb–Pt–Pd)	Operating (Cu–Au–Ag–Mo)
Intrusive plutonic	Twin Buttes (USA)	400	25–50 000	Cu–(Mo–Ag)	Closed
Intrusive plutonic	Palabora (South Africa)	3 521	56 000	(Cu–Ni–Fe–Ag–As–Bi–Au–Pb–Sb–Se–Te–Pt–Pd)	Operating (Cu–Ni–Fe–Ag–As–Bi–Au–Pb–Sb–Se–Te)

TABLE 10. URANIUM AS A PRIMARY OR CO-PRODUCT

Deposit type	Deposit name	Historical production (tU)	Geological Resources (tU)	Metals produced	Status
Intrusive plutonic	Poços de Caldas (Brazil)	1 055	22 700	U–Mo	Closed
Proterozoic unconformity	Cluff Lake district (Canada)	20 600	25 000	U–(Au)	Depleted
Collapse breccia pipe	Orphan Lode (USA)	1 656	2 050	Cu–Ag–U	Depleted
Sandstone	Uravan mill (USA)	67 300		U–V	Closed
Sandstone	White Mesa mill (USA)	7 348		U–V	Operating
Sandstone	Mexican Hat mill Moab (USA)	180	360	U–Cu	Closed
Sandstone	Akouta–Arlit district (Niger)	140 000	740 000	U–(Mo)	Operating (U–Mo)
Palaeo quartz-pebble conglomerate	Denison mine Elliot Lake district (Canada)	55 325	185 000	U–(Sc–Th)	Closed
Phosphate	Aktau Pricaspian region (Kazakhstan)	30 000	74 000	U–(Sc–REE)	Closed

TABLE 11. PROJECTS FOR THE PRODUCTION OF URANIUM AS A BY- OR CO-PRODUCT

Deposit type	Deposit name	Geological Resources (tU)	Metals to be recovered	Status (2017)
Intrusive plutonic	Ilimaussaq district (Greenland)	228 250	REE–U–Zn	Development
Volcanic-related	El Boleo (Mexico)	4 250	Cu–Zn–Ni–Mn–(U)	Operating (Cu–Zn–Ni–Mn)
Metamorphite	Lumwana (Zambia)	10 000	Cu–(U)	Operating (Cu)
Collapse breccia pipe	Canyon (USA)	937	U–Cu	Development
Sandstone	Falea (Mali)	17 430	U–Ag–Cu	Prefeasibility study
Sandstone-lignite	Mulga Rock (Australia)	28 710	U–(Cu–Zn–Ni–Co–Sc)	Development
Black shale	Talvivaara (Finland)	27 000	Ni–Cu–Zn–Co–(U)	Operating (Ni–Co–Zn–Cu)
Black shale	Häggån (Sweden)	308 000	U–Mo–V–Ni–Zn	Prefeasibility study
Black shale	Vicken MMS (Sweden)	447 755	U–Ni–Zn	Prefeasibility study
Black shale	Buckton Zone (Canada)	34 000	Mo–Ni–V–U–Zn–Cu–Co–REE–Li–Sc–Th	Prefeasibility study

4.1. INTRUSIVE PLUTONIC DEPOSITS

Intrusive plutonic deposits related to magmatic differentiation include large, low grade copper porphyries, such as the Bingham Canyon and Twin Buttes deposits in the USA, both of which have produced small quantities of uranium. Several large porphyry copper deposits are also located in Chile (Chuquicamata, Rodomiro Tomic), but even if some projects were initiated to recover uranium, no uranium was produced. The Ilimaussaq (Greenland) and Poços de Caldas (Brazil) peralkaline complexes and the Palabora carbonatite (South Africa) are also classed as intrusive plutonic.

4.1.1. Quartz monzonite: Bingham Canyon (USA)

Westinghouse Electric Corporation constructed the uranium recovery plant at the Bingham Canyon copper mine in 1977 and later expanded it twice for a total installed cost of over US $10 million.

Energy Fuels Nuclear (EFN) acquired the Bingham Canyon Ion Exchange Plant from Westinghouse in September 1986. At that time, the plant had been shut down for approximately one year. EFN operated the plant from 1987 through 1989.

The plant was designed to extract uranium values from solutions provided by Kennecott Copper Corporation as a part of its ongoing mine dump leaching programme. The uranium plant processed tailings solutions from the Kennecott precipitation plant after the copper values had been removed.

4.1.1.1. Geology

According to Dahlkamp [26], the Bingham Canyon copper deposit lies within a major epizonal intrusion consisting mainly of quartz monzonite porphyry. Hydrothermal alteration is present. Ore minerals are mainly sulphides and include molybdenite, bornite–chalcopyrite, chalcopyrite–pyrite, pyrite and galena–sphalerite. No discrete uranium mineral has been reported, but uranium may be present as uraninite or uranothorite. Uranium occurs mainly with copper and molybdenum ores at a reported grade of 0.002–0.005% U.

4.1.1.2. Licensing

EFN applied to the Bureau of Radiation Control in the State of Utah’s Division of Environmental Health for a radioactive material licence for the Bingham Canyon IX plant on 21 October 1986. The State of Utah became an agreement State in 1985 with the US Nuclear Regulatory Commission and as such was the responsible regulatory body which governed and exercised regulatory control over the processing and handling of radioactive material.

On 23 January 1987, the Bureau of Radiation Control issued Radioactive Material License No. UT-1800226 to EFN with respect to the Bingham Canyon operation. The thirty licence conditions listed in this publication describe the broad spectrum of process operation controls, environmental monitoring requirements, health physics requirements and analytical procedures, etc., which were to be maintained during normal process operations.

After acquiring the plant, EFN began completing needed maintenance items and general refurbishment of the plant in December 1986. The plant commenced operation in February 1987 and ran through early 1989, when operations were suspended due to low prevailing uranium prices.

4.1.1.3. Operations

EFN’s Bingham Canyon plant ran four Higgins Loop IX columns operating in parallel to process approximately 700 L/min of feed solution. Feed was sent to the IX columns at 26 C°, 1.02 centipoise viscosity and at a pH of 3.1. The uranium concentration was typically in the range of 0.00031–0.0041% U. The plant operated at a production rate of 50 tU/year.

Uranium anion complexes in solution will substitute for the sulphate ions attached to the ion exchange resin. The plant utilized strong base anion exchange (Dow MSA-1) recovery with 1M sulphuric acid strip. The strip solution averaged 0.0424% U. Additional concentration and purification was achieved by solvent extraction in a conventional mixer–settler circuit. The organic solvent used was a mixture of EPHA (di-(2-eythylhexyl) phosphoric acid) and TOPO (tri-n-octyl phosphine oxide) in a kerosene solvent. Uranium concentration in the organic phase was 0.4%. A 10% solution of sodium carbonate was used to strip uranium in three stages from the organics, at which point the uranium concentration was raised to 2.5%.

Barren raffinate was recycled back to IX elution to complete the process. Carbonate pregnant uranium solution was acidified with sulphuric acid and heated with steam to decompose the carbonate. Following precipitation with anhydrous ammonia, the yellowcake product was dried at 343°C to produce a final concentrate of ~90%.

The plant operated 24 h/d for 7 days per week and routinely recorded over 99.5% operating time in any one month. Plant operations required 14 personnel, including operators, maintenance, laboratory, radiation safety, clerical and plant management [27].

Although not in the original design, the plant also recovered scandium oxide in addition to uranium. The by-product scandium was contained in the residue which forms during the solvent extraction process. Scandium oxide was produced in the form of a dry powder containing approximately 3–5% Sc_2O_3. Scandium production was somewhat erratic and unpredictable.

4.1.1.4. Production data

The Bingham Canyon uranium recovery facility operated from 1978 through 1989, during which time it produced a total of approximately 420 tU. Historical annual uranium production is shown in Fig. 18.

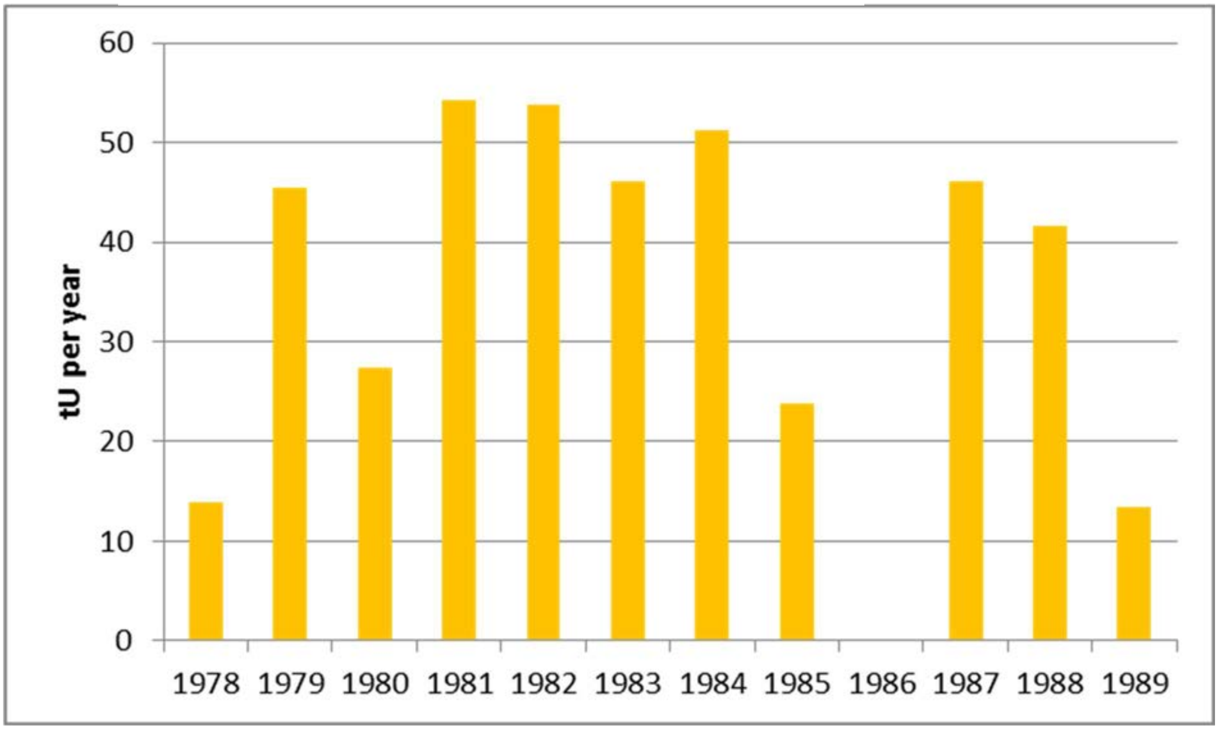

FIG. 18. Historical uranium production at Bingham Canyon (courtesy T. Pool).

4.1.2. Quartz monzonite: Twin Buttes (USA)

Anamax Mining, a 50:50 joint venture between Amax and Atlantic Richfield, operated the 10 000 t/d Twin Buttes copper mining and milling facility at Sahuarita, Arizona. In 1980, the company began recovering by-product uranium, which is present in the copper ore at a concentration of about 35 ppm [28]. Annual production capacity was 54 tU.

4.1.2.1. Geology

Twin Buttes is a large quartz monzonite body intruded into a folded and faulted complex of Palaeozoic limestone, siliceous limestone, impure argillaceous limestone and Cretaceous siltstone, arkose, tuffs and quartzites. Structural ore control occurred along the contacts of the dyke-like quartz monzonite porphyry with sedimentary strata. Mineralization consisted mainly of disseminated copper and molybdenum minerals with oxides in the upper levels of the orebody and sulphides in the lower part [29].

4.1.2.2. Operations

Twin Buttes was a large open pit mine 1500 m long, 1200 m wide and 250 m deep. Copper oxide ore was crushed and wet ground to liberate the copper. Ore pulp was acidified with sulphuric acid to dissolve copper and uranium. Counter-current decantation thickeners and a clarifier provided a clean solution to the compartmentalized fluidized moving resin bed IX columns. Uranium solution passed up the column as resin moved downwards from compartment to compartment. Loaded resin in the bottom compartment was transferred to a measuring chamber to provide a fixed volume for elution and to remove iron from the resin with weak acid. Iron free resin was transferred to an elution column where the uranium was eluted with 1M sulphuric acid. After elution, the resin was transferred back to the top of the load columns. Barren uranium solution, which still contained copper, was pumped to the copper solvent extraction plant.

The uranium solvent extraction plant consisted of four extraction stages, one acid scrub stage and three stripping stages. Organic solvent consisted of a mix of alamine 336 and isodeconal in kerosene. Uranium was stripped from the organic phase with ammonium sulphate solution at a pH of 4. Uranium in the strip solution was precipitated as yellowcake with the addition of ammonia to produce a final pH of 7. The precipitate was thickened, centrifuged, dried and packaged [30].

4.1.2.3. Production data

During the period 1980 through 1985, the Anamax facility produced a total of approximately 400 tU, as shown in Fig. 19. Detailed production statistics are given in Table 12.

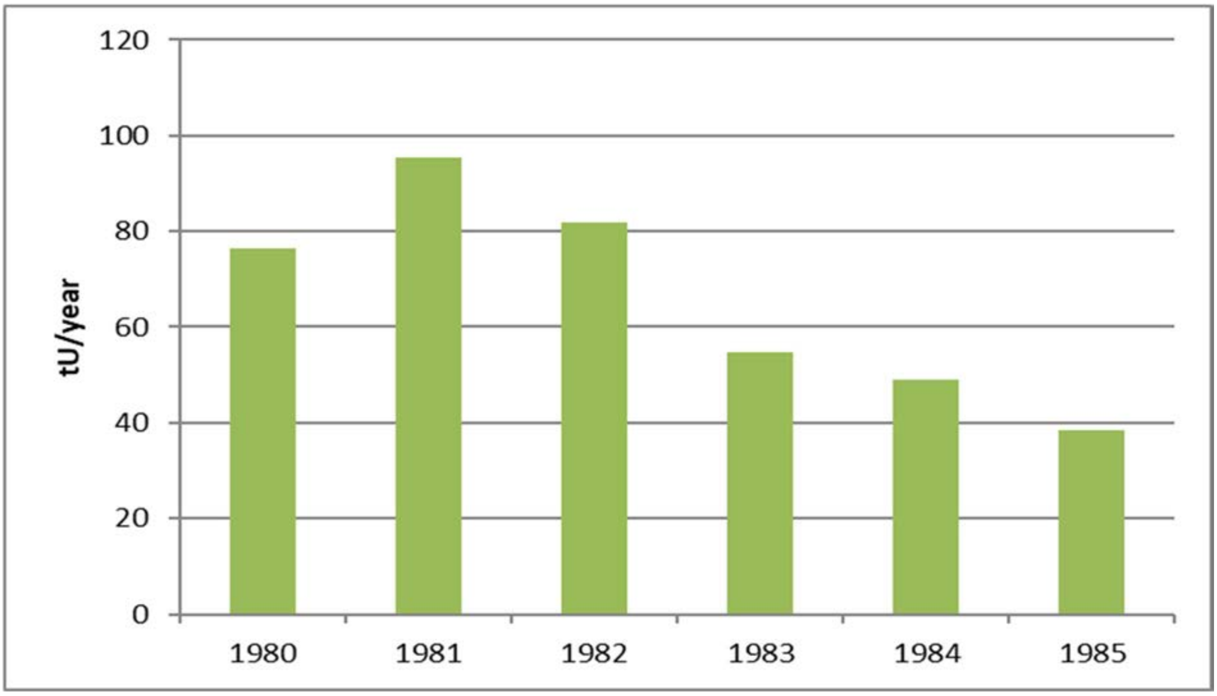

FIG. 19. Anamax historical uranium production (courtesy T. Pool).

TABLE 12. PRODUCTION STATISTICS: TWIN BUTTES (1981–1984)

Parameter	1981	1982	1983	1984
Oxide ore treated (short t)	3 743 116	3 739 052	3 585 629	3 513 490
Copper ore grade (%)	1.204	1.060	0.933	0.911
Copper recovery (%)	76.70	77.35	76.57	79.93
Copper produced (short t)	34 505	30 885	25 730	25 523
U produced (kg)	95 512	81 777	54 749	49 113
U recovered grade (%)	0.0028	0.0022	0.0017	0.0015
Nominal copper price ($/kg)[a]	1.85	1.61	1.72	1.48
Copper value (US$ million)	56.9	44.3	39.5	33.6
Nominal U_3O_8 price ($/kg)	45.22	37.2	42.95	32.28
U_3O_8 value (US$ million)	5.08	3.56	2.79	1.86

[a] all dollar values are given for that particular year

4.1.3. Quartz monzonite: Chilean copper

World copper production primarily uses two major process flowsheets: (i) concentrator/smelter/electro-deposition and (ii) heap leach/solvent extraction/electro-winning (Fig. 20). There are a variety of flowsheet alternatives, particularly when dealing with complex ores containing other metal products, most notably nickel, precious metals and base metals. About 20% of the world's copper production uses heap leach with sulphuric acid as the lixiviate, which has the potential for also dissolving uranium during the leach cycle.

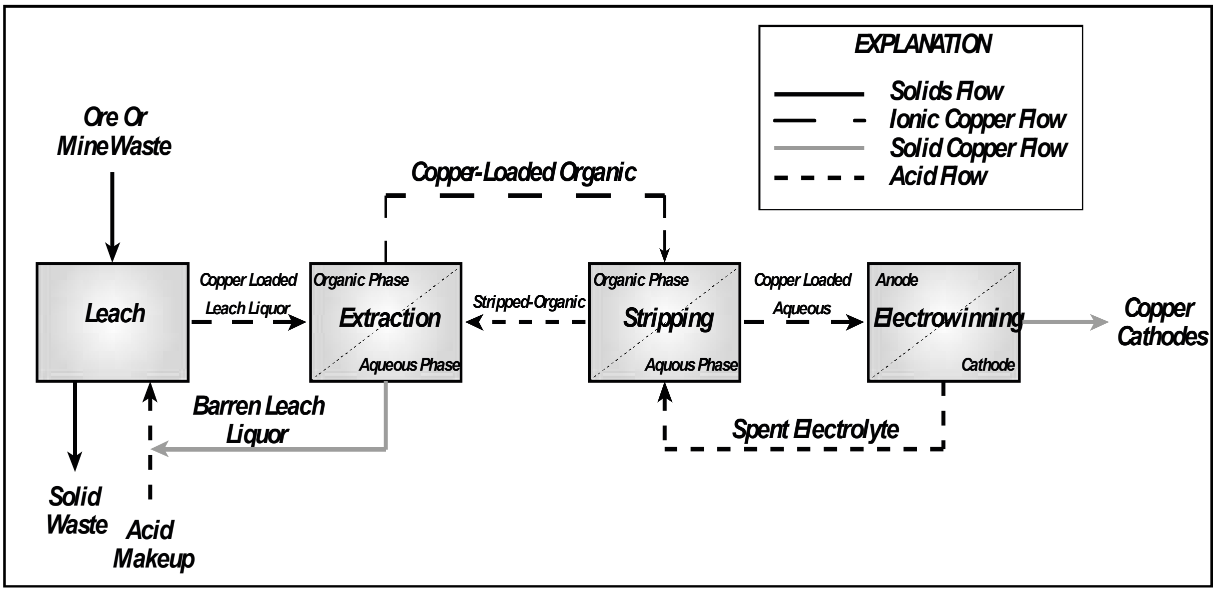

FIG. 20. Simplified flow chart: leach — SX — electrowinning process (adapted from Ref. [30]).

In Chile, copper production is primarily from ores containing copper with very small levels of other metals. Chile produces 30–35% of the world's copper, as shown in Fig. 21. Chilean mines produce about 5.8 million t of copper annually from about 10 billion t of ore extracted. Typically, about 20% of this copper production is with heap leach/SX/electrowinning processes. Assuming a uranium content in the copper ore of 3–5 ppm, Chile could recover an estimated 1000–2000 tU/year simply by extracting uranium from heap leach solvent extraction solutions.

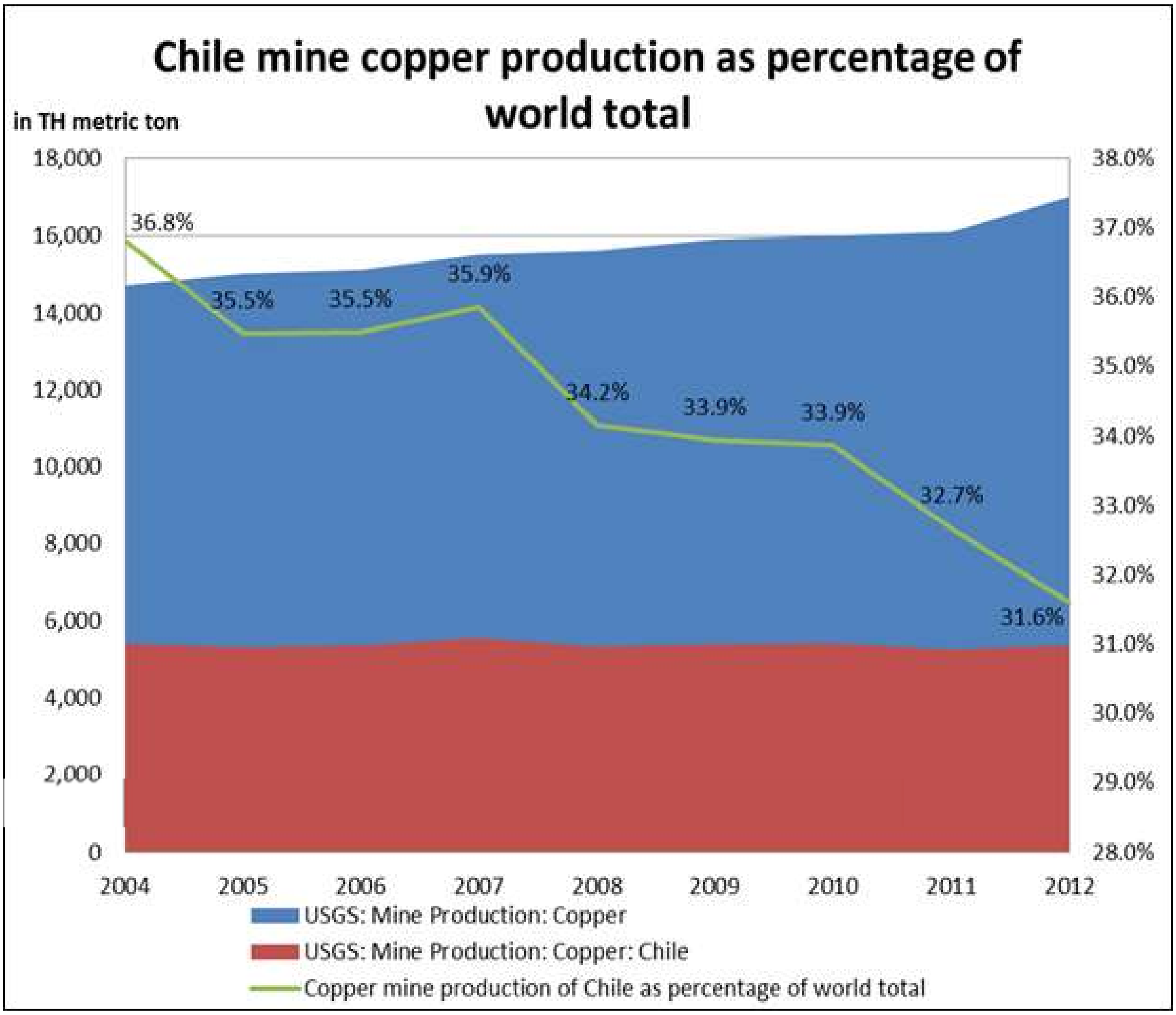

FIG. 21. Chile copper production (t Cu) as a percentage of world total (source of data: US Geological Survey, non-ferrous metal production).

4.1.4. Peralkaline complex: Ilimaussaq (Greenland)

4.1.4.1. History

The Ilimaussaq intrusive complex is situated in southern Greenland, about 8 km from the port of Narsauaq (Fig. 22). The complex contains the Kvanefjeld deposit which was discovered in 1956 during a regional exploration programme. In the early 1970s, aerial and ground prospecting were carried out, followed by a diamond drilling campaign in 1977. Based on this, a resource of 43 000t U at an average grade of 0.034% U was identified. Further exploration up to 1986 resulted in the identification of geological resources of the order of 60 000t U [31].

In 1979–1980, a 960-m long adit was completed and 4000 t of ore extracted for mining and metallurgical testing. In the mid-1980s, it was concluded that uranium recovery was not economic, mainly due to the low grade and the refractory nature of the ore.

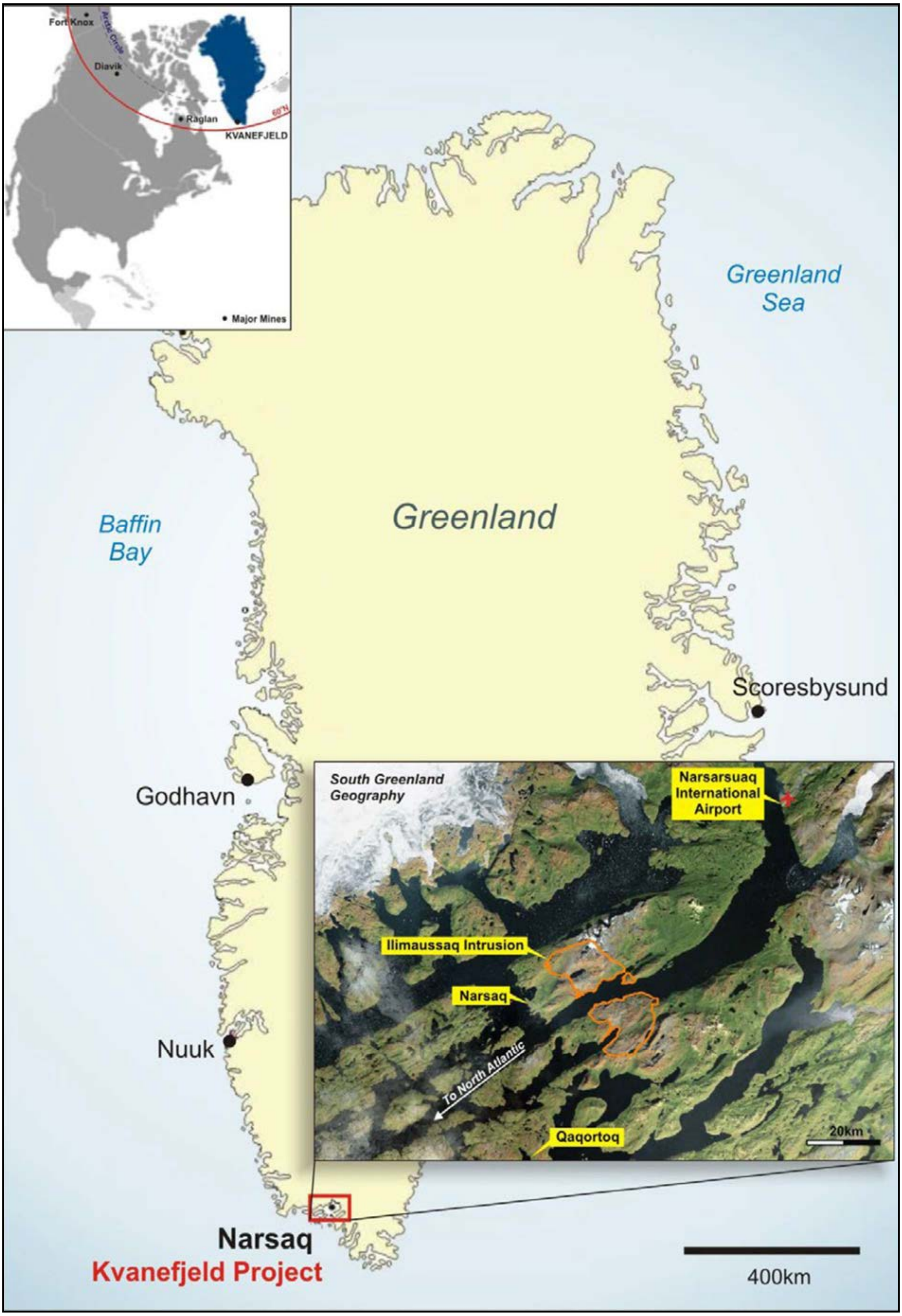

FIG. 22. Location of the Kvanefjeld project (Greenland). Reproduced with permsission [31].

The project has been operated since 2007 by Greenland Minerals and Energy Ltd (GMEL), an Australian based company. Exploration was initially focused on non-radioactive minerals, in particular the REE, as exploration for and exploitation of uranium and thorium were prohibited. In 2011, the company published resources of 141 610 tU at an average grade of 220 ppm U, 6.6 million t REE at a grade of 1.07% and 3 million t zinc at a grade of 0.22% Zn for the Kvanefjeld deposit. In addition, yttrium, fluorine and zirconium were considered anomalous. A higher grade zone near the surface with 49 300 tU at a grade of 345 ppm U and 1.7 million t REE at a grade of 1.4% was also defined which could improve mine scheduling. The Kvanefjeld project was recognized as the world's largest undeveloped compliant resource of rare earth oxides in a multi-element deposit [32].

In 2012, two new mineralized zones were recognized in the Ilimaussaq complex with resources of 62 370 tU at a grade of 258 ppm U for Zone 2 (known as Sørensen) with 1.10% REE and 0.26% Zn and 24 250 tU at a grade of 254 ppm U for Zone 3 with 1.16% REE.

In November 2012, the position of Greenland with regard to uranium policy was addressed in Parliament and in January 2013 a majority voted to allow extraction and exports of uranium from Greenland, marking a historical shift in Danish foreign policy after 30 years of opposition to nuclear power.

As of 2015, the project overall resource inventory stood at 228 240 tU, 10.33 million t REE and 2.25 million t Zn [31]. A feasibility study was completed with planned production of 1100 tU/year starting in 2016–2017, with a projected mine life of at least 30 years.

In September 2016, GMEL announced that Shenghe Resources Holding Ltd had acquired a 12.5% interest in the project, which is not now scheduled to start production prior to 2019–2020.

4.1.4.2. Geology

The Ilimaussaq plutonic intrusive peralkaline complex covers an area of about 156 km² (Fig. 23). The mineralized occurrences are connected both spatially and temporally to the alkaline igneous formations of the Gardar intracratonic rift province. Magmatism in the Gardar Province is related to two main rifting events dated at 1280 Ma and 1180–1140 Ma.

The complex hosts one of the most unusual examples of magmatic differentiation known. The Ilimaussaq intrusion comprises an early stage augite syenite preserved along the margins, followed by a suite of peralkaline nepheline syenites composed of pulaskite grading through foyaite, naujaite, kakortokite and, finally, to lujavrite, which is the last phase and which contains most of the mineralization. Lujavrites are extremely enriched in incompatible elements such as REE, lithium, beryllium and uranium.

Within the Ilimaussaq complex, the lujavrite series is at least 500 m thick and comprises fine-grained and layered rocks with locally some medium- to coarse-grained pegmatitic varieties. Black (arfvedsonite-bearing) lujavrite is the rock type that hosts the REE, uranium and zinc [31].

In the Kvanefjeld area, the general appearance is one of a huge intrusive breccia. The lujavrites intruded and deformed the overlying, older intrusive rocks and continental sandstones and lavas forming the roof of the intrusion. The mode of occurrence of the lujavrites ranges from small veinlets to continuous belts several hundreds of metres in width. The highest and most extensive concentrations of uranium (>300 ppm) are found close to, and within, contact zones between sheets of lujavrites and altered volcanic cover rocks [31].

The uranium and thorium contents increase from 10 ppm U and 20 ppm Th in the pulaskites to more than 60 ppm U and 60 ppm Th in the lujavrite. In some lujavrite varieties, the uranium concentration may attain 1000 ppm and the thorium up to 5000 ppm.

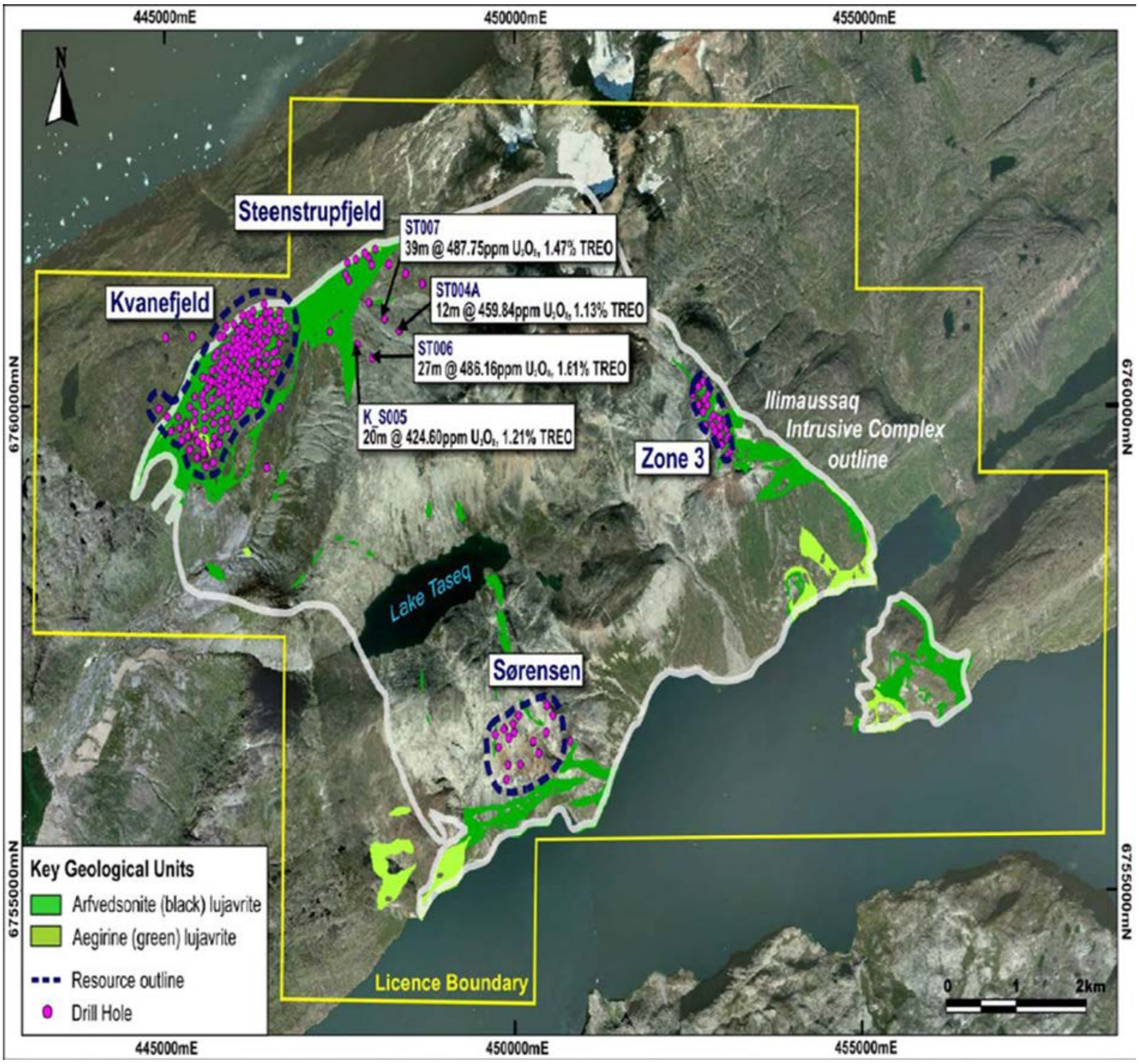

FIG. 23. Overview of the northern Ilimaussaq complex showing the location of Kvanefjeld, Sørensen and Zone 3 deposit. Reproduced with permission [31].

Steenstrupine, a complex sodic phospho-silicate mineral, is the main host to both REE and uranium in the lujavrite hosted deposits and is disseminated in the rock mainly as small interstitial crystals. Steenstrupine contains >15% total rare earth oxide. The uranium and thorium contents are 0.2–1.5% and 0.2–7.5%, respectively, and the whole rock radioelement contents vary in the range 0.01–0.30% U and 0.03–1.50% Th [31].

Other minerals that are important hosts to REE include the phosphate mineral vitusite and, to a lesser extent, britholite, lovozerite group minerals and, rarely, monazite. Aside from steenstrupine, uranium is also hosted in zirconium silicate minerals of the lovozerite group. In these silicates, a portion of the zirconium is substituted by several hundred parts per million each of uranium, yttrium, REE and tin. Zinc occurs as sphalerite, which is the dominant sulphide, disseminated throughout the deposits [31].

In the nepheline syenite, accessory silicate minerals of unusual composition such as eudialyte and rinkite are also present. The suite of rocks is characterized by the abundance of volatile phases (fluorine and chlorine) as well as by a number of incompatible elements, such as REE, Nb, Be, Li, Zn, Zr and Sn together with the radioactive elements U and Th.

The highest concentrations of radioactive elements are found in the minerals rinkite (fluorosilicate with Ca, Na, Ce, Ti, Zr), which has 0.3–1.2% U and 0.3–0.4% Th, and steenstrupine (phosphatosilicate with Na, Ce, Mn, Nb, Fe), which has 0.2–1.5% U and 0.2–7.4% Th. Other important minerals are britholite, eudialyte, monazite, pyrochlore and thorite. Steenstrupine is the main uranium mineral occurring in the medium- to coarse-grained lujavrite, where it occurs instead of eudialyte, which is the principal uranium–thorium mineral present in the other rocks of the complex [31].

4.1.4.3. Feasibility study

GMEL released a feasibility study in 2015. The study 'Base Case' evaluates the development of a mine, mineral concentrator, refinery and supporting infrastructure located in Greenland and treating 3.0 million t/year of ore. The initial mine life is estimated at 37 years with resources large enough to support multiple future expansions [31].

The Kvanefjeld deposit will be mined as a large open pit to a depth up to 300 m. The orebody crops out at the surface, with the higher grade portions occurring close to the surface.

The project's primary product will be a concentrate of mixed critical rare earth oxides. Critical rare earths are defined by GMEL as those considered to be particularly important for so-called 'green' technologies, which are forecast to be in short supply over time (neodymium, praseodymium, europium, dysprosium, terbium and yttrium). Kvanefjeld will also produce uranium oxide, lanthanum and cerium products, zinc concentrate and fluorspar. GMEL states that the project economics are relatively insensitive to the pricing of the by-products [31].

A key strength of the project is its attractive metallurgy: the unique REE- and uranium-bearing minerals can be concentrated into less than 10% of the original ore mass utilizing froth flotation. The minerals can be effectively treated using an atmospheric sulphuric acid leach. There is no requirement for complex mineral 'cracking'. This technique has effectively rendered the steenstrupine dominated Kvanefjeld REE–U mineralization non-refractory, improving recoveries and lowering capital expenditure and operating expenditure relative to previous methods. The process flowsheet has been rigorously developed by GMEL and has been the subject of extensive test work, including three pilot plant campaigns [31].

The flowsheet (crush and mill, beneficiation by flotation, hydrometallurgical leaching at atmospheric pressure and solvent extraction) is simple compared to the flowsheets of most other REE projects. End products will be heavy REE hydroxide (4200 t total rare earth oxide), mixed REE carbonate (10 400 t total rare earth oxide), light REE carbonate (26 200 t total rare earth oxide), uranium oxide (1000 tU/year) and a zinc sulphide concentrate (Figs 24–26).

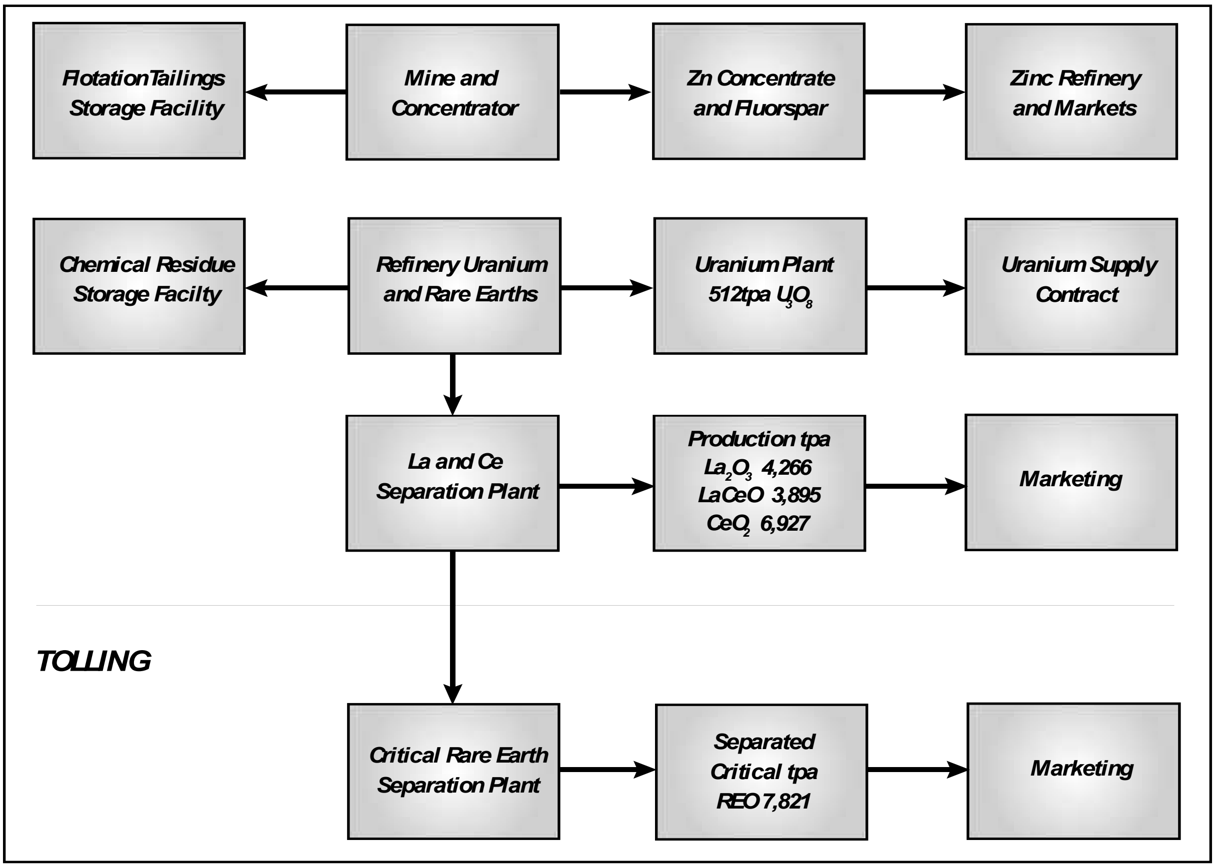

FIG. 24. Selected flowsheet and configuration for the Kvanefjeld feasibility study (adapted from Ref. [31]).

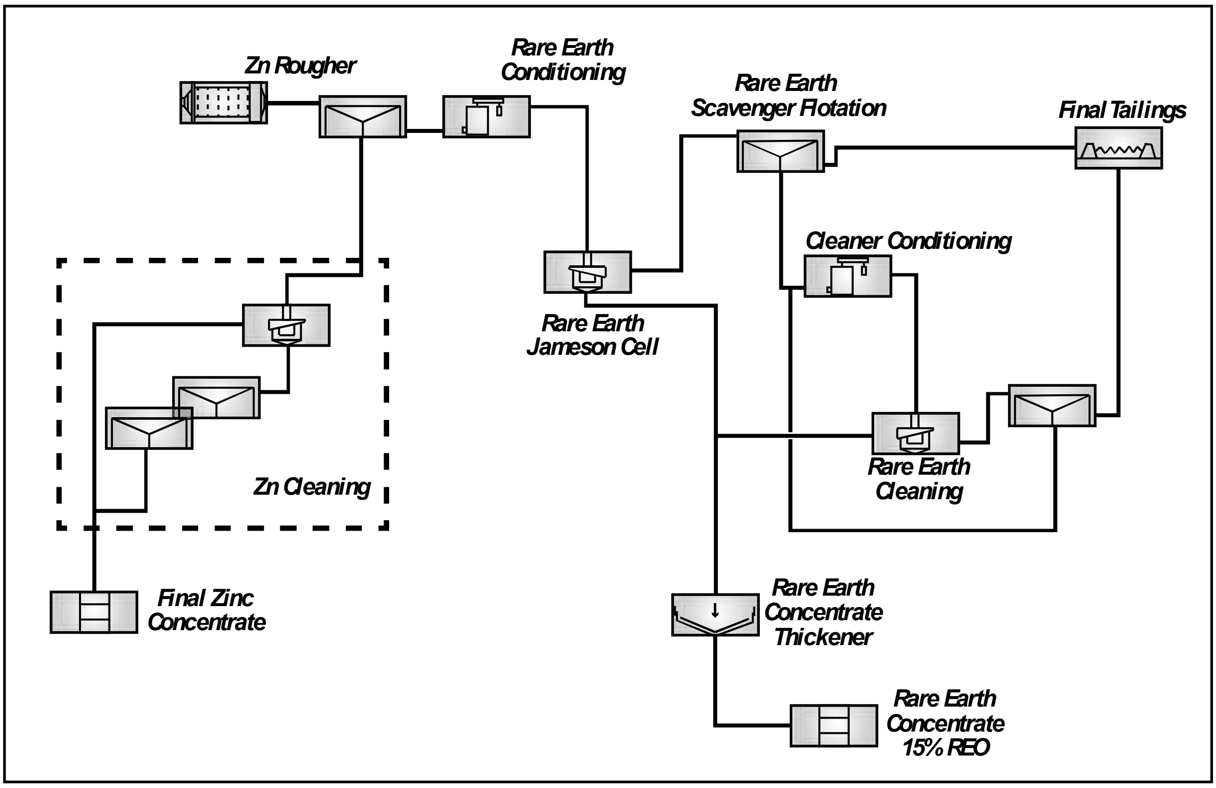

FIG. 25. Metallurgical diagram of concentrator circuit (adapted from Ref. [31]).

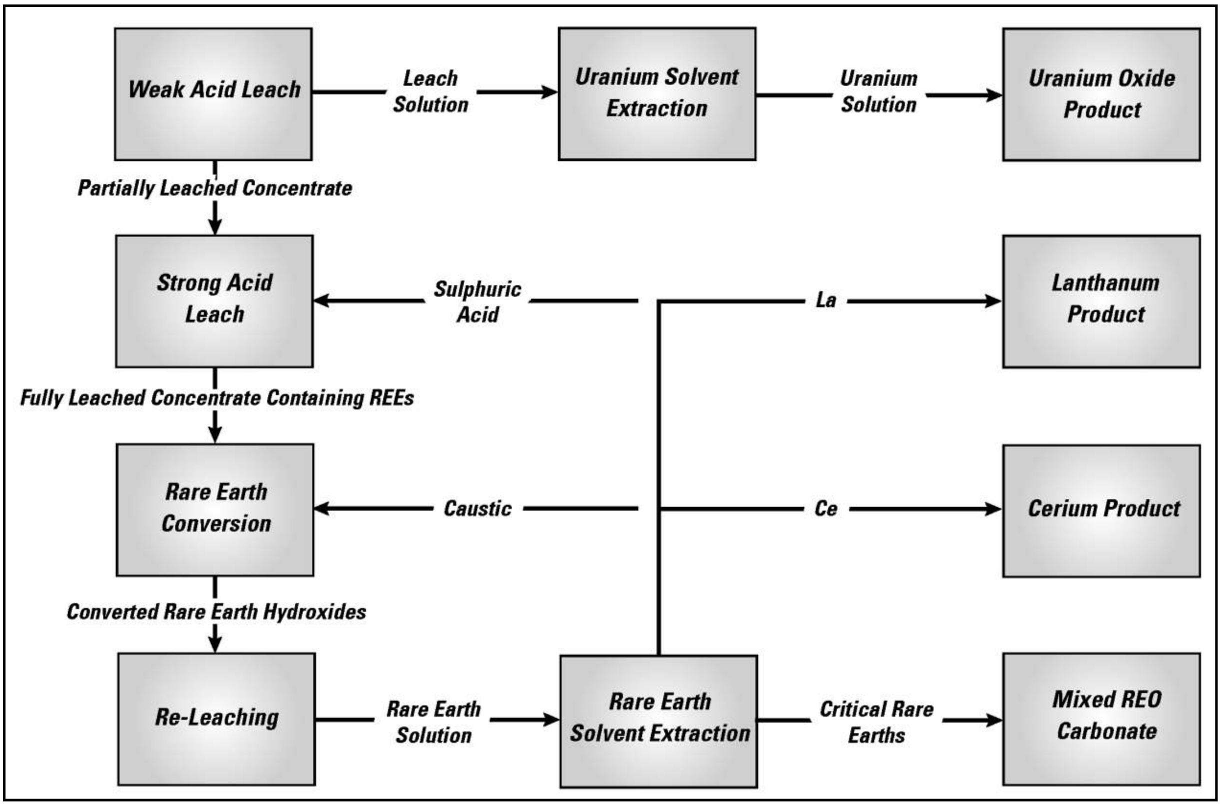

FIG. 26. Metallurgical design of the refinery circuit (adapted from Ref. [31]).

4.1.5. Peralkaline complex: Poços de Caldas (Brazil)

4.1.5.1. History

Brazil's uranium industry began in 1951 with the creation of the Conselho Nacional de Pesquisas, which started prospecting for uranium in cooperation with the US Geological Survey and the US Atomic Energy Commission. The Comissão Nacional de Energia Nuclear (CNEN) was formed in 1956 and initiated a uranium reconnaissance programme. Between 1961 and 1970, CNEN organized and carried out an intensive campaign of prospecting which resulted in the discovery of uranium deposits in the Poços de Caldas area.

In 1970, CNEN began detailed drilling and underground exploration of a deposit at Campo de Cercado, about 30 km from Poços de Caldas. NUCLEBRAS was formed in 1974 to oversee Brazil's nuclear programme and, in 1975, decided to exploit the Cercado deposit.

A contract was signed with Uranium Pechiney Ugine Kuhlmann in 1976 to design the mine and mill. Mine stripping began in 1977 and the mill was completed in 1981. The first production of uranium in Brazil, at the Osamu Utsumi mine (Poços de Caldas deposit), started in 1982. Production reached a maximum of only about 50% capacity (242 tU in 1982) in the first few years and much less in later years. Owing to escalating costs and reduced demand, this activity was put on stand-by between 1990 and 1992. The mine was restarted in 1993, but operations stopped again in October 1995. The cumulative production of the mine to 1996 was 1241 tU [33].

4.1.5.2. Geology

The Poços de Caldas peralkaline intrusive complex is located on the border between the states of Minas Gerais and São Paulo, approximately 350 km from Rio de Janeiro [34].

The topographical expression of the complex is that of a plateau, but the presence of ring dykes gives it the appearance of a caldera. The interior plateau has a mean elevation of 1300 m, the rim rises to 1600 m and the surrounding terrain to 950 m. The areal extent of the complex is about 850 km^2.

The country rock into which the complex is intruded is gneiss of Upper Proterozoic age. Pre-intrusive, Lower Cretaceous sediments, which are correlated with the Botucatu Formation (in the Paraná Basin), form thin, scattered remnants within the complex and exhibit alteration effects caused by the intrusion. Thus, the maximum age of the complex is Lower Cretaceous (79 Ma) for the main nepheline syenite. Both Precambrian gneiss and Cretaceous sediments are cut by diabase dykes.

Post-Cretaceous faulting has also disturbed the complex. The effects are expressed geomorphologically and the southeast part of the plateau has greater topographic relief than the other areas. In the north, erosional planation effects are visible. The southeast quadrant is set apart from the rest by the intersection of two major faults. The uranium–molybdenum deposits of Agostinho and Cercada are located where these lineaments intersect.

4.1.5.3. Mineralization

The primary uranium mineralization episode is believed to date from 80 Ma. Uranium is associated to fluorite, molybdenum, zirconium and thorium.

Uranium in the complex is always associated with zirconium. The major zirconium mineral is caldasite, a local term used to describe high grade ore, essentially a crystalline aggregate of baddeleyite (ZrO_2) and zircon ($ZrSiO_4$), together with minor uranium. There are over 30 caldasite mines and numerous small occurrences. Uranium grade is related to the zirconium content in some of these deposits, but in others this relationship is absent. Zirconium ranges from 70 to 95%, alluvial pebbles having the highest grade. Shipped ore is a mix of alluvial, eluvial and vein-type caldasite. All the caldasite veins are found in highly altered rocks.

Taquari Hill (280 m) is located on the eastern side of the plateau, near the town of Poços de Caldas, and is a conspicuous topographic feature. All three types of caldasite (alluvial, eluvial and vein) are present at Taquari Hill. Not only is the uranium found in the caldasite, but it is also disseminated in the foyaite country rock, which has undergone significant alteration. Pyrite, fluorite veins and large crystals of violet fluorite occur where alteration is greatest. The zirconium content of foyaite averages 1–5% ZrO_2, while the uranium content is generally in the range of 0.2–0.3% U. At depth there is also molybdenum. Mineralization extends to depths of 200 m without a significant difference in the alteration properties.

There appears to be a natural separation of uranium and thorium. Uranium is associated with zirconium and thorium is associated with oxides of iron and manganese. Thorium is concentrated in the centre of the complex, where the most important deposit is at Iron Hill (Morro do Ferro), which is considered to be one of the world's largest reserves of thorium (some 35 million t), in addition to 30 million t of REEs.

4.1.5.4. Resources

In the Cercado area of the complex, five uranium orebodies were recognized at an early stage. After 190 000 m of drilling and 2500 m of underground development at three different levels, the deposit was determined as consisting of three main orebodies, designated A, B and E. The three orebodies are discrete and separated from one another by barren tinguaite breccia.

Orebody A measures 800 m (N 30° W) × 520 m (N 60° E). Within orebody A, there are subunits in tinguaite breccia zones which are highly mineralized. The mineral association is uranium, pyrite, fluorite and molybdenum. The central part of orebody A is cut by breccias which are also mineralized. Orebody A comprises approximately 20% of the total reserves.

Orebody B body measures 1240 m (N 24° E) × 400 m (N 48° W) and is considered to be an accumulation of pyroclastics within a depression in the complex. The primary mineralization is associated with the contact between foyaite and tinguaite. There is a 50-m layer of brecciated tinguaite overlying the foyaite in a so-called ‘pipe’. As the contact with foyaite is approached, the uranium content increases but the zirconium level remains unchanged. Orebody B comprises alternating tuffs, lavas, ash and breccias, and intrusives of various sizes and morphology transect this pyroclastic body, giving credence to the hypothesis that the principal geological control is the contact between the intrusives and the pyroclastics. Secondary uranium mineralization is associated with surface oxidation. Pyrite is much less evident in orebody B body than in A, and there is almost no molybdenum. Fluorite has been reported. Orebody B accounts for 65% of the reserves [35].

Orebody E measures 1040 m (N 54° E) × 410 m (N 36° W). Mineralization in orebody E is secondary and is related to oxidation and alteration fronts. The reduced zone comprises an orebody approximately 20 m thick, consisting mainly of pitchblende. Orebody E accounts for 15% of the reserves.

The redox front at Cercado appears to follow the laterization front. As the laterization front extends deeper, so too does the redox zone. The brecciated condition of the tinguaites has facilitated the laterization process. Hydrothermal activity may have played an important role in the total mineralization process.

Seven different types of ore containing uranium are present at Poços de Caldas, with values ranging up to 0.848% U and accompanied by fluorite, molybdenum, zirconium and thorium. Available resources at closure were estimated to be of the order of 22 700 tU at an average grade of 0.07% U.

4.1.5.5. Mining

The Osamu Utsumi open pit mine is located on the Poços de Caldos plateau in the southern part of Minas Gerais. Consulting services to develop the mine and mill complex were provided by Uranium Pechiney Ugine Kuhlmann. After the drilling programme and engineering studies were completed (over a period of 10 years), stripping of overburden began in 1977. The first yellowcake (U_3O_8) production started in late 1981 [36].

The planned depth of the open pit was 350 m, requiring the excavation of 85 million m^3 of overburden, waste and ore. During the period 1977–1981, a total of 31 million m^3 of overburden were removed, of which 7 million m^3 required blasting. Equipment used during this period consisted of a conventional open pit fleet, comprising bulldozers, front-end loaders, bottom dump trucks, motor graders, scrapers, end-dump trucks, blasthole drills and water trucks.

Production of ore began in 1981 and over an 18-month period in 1981–1982 a total of 5.43 million m^3 of waste was mined, along with 4.44 million t of mineralized material consisting of 530 000 t of mill ore, 270 000 t of low grade ore for heap leaching and 3.64 million t of mine sorted waste. Production of ore for the mill reached its capacity of 2500 t/d in 1982.

The distribution of mine personnel was reported as: geology (85), engineering (13) and operations (649).

4.1.5.6. Milling

Run-of-mine ore ranging from 0.05% to 0.10% U was crushed in a primary crusher and stockpiled in an area having a storage capacity of 300 000 t. The various ores were stockpiled separately according to their respective grades. On the basis of careful sampling and computer control, ores were blended and further crushed in two stages to produce 2500 t/d as feed to the grinding circuit. Low grade ores were heap leached.

Crushed ore was ground in two stages; pyrolusite was added as an oxidant to the second stage grinding circuit. The product was thickened to 55% solids and stored in tanks. Slurry was pumped to the chemical treatment plant located approximately 2 km from the crushing and grinding plant. Slurry was leached with sulphuric acid at 70°C in four parallel lines; it was then filtered and the resulting liquors containing uranium and molybdenum were oxidized with sodium chlorate. The uranium content of the liquors was 0.3 g/L U.

Uranium and molybdenum were co-extracted by solvent extraction in a mixer–settler system using Alamine 336 as the organic extractant. The organic loading was approximately 4.2 g/L U. Uranium was stripped from the organic phase with an acidic sodium chloride solution and a sodium carbonate solution was used subsequently to strip molybdenum.

Uranium in the acid phase was precipitated with ammonia to form yellowcake (ammonium diuranate), which was washed in ammonium sulphate, thickened, filtered and dried for shipment.

Molybdenum was precipitated from the carbonate strip solution in the form of calcium molybdate. Calcium hydroxide was used as the precipitation agent. Calcium molybdate precipitate was thickened, filtered and dried for shipment.

The nominal annual production capacity of the plant was 550 t yellowcake averaging 75% U. Production capacity and output for molybdenum are unknown. Tailings and liquid wastes were neutralized with lime and deposited in a tailings pond. Overflow from the tailings pond was treated with barium chloride to remove radium. Figure 27 illustrates the flow diagram of the Poços de Caldas processing plant.

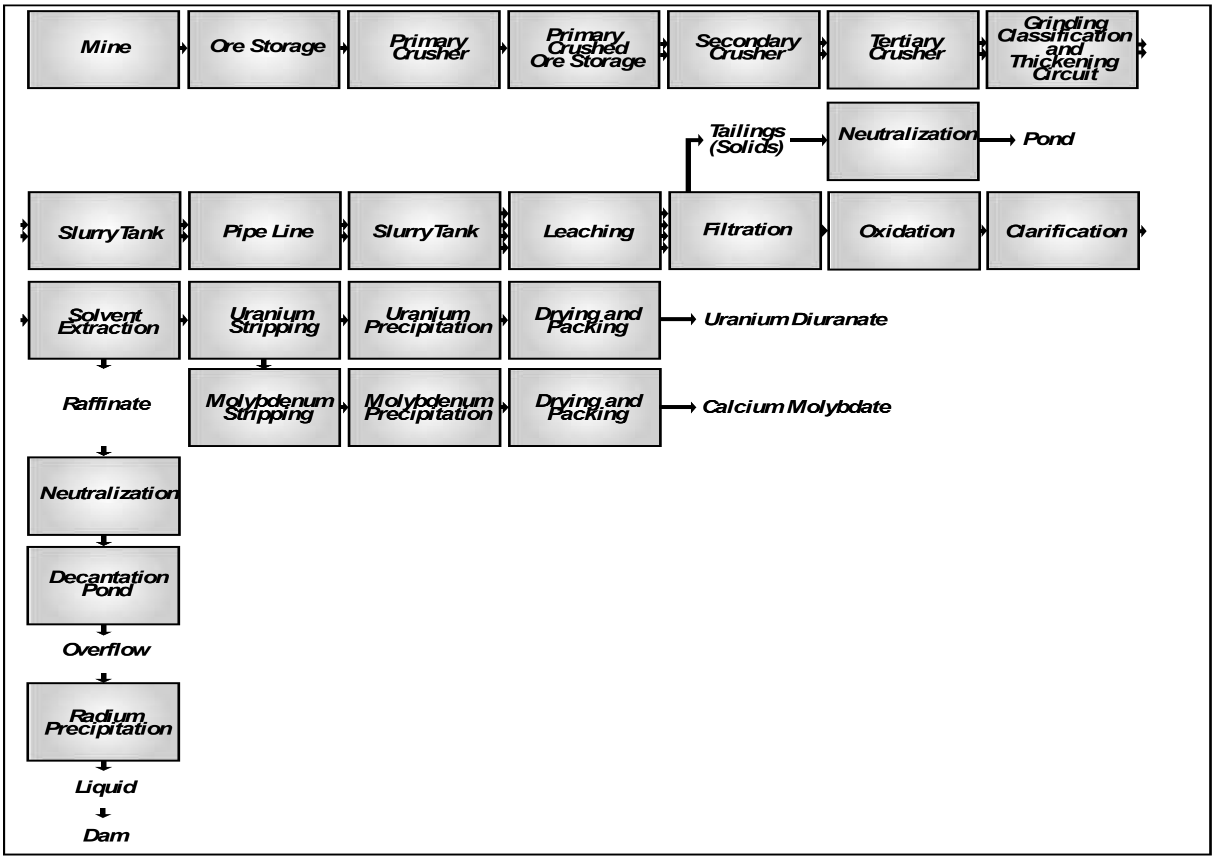

FIG. 27. Poços de Caldas uranium concentrate plant flowsheet (adapted from Source: Brazil — Poços de Caldas Industrial Complex, pp 169-172).

4.1.5.7. Production data

As illustrated in Fig. 28, uranium production at Poços de Caldas commenced in 1981 and ceased in 1995. Maximum annual output was 242 tU in 1982. Total output for the period of operation was 1241 tU.

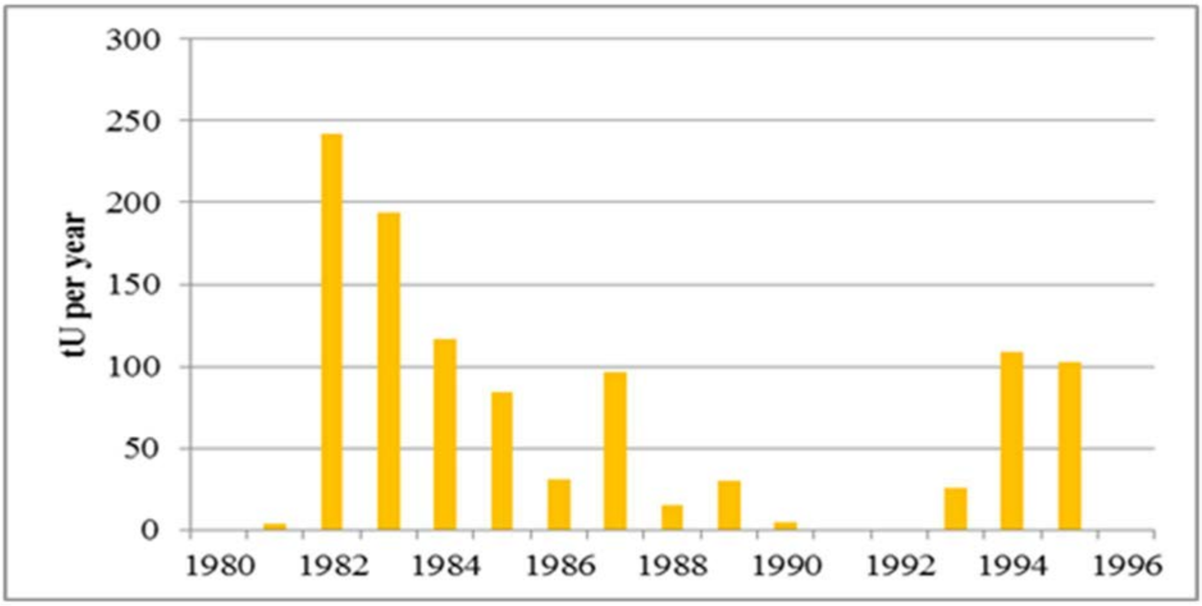

FIG. 28. Poços de Caldas historical uranium production (courtesy T. Pool).

4.1.6. Carbonatite: Palabora (South Africa)

4.1.6.1. History

Formed in 1956, Palabora Mining Company Ltd (Palabora) was originally a joint venture which included Rio Tinto Zinc Ltd of London (now Rio Tinto) and Newmont Mining Corporation of New York. The two companies' interests amounted to 38.9% and 28.6%, respectively, as of 1989 [37], although these equity stakes changed in subsequent years, when Rio Tinto assumed a 57.7% stake and Anglo American held 16.8% of the equity. On 5 September 2012, the two companies announced their intention to sell their respective interests in Palabora. On 11 December 2012, Rio Tinto announced that it had reached a binding sales agreement with a consortium which is committed to the ongoing operation of Palabora. The sales agreement was concluded in July 2013 and the company's name changed from Palabora Mining Company to Palabora Copper (Pty) Limited. The new shareholders are PMC (74%) and Empowerment Partners (26%).

The Palabora deposit, which is located in the Letaba district of north-eastern Transvaal, is an intrusive magmatic deposit emplaced in a carbonatite and foskorite host rock that forms the core of the much larger Palabora Igneous Complex. The carbonatite core has proven more resistant to erosion than the surrounding rocks and has thus formed a prominent hill, Loolekop, which became a landmark to early travellers through the region. Copper mineralization was reported on Loolekop around 1870 by the first European settlers. They described strong showings of copper mineralization throughout the highland area, plus evidence of primitive excavations where ancient people mined the ore and recovered the copper in small clay smelting pots. Archaeologists estimate that early mining began some 1200 years ago and continued until the late 1800s.

In 1912, Hans Merensky initiated the first detailed geological assessment of the Palabora Igneous Complex. In addition to copper, he noted the presence of large reserves of vermiculite and phosphate minerals (apatite). A small company, South African Phosphate Ltd, started mining phosphates in 1932

in a valley below Loolekop, but the operation failed financially after extraction of only 4500 t. In 1939, Merensky formed the Transvaal Ore Company for the purpose of exploiting the known vermiculite occurrence. The Phosphate Development Corporation (Foskor) began large volume mining of phosphates in the early 1950s.
Members of the geological unit of the South African Atomic Energy Board established the presence of uranothorianite in association with copper mineralization in 1952. The Mineral Development Section of the Atomic Energy Board evaluated the economic potential of uranium during the period 1953–1956. After considerable surface drilling and underground exploration drifting, the uranium content of the deposit was demonstrated to be well below economic levels of recovery and attention shifted back to the potential value of copper.

Palabora gained control of a key portion of the carbonatite core zone when the joint venture with the then Rio Tinto Zinc and Newmont Mining Corporation was formed in 1956, and detailed evaluation of the copper potential began later that year. Palabora operated the project and the exploration team drilled 111 inclined holes from the surface to an average depth of 370 m, a total of 41 000 m. Over 7000 m of horizontal drilling was completed in association with underground exploration drifting.

The underground work enabled geologists to determine the continuity of mineralization and to collect bulk samples of ore for metallurgical testing. Three inclined drill holes encountered almost continuous copper mineralization to a vertical depth of 1000 m. Drilling results indicated the presence of an estimated 286 million t of copper ore averaging 0.68% Cu. Several other minerals occur within the copper-bearing zone, including extremely low grade uranium.

In the early years of the project, Palabora concentrated on the evaluation of the main ore deposits as a source of copper. Following several years of property evaluation, Palabora determined that recovery of copper by large volume open pit mining would be economic and that recovery of at least four by-product minerals would improve project economics substantially [37]. Potential by-products included phosphate, magnetite, zirconia, vermiculite, precious metals, sulphuric acid and uranium. Actual mining began in 1964. Uranium was first produced in 1971. Uranium production was suspended in 2001 when the operation shifted to a smaller tonnage underground mine. Total uranium production from 1971 through 2001 amounted to 3521 tU.

4.1.6.2. Geology

The Palabora Igneous Complex was intruded sequentially into Archaean granite during the Lower Proterozoic. Overall, the complex influenced an area extending about 8 km in a N–S direction and 3 km E–W. There are 14 different rock types associated with the complex, of which the youngest intrusion is the transgressive carbonatite core that constitutes a major part of the Palabora ore deposit. The carbonatite core has been dated at approximately 2030 Ma [38].

The intrusions contain a series of rock forming minerals high in magnesium, iron, sodium, phosphate and carbonate. The transgressive carbonatite core intruded into a banded carbonatite segment; the younger rock is distinguished by its lighter colour, finer grain, lack of foliation and banding, and its high content of copper sulphides and accessory minerals. Both the transgressive and the banded carbonatite intrusions consist primarily of calcite and dolomite. The carbonatite intrusions are surrounded in turn by a vertical volcanic pipe of foskorite, a rock composed of olivine/serpentine, magnetite, apatite and phlogopite [39]. Figure 29 presents a generalized geological map of the Palabora Igneous Complex.

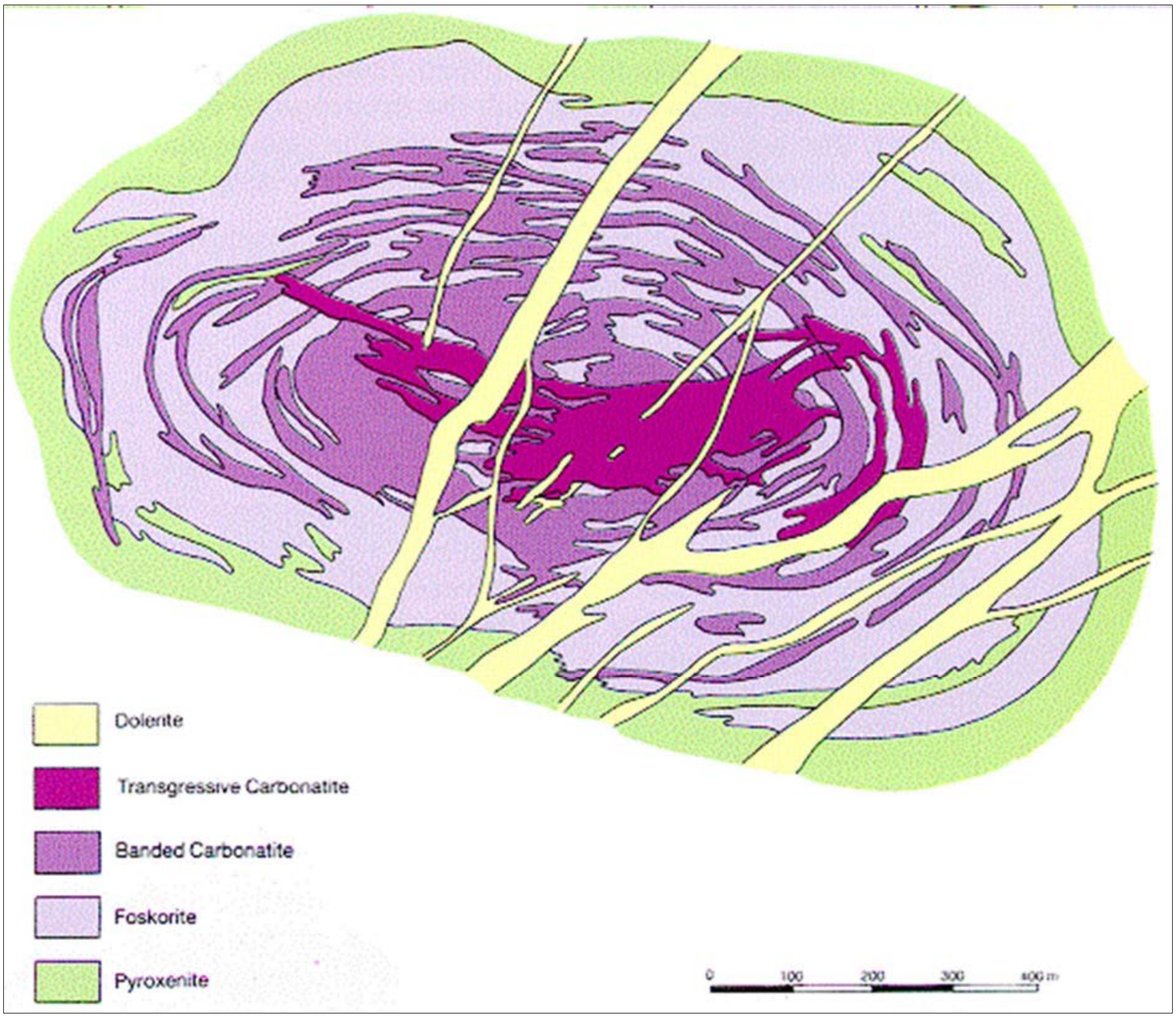

FIG. 29. Generalized geological map of the Palabora Igneous Complex. Reproduced under Creative Commons licence from [40].

It is clear from the position of the transgressive carbonatite core that the younger rock sequences of the Palabora Igneous Complex were intruded en masse or as radiating dyke swarms along lines of weakness developed during each preceding intrusion. The position of the transgressive core is controlled by two pre-existing fracture systems at the centre of the banded carbonatite. The system containing the main orebody trends about N 85° E; the other system trends about N 65° W and includes a series of carbonatite dykes extending 300–400 m along trend. The emplacement of the Palabora ore deposit was controlled by the two strong fracture patterns.

Copper mineralization in the main ore deposit is believed to be of epigenetic origin. Following the intrusion of transgressive carbonatite, the core cooled and intense fracturing ensued. This fracturing provided conduits for migrating fluids and led to the deposition of minerals in veinlets. Generally, the veins are less than two centimetres wide and are continuous for up to one metre, both horizontally and vertically. The banded carbonatite and foskorite contain widely scattered grains of copper sulphide mineralization (mostly bornite). Since this occurs as discreet grains disseminated throughout the rock, it is considered to be syngenetic. The copper mineral content decreases with distance from the core into the older rocks. However, essentially all banded carbonite and a substantial portion of foskorite are mined and processed as copper ore (Fig. 30).

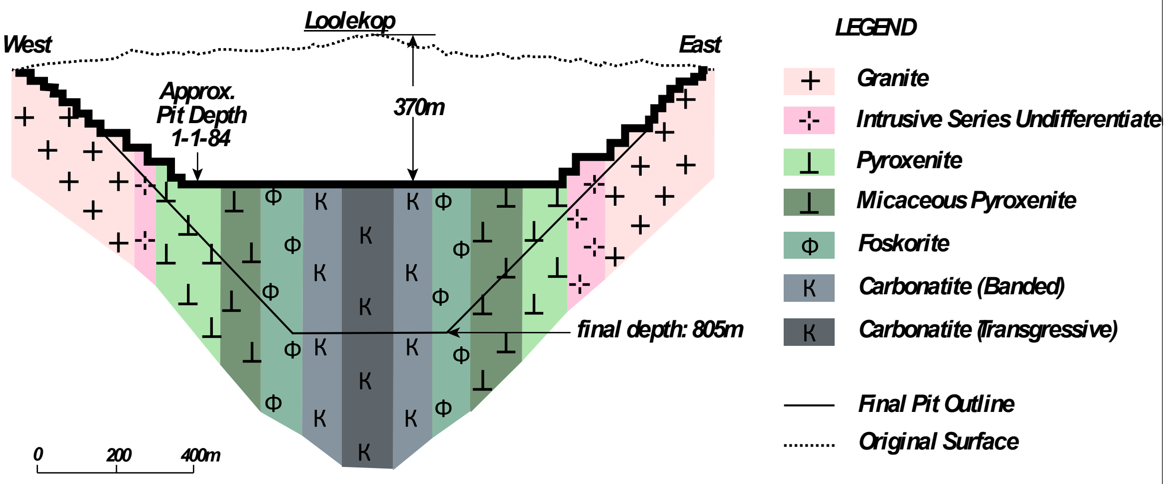

FIG. 30. Idealized E–W vertical section through the Palabora open pit, illustrating the mine plan in relation to the intrusive rock sequence (adapted from [39]).

The principal ore minerals at Palabora are chalcopyrite and bornite, both copper sulphides. Three other copper sulphide minerals are present in minor amounts and contribute to the total copper recovered. In addition, twelve other sulphides of copper, nickel and iron are present in minor to trace amounts. Gold, silver, platinum and palladium also occur in trace amounts. Those minerals are recovered from the final sludge generated in the electrolytic refining process, while nickel sulphate is recovered from the tankhouse electrolyte. The source of uranium at Palabora is the mineral uranothorianite (U, Th) O_2, which contains approximately 45% U and 43% Th. Uranothorianite occurs in small amounts in close association with copper sulphide mineralization.

The Palabora Mining Company copper pit was in operation from 1964 through 2002 and at one time was second only to Bingham Canyon as the largest open pit mining operation in the world [41, 42]. In 1990, the pit was being developed to reach its ultimate depth of 805 m by 1997. The deposit had been drilled to a depth of 1370 m with no decrease in grade or in diameter of the carbonatite pipe. In 1989, mine production totalled 62.1 million t (202 353 t/d) of which 29.4 million t of ore averaging 0.49% Cu was delivered to the concentrator. The waste to ore ratio is 2.1:1. The construction of the underground mine was completed in 2004 and by May 2005 the mine was consistently achieving 30 000 t/d. The major items of mining equipment used in 1989 included:

—	Rotary drills	6
—	Haul trucks (150 t)	52
—	Haul trucks (170 t)	3
—	Haul trucks (90 t)	3
—	Shovels (19 m^3)	6
—	Dozers and graders	25

Installation of a new trolley assist system for truck haulage involving 7.8 km of overhead trolley line resulted in considerable savings in truck fuel costs. During 1989, an in-pit crusher and conveyor project was completed which eventually reduced the truck haulage fleet. By the second half of 1989, 70.8% of the total ore mined was processed through the in-pit crusher [43].

Mining costs in 1992 were reported to be R 3.32 (US $1.16) per tonne of material moved, which totalled 37.8 million t.

4.1.6.3. Milling

During 1989, the average milling rate for ore totalled 80 571 t/d in two mill circuits. The older conventional circuit processed 53 465 t/d of ore. The 2 SAG mills processed 27 106 t/d. In the conventional circuit, ore was crushed in three stages and ground in rod and ball mills prior to entering the flotation circuit. The SAG mills produced from run-of-mine ore a product that was sent directly to the flotation circuit. In 1989, the flotation circuit produced a total of 323 120 t of copper concentrates at an average grade of 37.43% Cu with an 83.57% recovery rate.

The average copper concentrate smelting rate in 1989 was 984 t/d. New anode production, including purchased copper concentrates, totalled 125 794 t of fine copper. Sulphuric acid production totalled 135 661 t of 100% sulphuric acid. Production of cathode copper totalled 126 206 t. The casting plant produced 69 646 t of continuous cast copper rod. The nickel sulphate refining plant produced 548 t of refined hexahydrate. Precious metal recovery from refinery slimes totalled 17 118 kg. Milling costs in 1992 were reported to be R 3.81 (US $1.34) per tonne of ore milled, which totalled 29.0 million t.

Uranium recovery

In 1989, some 60 000 t/d of tailings from the copper flotation plant were initially processed in the magnetic separation plant where a total of 178 204 t of magnetite concentrates were produced at an average content of 64.8% Fe and 2.31% TiO_2. After magnetite removal, tailings were deslimed in cyclones and subjected to gravity concentration. The gravity concentration plant consisted essentially of Reichert cones to provide a heavy mineral concentrate, followed by jigs and tables used to separate the heavy mineral concentrate into uranothorianite and baddeleyite (ZrO_2) fractions.

Residual magnetite was also recovered in the gravity concentration circuit (Fig. 13, Table 2). Baddeleyite concentrates were cleaned by flotation methods to recover copper minerals which were sent to the smelter. A total of 13 022 t of baddeleyite concentrates and 357 t of copper contained in the concentrate were recovered in 1989. Tailings from the gravity concentrates plant were sent to Foskor for phosphate recovery [43]. The production of uranium at Palabora was not significant in terms of world supply. However, in terms of grade of feed treated and concentration ratio (Table 13) it was an exceptional operation.

TABLE 13. PALABORA GRAVITY CIRCUIT DATA

Product stream	U (%)	ZrO_2 (%)
Rougher cone feed	0.0030	0.40
Rougher and scavenger cone tailings	0.0008	0.12
Rougher cone concentrate	0.0059	1.60
Scavenger cone tailings	0.0011	0.13
Cleaner cone feed	0.0051	1.00
Cleaner cone tailings	0.0019	0.15
Recleaner cone feed	0.0144	4.20
Recleaner concentrate (quaternary cone feed)	0.0314	8.70
Quaternary cone concentrate (rougher table feed)	0.0696	17.50
Uranothorianite table concentrate	2.5445	77.40
Zirconia table concentrate	0.1442	40.00
Recovery	50.7294	67.64

Fig. 31 presents a general schematic of the uranium recovery process at Palabora. Uranothorianate concentrates were leached with 60% nitric acid at 75°C. The reaction is exothermic and temperatures of 100°C were reached during the 45-minute leach period. Nitric acid consumption totalled 430 kg/t of concentrates. Leached pulp was thickened and filtered to remove sludge and pregnant solution was clarified and sent to the SX circuit, which consisted of six extractions, four scrubbing, eight stripping and two organic regeneration cells. Pregnant solution containing the U_3O_8 also contained 120–140 g/L of ThO_2 and about 120 g/L of free nitric acid. The organic consisted of 10% tributyl phosphate in kerosene. Loaded organic was scrubbed with uranyl nitrate solution to remove traces of thorium. Scrubbed solvent was stripped using water at 40°C, which contained 3 g/L of nitric acid. Uranium in the stripped liquor was precipitated with ammonia forming ammonium diuranate, which was thickened and filtered and then calcined in a multiple hearth furnace at 650°C, cooled in a screw conveyor and packaged in steel drums for shipment [44].

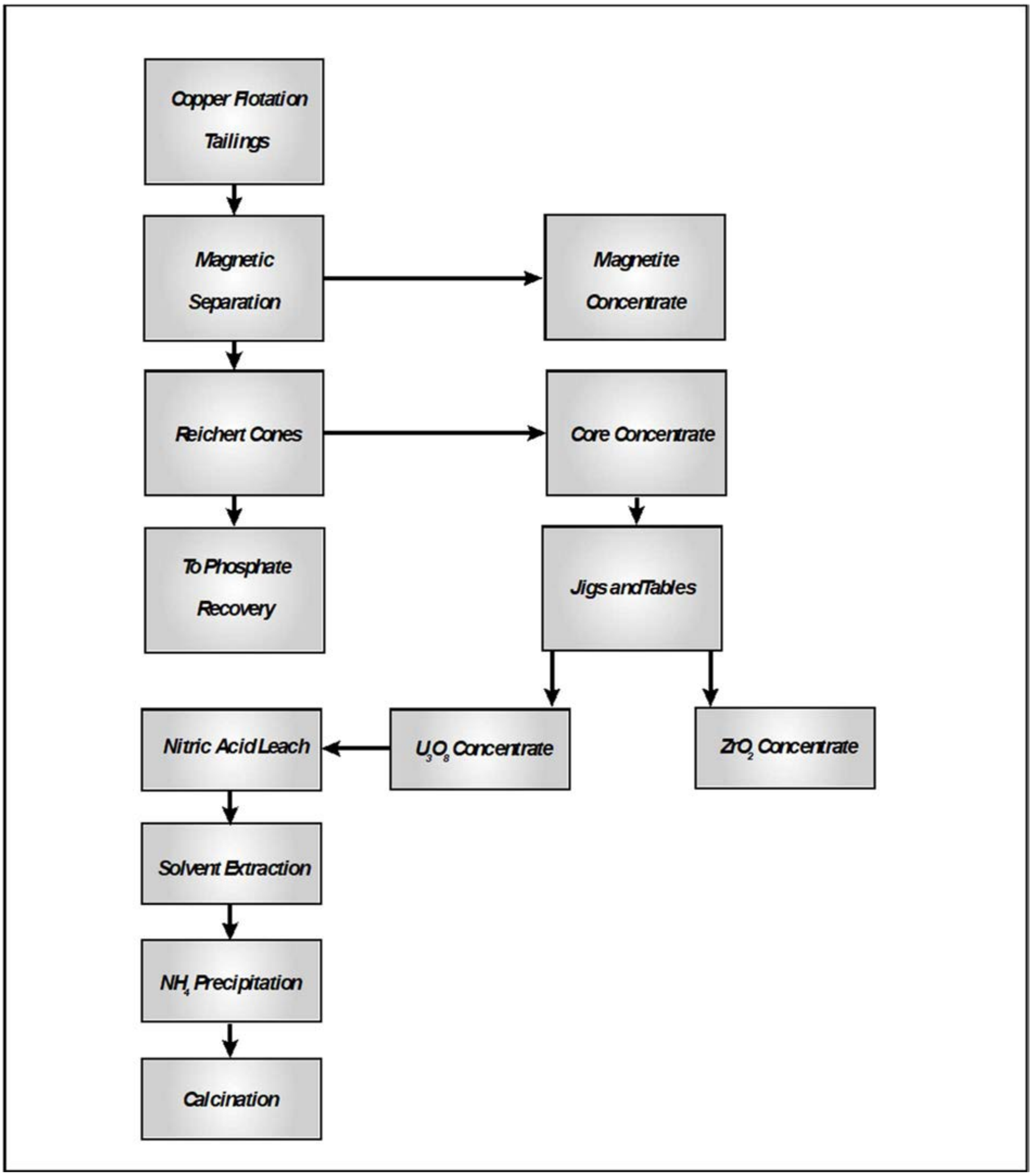

FIG. 31. Palabora uranium flowsheet (courtesy T. Pool).

4.1.6.4. Uranium production data

During its 32-year history of operation, from 1971 through 2001, Palabora's uranium production totalled 3047 tU. Maximum production of 218tU was achieved in 1982 (Fig. 32).

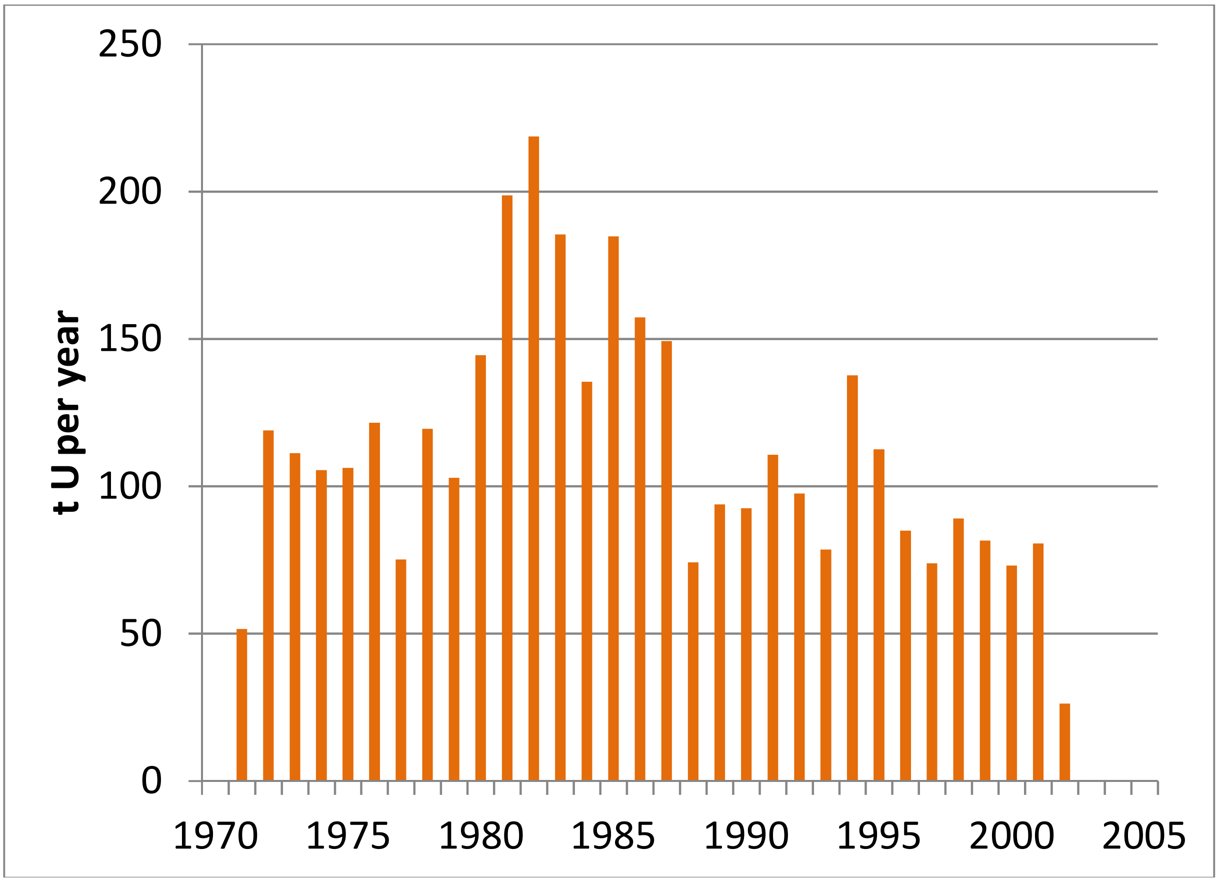

FIG. 32. Palabora annual uranium production (courtesy T. Pool).

Relative value contribution of uranium and copper to the Palabora operation for 1996–2000 is presented in Table 14.

TABLE 14. RELATIVE VALUE CONTRIBUTION AT PALABORA

Year	Production		Nominal prices (US$/kg)		Value (US$ million)	
	U (kg)	Cu (t)	U	Cu	U	Cu
1996	84 961	100.2	46.23	3.79	4.6	220.8
1997	73 852	105.6	35.94	3.68	3.1	232.7
1998	89 017	104.0	31.7	2.78	3.3	229.2
1999	81 604	100.0	30.52	2.65	2.9	220.4
2000	73 114	87.7	23.32	2.89	2.03	193.3

4.2. POLYMETALLIC IRON-OXIDE BRECCIA COMPLEX

4.2.1. Olympic Dam (Australia)

The Olympic Dam copper–uranium–gold–silver deposit is situated in South Australia, about 520 km north-west of Adelaide. It is categorized as a polymetallic iron-oxide breccia complex in the IAEA classification scheme.

4.2.1.1. History

The Olympic Dam copper–uranium–gold occurrence is one of the world's largest known mineral deposits. It was discovered in July 1975 by exploration geologists at Western Mining Corporation (WMC). The first hole drilled to test a geological concept conceived by WMC, based on regional geophysical anomalies, penetrated 38 m of copper sulphide mineralization at a depth of 353 m [45].

WMC announced on 27 July 1979 that an agreement in principle had been reached to conduct further project evaluation under a joint venture arrangement. Roxby Mining Corporation Pty Ltd, a wholly-owned subsidiary of WMC, would retain a 51% interest in the project and BP Australia Ltd and BP Petroleum Development Ltd, both subsidiaries of British Petroleum (BP), would collectively earn a 49% interest in the Olympic Dam Joint Venture (ODJV). The terms of the agreement required BP to provide A$50 million for further exploration, metallurgical testing and other work directed towards the completion of a feasibility study for commercial development of Olympic Dam.

The ODJV entered into an indenture agreement with the Government of South Australia in June 1982. The agreement established site boundaries for the mine and processing plant and specified that special mining leases would be granted if applied for. In addition, the ODJV was obligated to invest at least A$50 million on project evaluation, including completion of a feasibility study, between 1 January 1982 and 31 December 1984. The ODJV was also obliged to notify the Government of South Australia before 31 December 1987 of any decision taken with regard to further project development [46].

An exploration shaft was completed in October 1982 and a pilot metallurgical plant operated from March through December 1984. The decision to proceed with full development was announced in May 1986. Owing to the long term nature of the Olympic Dam project, the ODJV partners chose to establish a permanent community facility for their workforce, rather than operate a fly-in, fly-out operation as is often done at remote mining projects in Australia [47].

Total expenditures through commencement of construction amounted to A$200 million. Construction was completed in late 1988 at a cost A$550 million. Initial annual production capacities were 1.3 million t of ore, 45 000 t of copper and 1000 tU [48].

WMC reacquired BP's 49% interest in 1993. A major expansion was completed in 1999 which lifted copper output to 200 000 t/year and uranium output to 3800 tU/year. In mid-2005, BHP Billiton gained control of WMC in an A$9.2 billion takeover. Another major expansion plan was announced in 2007 but this was later shelved due to unfavourable market conditions. This plan would have created the world's largest open pit mine with annual ore production of 70 million t, representing a sevenfold increase. In terms of this expansion, copper production would have grown to approximately 750 000 t/year and uranium production would have risen to 12 700 tU/year.

4.2.1.2. Geology

Olympic Dam in South Australia is an iron oxide–copper–gold–uranium (IOCG–U) deposit located on the margin of the Archaean–Proterozoic Gawler Craton [49]. The deposit belongs to the shallow crustal level type IOCG characterized by a Fe–Cu–Au–U ore hosted within a multi-stage breccia complex with haematite dominated breccias [50]. The breccia complex cross-cuts the 1590 Ma Roxby Downs Granite (Figs 33 and 34) [51]. Geodynamic models at the start of Early Mesoproterozoic magmatism in the

Gawler Craton vary from intracontinental anorogenic rift setting, foreland basin, plume modified back-arc or lithospheric delamination [52]. Most authors consider that ore deposition is coeval with the ~1.6 Ga magmatic activity of the Hiltaba granitoid suite and consanguineous Gawler Range Volcanics [53]. The Olympic Dam deposit is thought to have formed at a shallow depth, with multiple magmatic, phreatic and hydrothermal explosions [54] or beneath a sedimentary basin [55]. The fluids involved in the ore genesis involve an early hot, reduced Fe-rich fluid (magmatic?) that mixes with a cooler, oxidized meteoric fluid [56].

Olympic Dam is one of the largest resources of low grade uranium [57], hosting about 2.1 million tU at an average grade of 220 ppm U [49] and the only one in operation recovering uranium as a by-product. The breccia comprises haematite ± sericite-altered clasts of host granite, clastic sediments (conglomerate, sandstone and mudstone) and porphyritic Gawler Range Volcanics cemented by a fine-grained haematite-rich matrix. Mafic to ultramafic dykes cross-cut both the granite and the breccia. Major alteration minerals within the breccia are quartz, orthoclase, sericite, chlorite, haematite (± magnetite), baryte, fluorite, apatite and siderite.

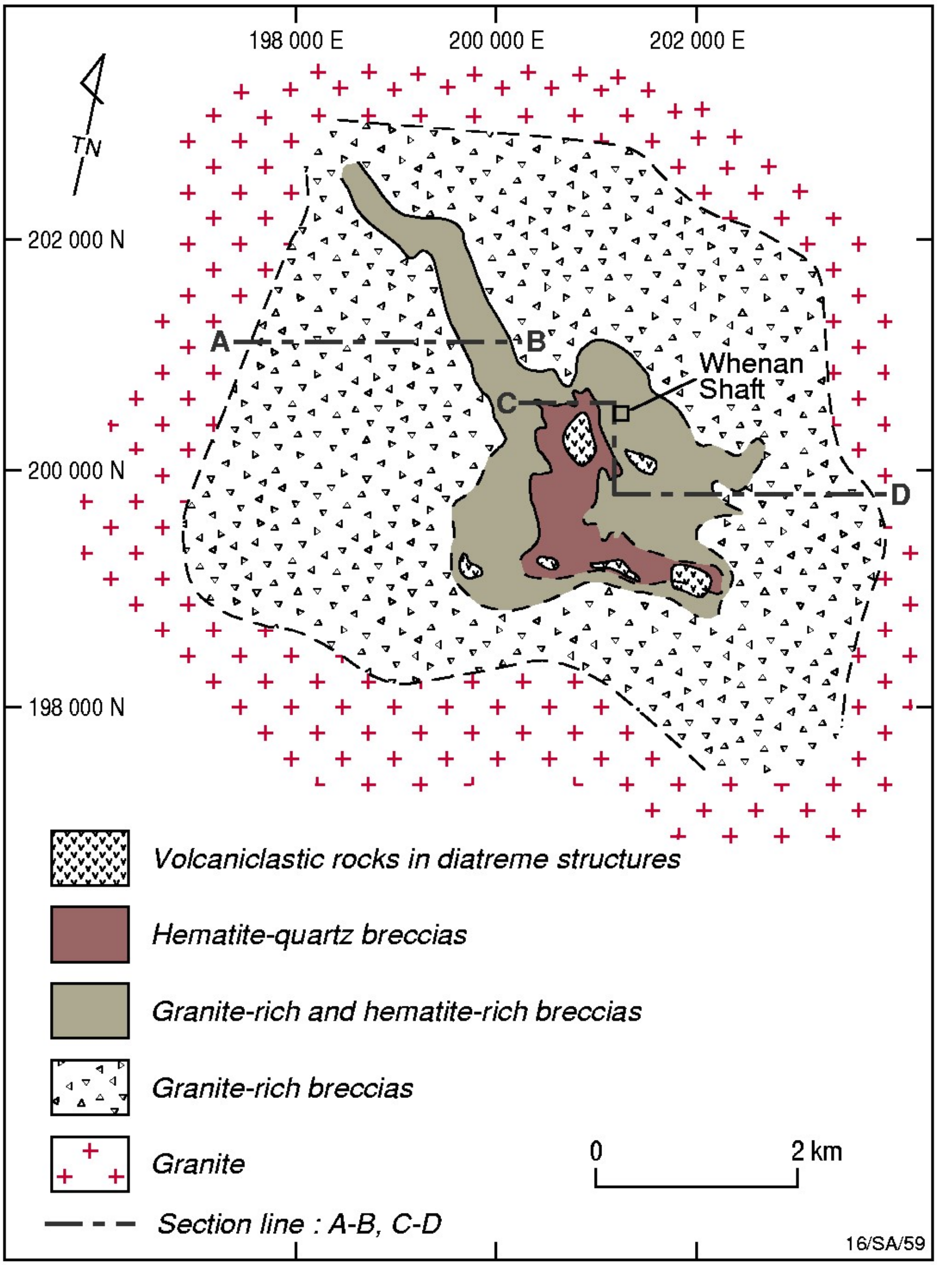

FIG. 33. Simplified geological map of the Olympic Dam breccia comple. Adapted from [58].

4.2.1.3. Mineralization

Mineralization is known to extend from a depth of 343 m to a drilled depth of 1150 m. The greatest concentration of mineralization is usually associated with areas of intense haematite and chlorite alteration.

The sulphides are generally disseminated within the breccia matrix, more rarely in veins, and the main sulphide phases are pyrite, chalcopyrite, chalcocite, bornite (± sphalerite), molybdenite, tennantite and galena.

Uraninite (both euhedral and collomorph), coffinite and brannerite are the main uranium minerals. They occur as disseminated isolated grains and aggregates, as well as micro-veinlets. Other minor uranium-bearing minerals are florencite, apatite, zircon, monazite, xenotime and haematite. Uranium–lead dating of uraninite inclusions within haematite grains indicates that at least some of the U has an age of 1.59 Ga [59], similar to the age proposed by Johnson and Cross [54] and other IOCG deposits in Southern Australia [53]. Some of the U in the deposits was possibly introduced during a later event, possibly as late as about 0.5 Ga [60].

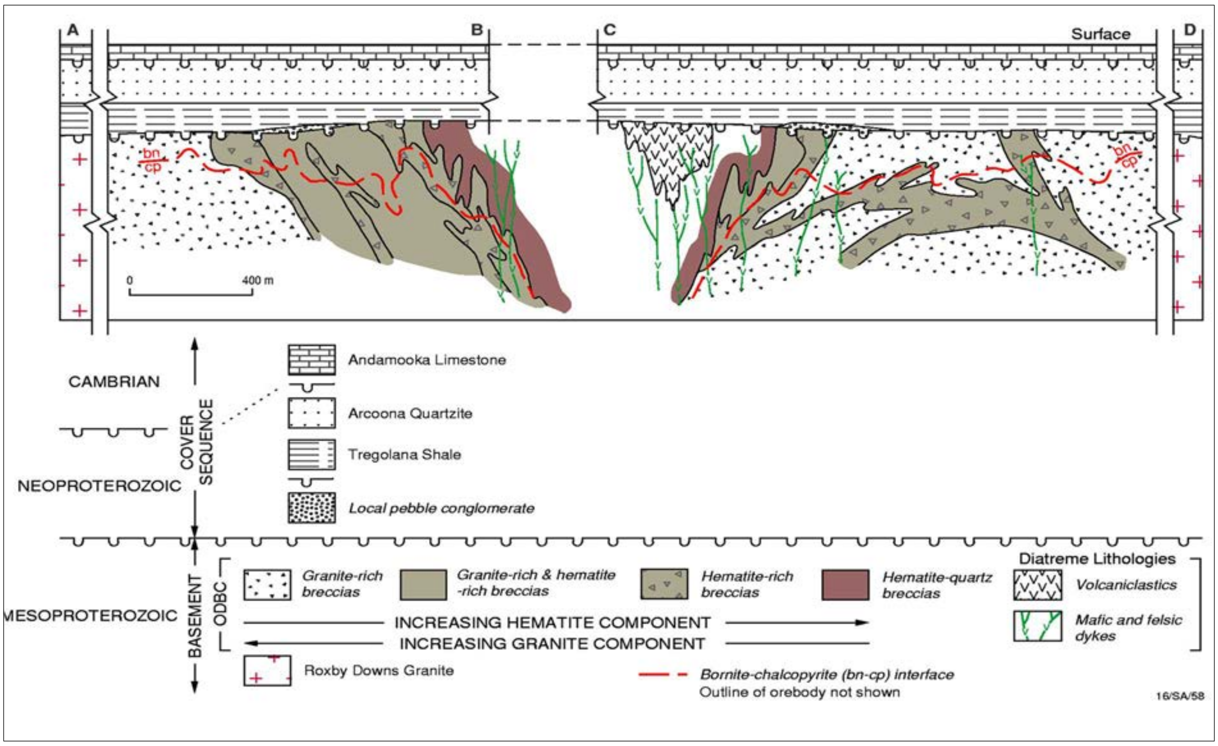

FIG. 34. Simplified geological cross-section of the Olympic Dam breccia complex. Adapted from [58]. Refer to Fig. 33 for location of section A–B.

Free gold is scattered throughout both sulphide zones. The concentration is higher in the transgressive ore zone. It also occurs with haematite and is disseminated in the matrix. Silver is closely associated with gold and the silver content is higher in the bornite-rich sulphide zones. It occurs both as native silver and as silver tellurides.

The REE minerals bastnaesite $((Ce, La,)\ CO_3(F,OH))$ and florencite $(CeAl_3(PO_4)_2(OH)_6)$ average 0.50% combined rare earths. They occur primarily in association with the sulphides, but commonly with haematite and sericite alteration. The recovery of rare earths is only a minor consideration in project planning owing to the low grade nature of the occurrence.

A summary of the resources at Olympic Dam is given in Tables 15 and 16. As of 2014, uranium resources stood at 2.075 million t at an average grade of 0.022% U.

TABLE 15. OLYMPIC DAM INITIAL RESERVE/RESOURCE SUMMARY

Initial mining reserve of copper/uranium ore		Initial mining reserve of gold ore	
Ore (10^6 t)	7.7	Ore (10^6 t)	2.3
Copper (%)	3.7	Copper (%)	0.3
U (kg/t)	1.6	U (kg/t)	0.1
Gold (g/t)	0.4	Gold (g/t)	5.8
Silver (g/t)	14.7	Silver (g/t)	1.5
Probable in situ ore		Overall resource estimate	
Ore (10^6 t)	450	Ore (10^6 t)	2000
Copper (%)	2.5	Copper (%)	1.6
U (kg/t)	0.8	U (kg/t)	0.6
Gold (g/t)	0.6	Gold (g/t)	0.6
Silver (troy oz/t)	6.0	Silver (g/t)	3.5

TABLE 16. OLYMPIC DAM RESERVE/RESOURCE SUMMARY
(as of June 2014)

Resource category	Ore (10^6 t)	Copper (%)	Uranium (%)	Gold (g/t)	Silver (g/t)
Measured	1 220	0.99	0.025	0.38	2
Indicated	4 480	0.87	0.021	0.30	2
Inferred	3 850	0.73	0.021	0.24	1
Total	9 550	0.81	0.022	0.29	2
Resource category	Ore (10^6 t)	Cu (10^6 t)	Uranium (tU)	Au (t)	Ag (t)
Measured	1 220	12.1	310 255	464	2 440
Indicated	4 480	39.0	949 415	1 344	8 960
Inferred	3 850	28.1	815 804	924	3 850
Total	9 550	79.2	2 075 474	2 732	15 250

4.2.1.4. Mining

The first phase of production was designed to mine 1.5 million t/year of ore and about 500 000 t/year of waste rock. Construction and start-up of the metallurgical plant was completed in 1989.

Mine access is affected by vertical shafts and a service decline. The initial shaft was a six-compartment rectangular (6.3 m × 3.5 m) shaft 500 m deep. Hoisting was undertaken at the rate of 480 t/h (about 360 t/h ore and 120 t/h waste) using two 9.25 t bottom dump skips. An automatic operation cage with a 20-person capacity provided access for underground workers. The shaft also served as a fresh air intake.

The service decline is 4 km in length and 6 m × 5 m in cross-section with a decline gradient of 1:9. The decline, in addition to providing access for major mining equipment, also serves as a fresh air intake. Mine development headings are 6 m × 4.8 m; stope drill headings are 4.5 m × 4.8 m. Major development equipment included: two-boom electro–hydraulic jumbos, ANFO (blasting agent) loaders, front-end loaders with 4.2 m^3 capacity buckets and haulage trucks (21 or 18 m^3 capacity).

Ground support is by galvanized split set rock bolts. Raise drilling is performed extensively. Surface raises (for ventilation) vary in diameter (2.4 m, 3.0 m and 4.0 m), the longest being 479 m long. Underground drilled raises are 1.8 m in diameter for slot raises or 2.4 m in diameter for ore passes and ventilation raises.

The stoping method is dictated by the ore zone height; if more than 30 m, a modification of sub-level open stoping is used; if less, a mechanized cut-and-fill method is used. The principal method employed is sub-level stoping. Stope production drilling is by electro–hydraulic long hole jumbos drilling ring patterns developed by the staff. ANFO blasts range in size from 500 t when opening an undercut slot to 20 000 t in ring firing.

Ore from the stoping areas is mucked by 7.2 m^3 load–haul–dump units and trammed to ore passes. A truck loading station, using a 2.0 m × 6.5 m pan feeder, loads ore or waste directly into haulage trucks.

When the final extraction of open stopes is complete, cemented aggregate backfill is introduced. The cemented fill design allows the fill to stand unsupported during extraction of adjacent pillars. Fill material comprises 40% crushed mine waste and 60% coarse mill tailings. Portland cement and fly ash are added at the rate of 4% each to the solids total. A pulp density of 80% is maintained. Backfill is introduced from the surface to completed stopes via 300 mm boreholes.

All ore and waste are crushed underground to -200 mm size by a 1676 mm × 1219 mm double-toggle jaw crusher rated at 550 t/h. Crushed rock is conveyed by an inclined (11:6) belt conveyor 458 m to two fine rock 2500 t capacity storage pockets. The belt, which is 1050 mm wide and 458 m long, is driven by a 400-kW motor.

Hoisting and skip loading is wholly automatic. Skips are hoisted by a 1370 kW double-drum hoist. Hoisted ore is conveyed 1.2 km from the shaft to metallurgical plant stockpiles.

4.2.1.5. Milling

The metallurgical plant consists of a copper concentrator, hydrometallurgical plant, copper smelter, sulphuric acid plant, and copper and gold–silver refineries (Fig. 35). The metallurgical plant produces four main products: refined copper, gold bullion, silver bullion and yellowcake (U_3O_8).

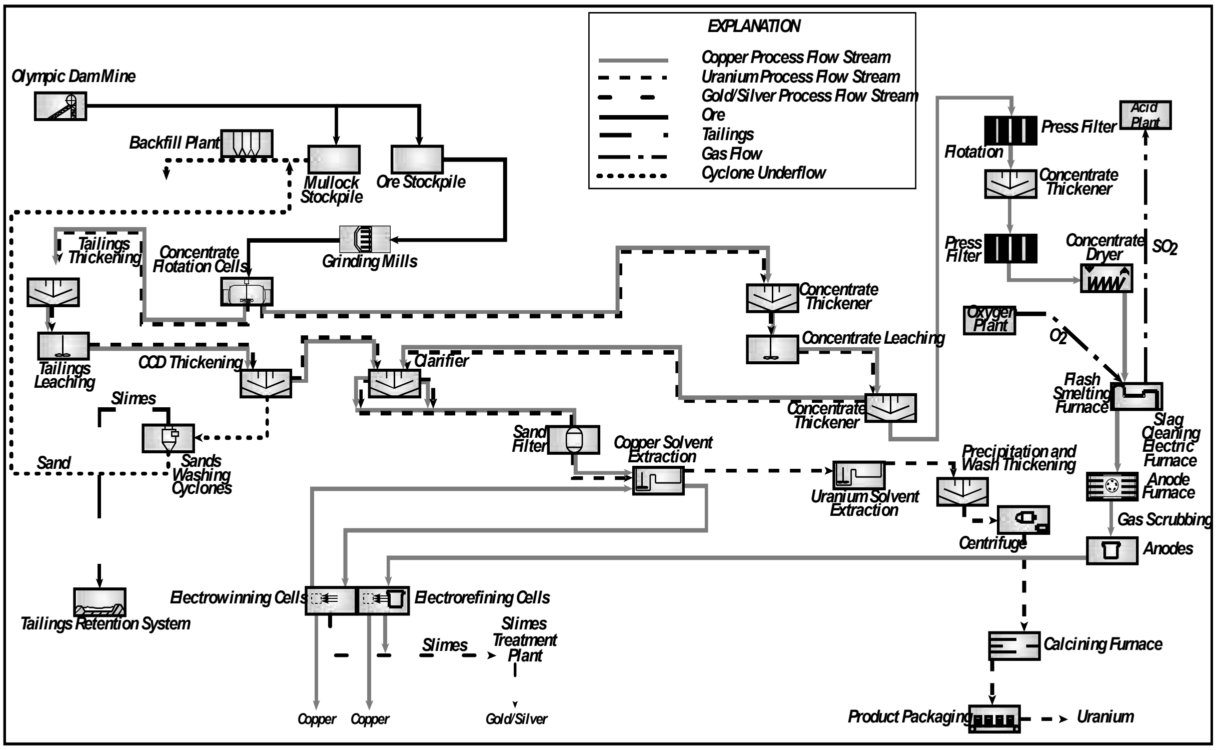

FIG. 35. Process diagram for Olympic Dam (adapted from Ref. [61]).

In the initial milling process, stockpiled ore was fed to a 7.4 m × 3.0 m SAG mill where the ore was reduced to 12 mm at the rate of 190 t/h. Oversize (60% of the SAG mill feed) was crushed in a 1.68 m short head crusher to -12 mm and screened on DSM screens. Oversize was returned to the SAG mill feed. Ore passing through the DSM 4 mm screen cloth was fed to the secondary ball mill (4.42 m × 5.69 m) circuit where it was reduced to 75 µm (200 mesh) and sent to the flotation circuit for copper recovery. All flotation cells were Dorr Oliver; the rougher and scavenger were 17 m^3 dual cell assemblies and the cleaners were 4.25 m^3 dual cell assemblies. The two reagents used were sodium ethylxanthate (50–100 g/t*)* and methyl isobutylxanthate (60 g/t).

First stage cleaner concentrates were reground to 20 µm (400 mesh) and then recleaned in two stages. Flotation tailings were dewatered in a 15.2 m diameter thickener prior to being pumped to the uranium leach section. Concentrates were dewatered in a 12-m diameter thickener and pumped to a concentrate leach section for removal of the uranium prior to smelting.

Flotation retention time amounted to 18 min (rougher and scavenger) and 10 min in each cleaning stage.

The uranium leaching section treating flotation tailings consisted of six mechanically agitated tanks arranged in series. The conditions of the initial agitator were monitored for:

- Acidity (sulphuric acid): 10–15 g/L;
- Redox potential: 430–450 mV (calomel);
- Temperature: 60°C.

Total retention time was 12 h. Slurry from the last tank was pumped to a counter-current decantation circuit for solid–liquid separation. Washed solids (tailings) were pumped to the tailings dam or were deslimed and used for backfill in the mine. Pregnant overflow, joined by concentrated leach solution, was clarified and sent to the SX plant.

Copper flotation concentrates were also leached for uranium in six mechanically agitated, rubber lined, covered tanks.

The following leach conditions were maintained to achieve maximum uranium extraction while minimizing copper extraction:

- Long residence time: 48 h minimum;
- Leach density: <50% solids;
- High acidity: Sulphuric acid fed to tanks 1, 2 and 4;
- Low temperature: 40°C.

Air entrainment was minimized to prevent oxidation. Leached copper concentrate slurry was thickened and overflow containing copper and uranium in solution joined the pregnant uranium solution from the tailings leach circuit. Copper concentrate (thickener underflow) was filtered, washed, neutralized with caustic soda and refloated to produce a 55% copper concentrate for the smelter. Aqueous feed to the SX plant was clarified pregnant solution containing copper and uranium in solution. There were two SX plants: the first selectively recovered dissolved copper from the leach liquor in three mixer–settler tanks and an after-settler using a modified oxime/kerosene solvent. The resulting copper electrolyte was pumped to the copper refinery. Copper raffinate (tailings solution) from the copper SX plant was fed to the uranium SX plant. Dissolved uranium was extracted using an amine/kerosene solvent in mixer–settlers. The SX circuit consisted of three extraction stages, one after-settler, three solvent and scrubber stages, four stripping stages and one regeneration stage.

Loaded solvent from the extraction phase was washed (counter-current) in acidified water and uranium was stripped into an aqueous ammonium sulphate solution and pumped to the precipitation section. This solution was treated with ammonia to precipitate uranium as yellowcake, which was washed, thickened, centrifuged and roasted at 600–700°C to remove moisture and impurities. The uranium oxide product was packaged in steel drums for shipment. Uranium recovery in 2014 was reported to be 72%. After filtering and drying to attain a 0.2% moisture content, the leached copper flotation concentrates were smelted in an Outokumpu flash furnace. The furnace consisted of:

(a) A reaction combustion shaft where copper concentrates and fluxes were smelted in an atmosphere comprising 70–95% oxygen;
(b) A rectangular settler where blister copper and slag were removed by tapping;
(c) An uptake shaft from which sulphur dioxide gas was recovered to produce 70% of the sulphuric acid required in the metallurgical plant.

Oxygen was supplied by the smelter’s 100 t/d oxygen plant. Blister copper containing 0.6–1.0% sulphur and 98.5% copper was fire refined in two 4 m × 9.1 m furnaces and cast into anodes containing 99.6% copper and sent to the refinery. Gold and silver in the smelter feed concentrates followed the copper in the anodes. An electro-refining process developed in Australia was used to produce high quality copper cathodes. The anodes were suspended in an electrolyte in seven electro-refining sections (total of 224 cells). The electrolyte contained 45 g/L copper and 185 g/L sulphuric acid at an overflow temperature of 62°C. Direct current was passed through the anode and electrolyte to a stainless steel ‘mother’ plate where the copper was redeposited. The plates were stripped and the resulting cathodes bundled for shipment. Copper recovery in 2014 was reported to be 94%.

Insoluble impurities that settled in the refinery tanks contained most of the gold and silver in the original ore. Anode slimes were processed to remove metallic copper and uranium; the gold and silver were extracted by intensive cyanidation followed by electro-winning to produce gold bullion and silver bullion. Gold and silver recoveries in 2014 were reported to be 70% and 64%, respectively. Coarse gold present in the ore was recovered in the mill gravity concentration circuit by spirals which produce a coarse concentrate. This concentrate was further treated and co-refined with the anode slimes.

4.2.1.6. Production data

Recent production at Olympic Dam is summarized in Fig. 36 and Table 17. In 2016, uranium production was 3235 tU. An analysis of Olympic Dam revenues based on nominal historical prices for copper, uranium and precious metals is given in Fig. 37.

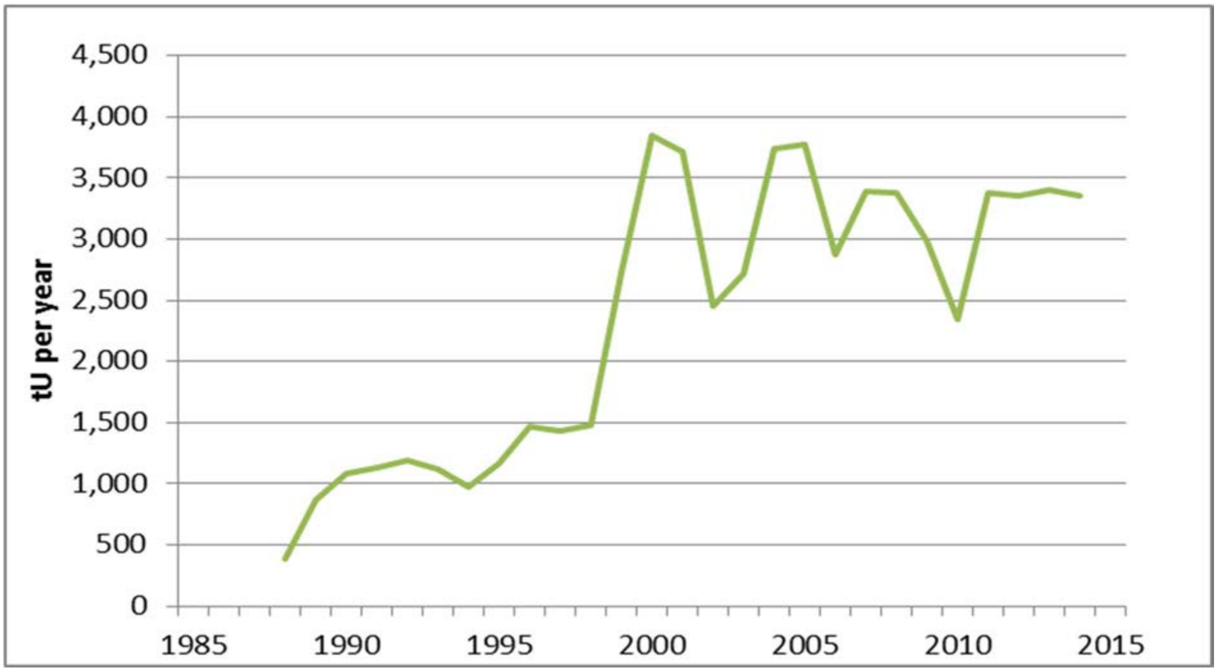

FIG. 36. Historical uranium production at Olympic Dam (courtesy T. Pool).

TABLE 17. OLYMPIC DAM: ANNUAL PRODUCTION FOR PERIOD 1994–2014

Year	Copper (t)	Uranium (tU)	Gold (kg)	Silver (kg)
1994	66 999	969	853	11 360
1995	75 358	1 162	943	10 363
1996	81 323	1 466	1 061	12 573
1997	77 204	1 426	881	10 061
1998	73 645	1 476	983	9 539
1999	138 272	2 732	949	7 626
2000	200 423	3 850	2 176	19 445
2001	200 523	3 714	3 528	28 394
2002	178 120	2 451	2 000	20 029
2003	160 079	2 716	2 679	18 706
2004	224 731	3 734	2 757	26 800
2005	207 815	3 801	3 061	25 415
2006	183 000	2 877	2 928	25 692
2007	177 600	3 387	2 887	28 087
2008	195 900	3 383	2 935	25 692
2009	155 500	2 980	3 199	25 941
2010	131 800	2 347	2 128	17 636
2011	196 800	3 381	3 623	30 109
2012	168 200	3 419	3 326	27 341
2013	174 400	3 399	3 730	27 465
2014	187 800	3 351	4 240	33 188

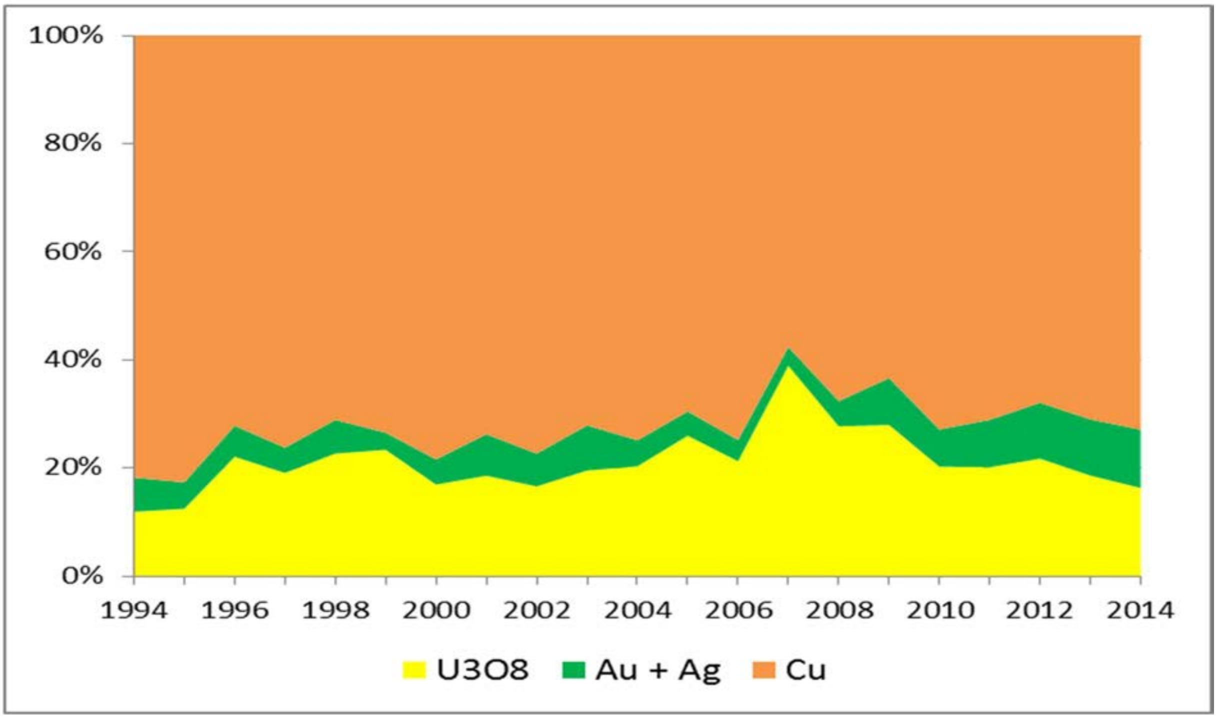

FIG. 37. Nominal revenue distribution at Olympic Dam (courtesy T. Pool).

4.3. VOLCANIC-RELATED DEPOSITS: THE CU–CO–ZN–MN BOLEO DEPOSIT (MEXICO)

4.3.1. History

The Boleo Cu–Co–Zn–Mn project is located in Baja California Sur (Mexico). Copper ore was discovered in 1868 and small-scale mining operations were carried out on-site by German and Mexican companies up until 1885. The remote location and the small scale of the activties made the enterprise only economically marginal.

El Boleo was first mined, on a large scale, in 1885 by Compagnie du Boleo, a French company which took ownership of the property and commenced mining after receiving a significant concession and a tax exemption for 70-years. As the ore was extremely rich (a complex mixture of sulphides and oxides with up to 15% Cu), it was fed directly to the smelter with only chrushing as pre-processing. No attempt was made to extract zinc, manganese and cobalt due to the complex metallurgy[62, 63].

In the early part of 20th century, the company was known for using state-of-the-art technology for ore processing and refining. The mine’s powerhouse was considered the most advanced electrical system of its time in the country. The deposit was continuously mined, mainly by underground methods, from 1886 up until 1972, with an estimated 18 million t of ore were treated. Both open pit and underground mining was sporadically carried out after 1972 until the Santa Rosalía copper smelter closed in 1985 [63].

Renewed interest in 1992 by Canadian investors resulted in the establishment of a new mining concession by Baja Mining Corporation Extensive reserves of copper ore, as well as commercial quantities of cobalt, zinc and manganese, were established to exist on the site during subsequent exploration. Over a fifteen-year period, two test mining programmes, two process pilot campaigns and a 38 000 metre infill drilling programme were carried out, culminating in a definitive feasibility study, which was completed by the lease holder, Minera y Metallurgica del Boleo (MMB), in 2007. This study was followed by a technical report issued in 2010. Boleo was predicted to produce Cu–Co–Zn–Mn.

However, the previous metallurgical process used by earlier operators was found to be unfit for the recovery of these elements. Small scale testing had indicated that effective recovery of all four metals, with competitive economics, could be achieved by a hydrometallurgical process. This process and the lack of fresh water at the site necessitated the construction of a desalination plant [63].

Construction cost overruns reported by Baja Mining Corp. in 2012 were thought to possibly halt or delay construction of the project. An agreement transferring majority ownership and control of the project to a Korean consortium, in return for funding the reported cost overruns, was reached in July 2012. The consortium led by Korea Resources Corporation (KORES) holds a 90% interest in MMB.

November 2010 construction of the project started and mining operations began in the first half of 2014. Copper was first produced in January 2015. The Co–Zn circuits were then commissioned and came into production by the end of 2015. The mine has an estimated life of 22 years.

The deposit contains an estimated 265 million t of measured and indicated Cu–Co–Zn–Mn resources and 165 million t of inferred resources. The proven and probable reserves are estimated at 70 million t [63]. The mine life (open pit and underground) is expected to be at least 22 years.

4.3.2. Geology

The deposit lies within the Late Miocene El Boleo Formation within fine- to coarse-grained clastic volcano-sedimentary rocks. The rocks lie unconformably over the Comondú Volcanics, andesitic rocks of Early–Middle Miocene. The andesitic rocks are underlain by Cretaceous granodiorite.

Five coarsening upwards sedimentation cycles represent near-shore shallow marine basins with deltaic deposition, and these characterize the El Boleo Formation, which ranges in thickness from 50 to 250 m and averages 140 m. The formation consists mainly of interbedded tuffs and tuffaceous conglomerates of latitic to andesitic composition, together with a basal marine limestone, thick layers of gypsum and a few lenses of fossiliferous sandstone. The formation is considered to represent deltaic deposition along a shoreline.

The deposit hosts seven mineralized units or mantos designated, from top to bottom, as 0, 1, 2, 3AA, 3A, 3 and 4. The mantos represent clay-rich volcanic ash horizons (ash altered to montmorillonite). They consist of laminated basal zones of less than 1 m thick overlain by intrabasin slump breccias up to 20 m thick. They are overlain by increasingly coarser material, siltstone, tuffaceous claystone, pebbly sandstone, feldspathic sandstone, and locally cobble- to boulder-sized conglomerates. The major producing manto has been number 3, which yielded approximately 83% of all production between 1886 and 1985 when the plant shut down [63].

4.3.3. Mineralization

Each manto is distinct in terms of its copper/cobalt ratio and relative zinc, manganese and carbonate concentrations. Mineralization is finely disseminated in the 20-m thick slump breccias. Ore minerals include pyrite, covellite, chalcopyrite, chalcocite, bornite, carrolite and sphalerite categorized as primary Cu–Fe–Zn sulphides and chrysocolla, malachite, azurite, smithsonite, cryptomelane, pyrolusite and the rare cobalt minerals of pseudoboleite, boleite, and cumengite, which are classified as secondary Cu–Zn–Mn minerals [63].

In addition to Cu, Zn, Co and Mn, the deposit also contains low grade uranium within the 10–30 ppm U range, with values of up to 250 ppm [64], which correspond to potential geological resources of about 5000–10 000 tU. The deposit is classified as volcanic-related or volcano-sedimentary in the IAEA classification scheme for uranium deposits.

4.3.4. Processing

The initial feasibility study proposed by Baja Mining Corp. [63] is presented in Fig. 38. The ore would be mined by continuous mining methods and treated by a hydrometallurgical process involving acid–sea water leaching, with recovery of copper and cobalt metal cathode and zinc sulphate monohydrate crystal and could eventually recover manganese carbonate precipitate. The mine is expected to produce 3.1 million t/year over the mine's lifetime. For the first six years it will produce: 56 700 t/year Cu, 1708 t/year Co, 25 364 t/year $ZnSO_4.H_2O$ and over 100 000 t/year Mn as $MnCO_3$.

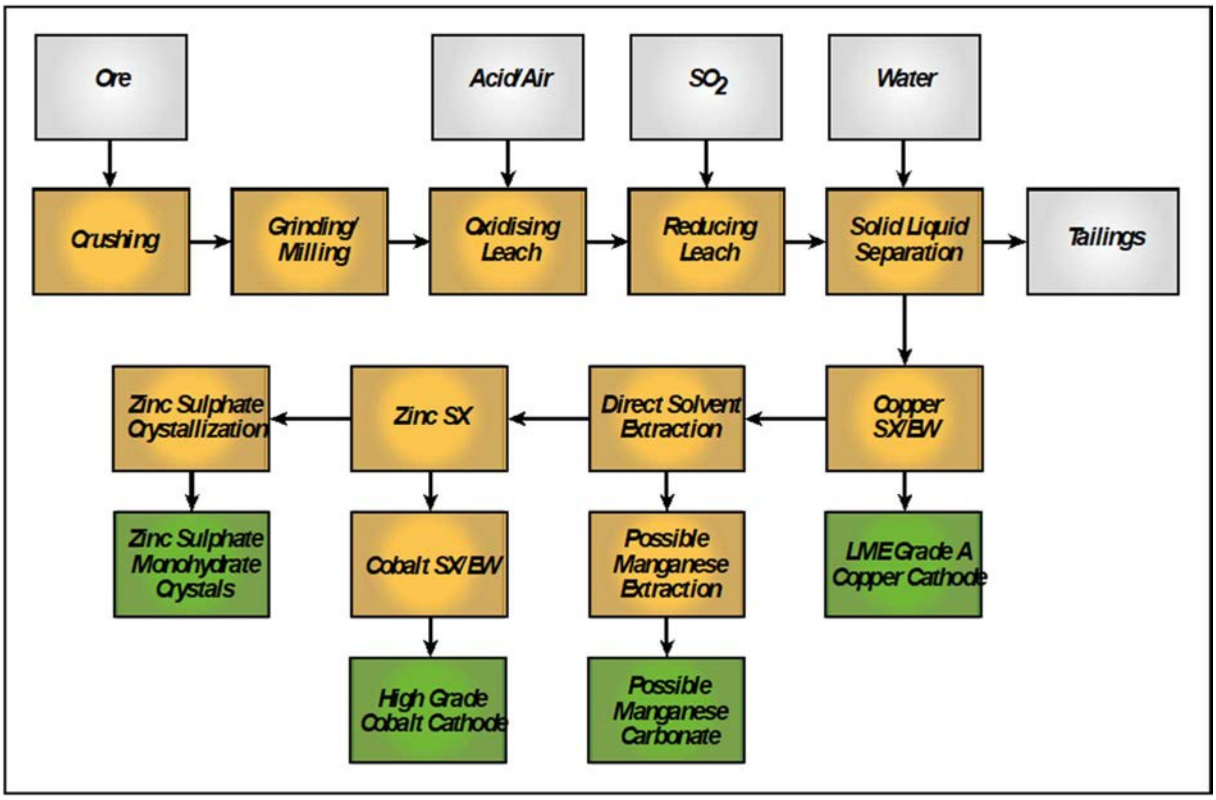

FIG. 38. Summary of the proposed flowsheet for the Boleo mine [63].

Uranium recovery has also been intended to produce a manganese product at some point in the future, and also to ensure the quality of zinc and cobalt production and address any environmental concerns. The initial work was meant to address flotation recovery. The results were poor and it was evident that another flowsheet was required.

The two-stage leaching process has oxidizing and reducing steps, with manganese providing the major oxidation and the SO_2 from the acid plant will be used for the reduction part of the leach.

- It appears that there is leaching both forms of uranium:
  - $UO_3(s) + H_2SO_4 = UO_2SO_4(aq) + H_2O$
  - $UO_2(s) + MnO_2 + 2H_2SO_4 = UO_2SO_4(aq) + MnSO_4(aq) + 2H_2O$
- SGS PP ~20 ppm of U 67% U leach efficiency.

However, it is not clear that the uranium did not re-precipitate during the reduction leach part of the process. It is likely that a U^{+4} species did oxidize initially to become soluble and was then reduced in the second leach reactions. This is to be investigated by BMM.

From pilot tests the following observations were made:

- Washed through counter-current decantation <1 ppm solution loss;
- Tailings ~10 ppm unleached;
- No extraction in Cu SX;
- No precipitation in iron removal.

In this case, the U does not build up in the copper SX extraction circuit (~1700 m^3/h flow rate), unlike the case in the Chilean heap leach operations, because the raffinate is not recycled.

The next part of the Boleo flowsheet is iron precipitation. To date, a pH of 4.0 has been used without consideration for U recovery, which will partially precipitate at this pH. BMM will need to investigate U losses during iron precipitation and perhaps lower the pH slightly to prevent U precipitation.

Solvent extraction work on the direct solvent extraction (DSX) for Zn and Co removal, showed the uranium loading as per the following:

- DSX (LIX 63 and Versatic 10) Zn and Co and U metals;
- Loading: $UO_2SO_4(aq) + 2HR(org) = UO_2R_2(org) + H_2SO_4\ (aq)$;
- Stripping: $UO_2R_2(org) + H_2SO_4(aq) = UO_2SO_4(aq) + 2HR(org)$;
- SGS PP bulk strip solution: 250 ppm U, 13.7 g/L Co.

The major problem is U and Cd contaminating any future manganese production and planned zinc and cobalt production. Ion exchange has not shown promise to date, but additional discussions with DOW were recommended. BMM has to date primarily considered Zn dust precipitation, followed by U recovery from the Zn–Cd–Cu precipitate as per flowsheet below presented in Fig. 39. Alternatively, there is consideration for a uranium leach cycle following Cd recovery on the resulting filter cake.

The proposed process flowsheet consists now of a two-stage atmospheric leach of a whole ore stream in an acidic, chloride environment. The leach circuit consists of both reductive and oxidative leach, processes that over the years have been proven in extensive batch and pilot level testing.

The leached metals are separated from the leach slurry in a counter-current decantation washing circuit. Experience in the nickel laterite plants in Western Australia and the Sepon copper project in Laos has demonstrated that large counter-current decantation circuits can operate effectively on problematic clay ores to recover dissolved metal values. This key process step has been extensively tested at both the bench scale and at pilot plant level and found to be successful in the recovery of metals in solution from leached Boleo solid residues.

Finally, the dissolved metals are recovered from the wash solution and concentrated employing four separate solvent extraction circuits, two electro-winning circuits and a fluid bed drying operation to produce high quality copper and cobalt metal cathode and zinc sulphate monohydrate crystals.

Key to the revised process was the successful demonstration of the solid–liquid separation characteristics of the leached clay ore followed by an effective process for dealing with the manganese in the pregnant leach solution via the direct solvent extraction process. This process allows for the removal of zinc and cobalt and their separation from manganese, magnesium and calcium.

The process plant is being designed to produce and treat 3.1 million t/year of plant feed at an average head grade 2.2% Cu, 0.1% Co, 0.6% Zn and 2.2% Mn through an integrated hydrometallurgical facility to produce the following products and tonnages:

- Up to 60 000 t/year of copper cathode;
- Up to 3100 t/year of cobalt cathode;
- Up to 36 000 t/year of zinc sulphate monohydrate salt.

In addition, there is the potential to produce up to 100 000 t/year of a manganese carbonate product via the relatively simple addition of a small number of unit operations to the current flowsheet. Annual uranium production would be of the order of 62 tU, assuming a level of 20 ppm U in the ore [64].

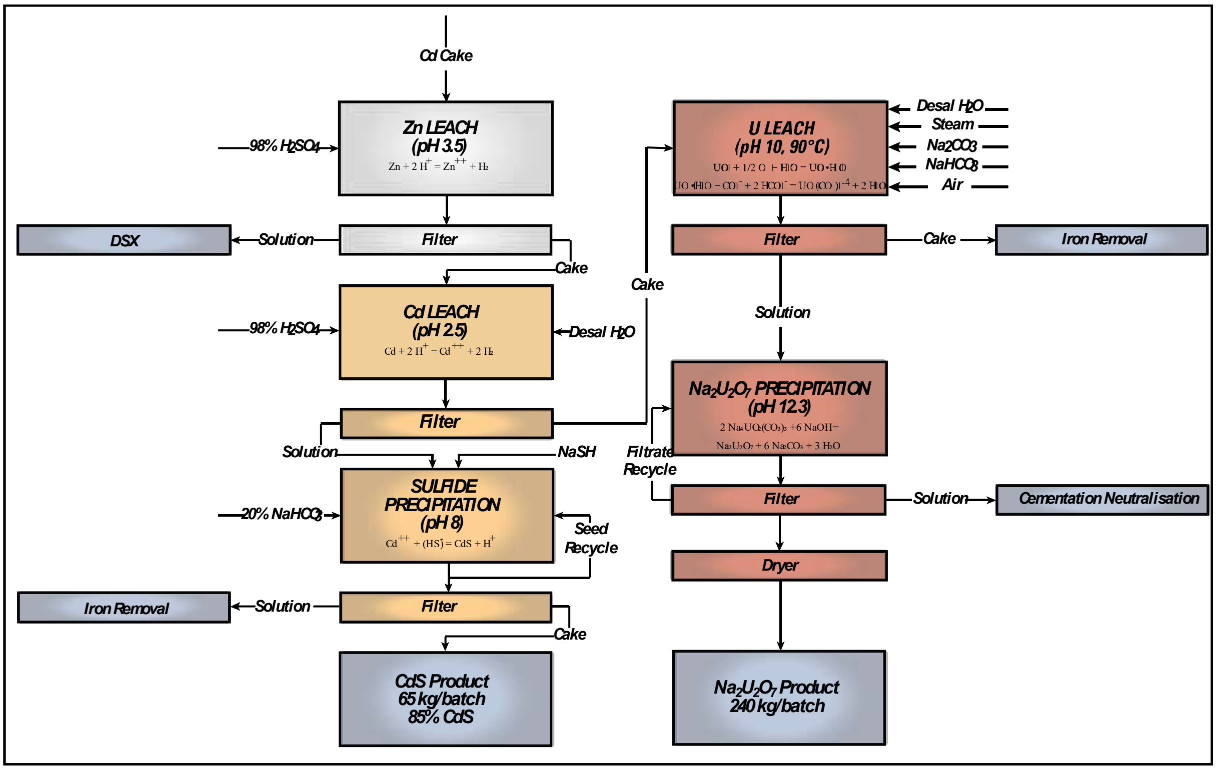

FIG. 39. Proposed uranium flowsheet at Boleo [64].

4.4. METAMORPHITE DEPOSITS (CU–U)

4.4.1. Rakha, Surda and Mosaboni (India)

4.4.1.1. History

Rakha, Surda and Mosaboni are three historical copper mines located in the East Singhbhum district of Jharkhand, India. Each of these mines, in addition to copper, produced small quantities of uranium concentrates from the 1970s up to about 2000.

A British company, the Indian Copper Corporation Ltd (ICC), established the Mosabani mines and crushing plant and the Moubhandar smelter plant in 1928. In 1972, ICC was taken over by Hindustan Copper Ltd. The Mosabani concentrator plant was commissioned during 1973–1974 with an ore capacity 2000 t/d.

Mines started to shut down in the 1990s owing to the fall in copper price. Surda was the last to close in 2003. In 2007, India Resources Limited refurbished the Surda copper mine and Mosabani concentrator. Copper production resumed from the mine and plant in November 2007, but no information is available regarding any plans to resume uranium production.

4.4.1.2. Geology

The Surda copper mine is located in the country’s prime copper producing belt known as the Indian Copper Complex, which is a zone of major thrusting and later extensional shearing and forms part of the Singhbhum shear zone. The Singhbum shear zone is composed of Proterozoic rocks to the east, which overly the Archaean rocks of the Singhbhum Craton to the west. Surda is one of several copper deposits which have been mined since ancient times and which lie along the Singhbhum shear zone.

It should be noted that the Singhbhum belt is also host to some of India’s most significant uranium deposits, including Jaduguda, Bhatin and Narwapahar. Dahlkamp [26] classified these deposits as sub-unconformity epimetamorphic types hosted in albitized metasediments and made the following observations concerning uranium mineralization in the Singhbhum Belt:

- “Mineralization is largely confined to Singhbhum Group metasediments and is spatially associated with Dhanjori volcanics;
- Host rocks are mainly biotite-rich and/or chlorite-rich (+/- apatite and magnetite) rocks which originated from greywacke (Surda–Bhatin sector) or mafic volcanics;
- Uraninite with low Th and high Pb contents is the principal U mineral;
- Orebodies are tabular and generally concordant with secondary intraformational structures.”

According to Bhasin [65], uranium mines of the Singhbhum thrust belt contain small quantities of copper, nickel and molybdenum, which are recoverable by flotation as by-products. A combined copper–nickel concentrate containing about 20% CuS has been produced and sold to a copper smelter and a molybdenum sulphide concentrate has been produced, converted to ‘ferromoly’ and sold to an ordinance factory.

While the Surda–Rakha mines sector is important for copper, ore lenses may contain as much as 0.067% U. Uranium mineralization occurs in close association with copper orebodies, but is not necessarily coincident. The host rock for both copper and uranium is chloritic quartz schist with apatite, magnetite, biotite and tourmaline [66]. In the IAEA’s 2013 classification scheme [1–3], the Surda, Rakha and Mosabani deposits are classified under the Metamorphite type, structurally-controlled subtype.

4.4.1.3. Operations

The old Rakha copper mine, along with the Surda copper mine, are part of the Singhbhum shear zone hosted copper deposits. The old Rakha mine workings extend over a strike length of 1300 m and to a depth of 200 m (mining level 7). The mining levels are 35 m apart. The site lies within a forest reserve and owing to environmental constraints it has never been mined using open pit methods. The mine was accessed and developed by inclines and vertical shafts with winzes. The previous mining operation was suspended in July 2001. The mine is now flooded and the water level is approximately 30 m from the surface [67].

The Surda mine has a strike length of 2200 m and a maximum depth of 450 m. The width of the orebody varies from several metres up to 60 m as the copper mineralization pinches and swells. The orebody is made up of four distinct lodes within a shear zone of quartz biotite schist that dips 30–35° E and sometimes coalesces to become laterally extensive [68].

Access to the Surda mine is via shaft #3 (a vertical shaft and sub-incline system) and shaft #4 (an inclined shaft).

Uranium recovery plants at Rakha, Surda and Mosaboni came into production in the 1970s and 1980s and recovered uranium from mill tails from copper concentrators. Following extraction of copper by flotation, the tails, with an average content of about 0.008% U, were sent to uranium extraction plants operated by Uranium Corporation of India Limited. A uranium-bearing heavy mineral concentrate was produced at these plants by ‘tabling’ (Wilfley) the tails. The concentrates were transported by truck to the Jaduguda mill for further processing. The combined production of the three plants amounted to 150 t/d of concentrates [69]. Table 18 presents unannotated information on beneficiation of low grade uranium ores in India.

TABLE 18. SHAKING TABLE BENEFICIATION OF LOW GRADE INDIAN URANIUM ORES USING AUTOMATIC SPLITTER CONTROL

Ore	Product	Weight (%)	Grade (% U)	Distribution (%)
Jaduguda	Concentrate	32.0	0.127	70.2
	Tails	68.0	0.025	29.8
	Feed	100.0	0.057	100.0
Rakha	Concentrate	8.2	0.137	77.0
	Tails	91.8	0.0034	23.0
	Feed	100.0	0.013	100.0

Given the concentrate grade at Rakha of 0.14% U and a combined output for the three plants of 150 t/d, the uranium production capacity of the three plants could be of the order of about 50 tU/year.

Production details are not available for most of India's uranium mines and mills. Certain comments are, however, available from published sources, as follows:

- "While the production of uranium mineral concentrates at the Rakha uranium recovery plant was above the target, the production at Surda and Mosaboni plants was marginally less than the target because of erratic power supply" [70];
- "The uranium mineral concentrates produced at the Surda and Rakha Uranium Recovery Plants were below the targets as the power supply at these plants continued to be highly erratic and the plants could be operated only for about 50 per cent of the available time. As the feed grade at both these plants also came down, the resultant grade of mineral concentrates was lower" [71];
- "The expansion of Mosaboni Uranium Recovery Plant was completed in July 1988. Provision was also made in the expanded plant for carrying out test works on recovery of uranium from ultrafine particles. Modernization of Surda Uranium Reovery Plant was also completed during the year and the plant has been recommissioned from March 1988. The modernization will help in increasing production, reducing costs and saving power" [72];
- "The Corporation has three uranium recovery plants based on the copper tailings at Surda, Rakha and Mosaboni (Bihar). In 1986-87 the production of uranium mineral concentrate rates from the recovery plants at Surda and Rakha were much higher than that of the last year. There was significant increase in capacity utilization. For the plant at Mosaboni which commenced operations from December 1986, full scale production is available from 1987-88 onwards" [73];
- "The three uranium recovery plants based on copper tailings functioned normal [*sic*] and production of uranium mineral concentrates were higher than previous year[s] in [the] case of Musaboni and Surda plants. The production at Rakha plant was slightly less than last year but more than the target fixed for the year" [74];
- "The production of uranium mineral concentrates from the three uranium recovery plants exceeded the target set for the year" [75].

4.4.2. Lumwana (Zambia)

Copper and uranium mineralization co-exist at the Lumwana copper project in Zambia. The Malundwe and Chimiwungo deposits are structurally-controlled, disseminated copper sulphide deposits of the Central African Copperbelt type. Copper mineralization within these deposits is hosted almost entirely within high grade metamorphosed, intensely mylonitized, recrystallized muscovite–phlogopite–quartz–kyanite schists along with disseminated sulphides (typically less than 5%) dominated by chalcopyrite and bornite. The UDEPO classification of this uranium deposit is metamorphite.

Malundwe is the smaller deposit, but with a higher copper grade and contains discrete zones of uranium and gold mineralization. Chimiwungo is a much larger deposit that is lower in copper grade, but which contains some uranium mineralization.

Initial project development plans promulgated by Equinox Minerals included recovery of both copper and uranium as separate ore streams in separate plants [76]. The 2006 initial resource estimates for the Lumwana project are given in Table 19.

TABLE 19. INITIAL LUMWANA RESOURCE
(as of 2006)

Copper, cobalt and gold resource				
Classification	Ore (10^6 t)	Cu (%)	Co (ppm)	Au (g/t)
Measured	129.5	0.89	238	0.03
Indicated	228.7	0.68	153	0.02
Inferred	564.4	0.63	46	0.01

Uranium resource			
Classification	Ore (10^6 t)	Grade (% U)	Contained tU
Indicated	9.5	0.078	7 465
Inferred	2.6	0.035	783
Total	12.1	0.068	8 248

The initial mine plan developed by Equinox envisaged that uranium ore would be mined concurrently with the mining of copper ore from within the existing Malundwe and Chimiwungo pits and stockpiled in a dedicated facility. Following commencement of production from the dedicated and stand-alone uranium processing plant, the stockpiled uranium ore would be reclaimed and trucked along a dedicated haul road to the plant. Neutralized tailings would be stored in a separate uranium tailings storage facility with tailings return water reused in the plant. Existing infrastructure (including waste rock dumps, raw water, potable water, housing, access roads, power supply, etc.) can be used and supplemented.

The plant design uses conventional milling and flotation to produce copper concentrate for transport and sale to a copper smelting and refining facility. The plant flotation tailings report to the uranium leach circuit, which uses conventional acid leach, solvent extraction, precipitation and calcination to produce uranium oxide.

The uranium ore is proposed to be processed through the plant at a rate of 1 million t/year to recover approximately 910 t/year of uranium oxide (U_3O_8) and 15 000 t/year of copper concentrate. Uranium recovery to uranium oxide is estimated to be 93% and copper recovery to the plant copper concentrate is estimated to be 80%.

Barrick Gold Corporation acquired the project in 2011 and has elected not to recover uranium, preferring instead to discard or blend down uranium to acceptable levels in its copper concentrates [77]. Copper production is ongoing.

4.5. PROTEROZOIC UNCONFORMITY (U–AU)

Proterozoic unconformity deposits appear to be one geological type of deposit where potential by-products of uranium production are relatively common. This type of deposit is preferentially located in two major districts: the Athabasca Basin (Canada) and the Pine Creek Orogen (Australia).

The Athabasca Basin hosts the Cluff Lake, the Key Lake, Midwest and Shea Creek deposits. High grade mineralization at Key Lake averaged about 4.24% U and 4% Ni. Key Lake is depleted, but its complex metallurgy has, to date, prevented the economic recovery of the by-product nickel. Unconformity mineralization at Midwest was reported to average 10% U and 9% Ni. Midwest is undeveloped.

AREVA/UEX Corporation reported that locally high grades of gold are present in the undeveloped Shea Creek deposits, which are located south of Cluff Lake. The high gold grades frequently, but not always, occur in areas of higher grade uranium mineralization and are known to be present both in unconformity and in basement mineralization [78]. Significant composited gold-bearing intercepts include: 13.75 g/t Au over 2.50 m; 14.02 g/t Au over 3.30 m; 20.79 g/t Au over 2.40 m; 9.70 g/t Au over 3.50 m and 5.95 g/t Au over 5.70 m.

The Pine Creek area of Australia's Northern Territory is host to two major undeveloped uranium deposits with by-product gold potential: Koongarra and Jabiluka. At Koongarra, uranium occurs in quartz–chlorite schists and in graphite schists of the Cahill formation. Mineralization includes uraninite with pyrite and traces of chalcopyrite and galena. Uranium reserves at the Koongarra 1 deposit are estimated to be 12 300 tU grading 0.68% U and 3.1 t Au grading 3 g/t Au. The in situ ore value at 2015 prices would be about US $600/t for uranium and US $110/t for gold. Future development of the Koongarra deposit is seen, at this time, as problematic owing to its location within Kakadu National Park, a World Heritage site. Its development is opposed by traditional owners of the Kakadu region.

Gold mineralization at Jabiluka occurs in graphite horizons in the western part of the Jabiluka No. 2 orebody. The gold occurs mainly in breccia zones of the Main Mine Series, where the ore thickness averages 2 m. The gold zone contains 2.392 million t ore averaging 3.7 g/t Au and 0.4% U. In situ ore values at 2015 prices equate to about US $350/t for uranium and US $140/t for gold.

4.5.1. Cluff Lake (Canada)

4.5.1.1. History

Around mid-1967, Amok Ltd was conducting airborne radiometric surveys in three separate locations within the Athabasca Basin. The Carswell dome was selected as a prime target area because of obvious structural deformation of the Athabasca Formation. Several low intensity radiometric anomalies were detected within the structure near Cluff Lake in late 1967. Follow-up ground evaluation of the anomalies resulted in the discovery of high grade mineralized boulders in 1968.

Subsequent detailed geological exploration by Amok, including core drilling, led to the discovery of the high grade D zone deposit in 1971. Exploration continued and by early 1975 several more important orebodies were discovered. Two of the largest were designated as the 'N' and the 'Claude' deposits, the final deposit being designated the 'O-P' orebody.

By early 1975, Amok had determined the presence of large quantities of uranium in four ore deposits at Cluff Lake. Plans for project development were initiated almost immediately and on 24 November 1976, Amok submitted an environmental assessment and safety report for the project to the Saskatchewan Department of Environment.
On 1 February 1977, the Cabinet of Saskatchewan appointed a commission to conduct an inquiry into all aspects of the project (the Cluff Lake Board of Inquiry).

The Board submitted the final report to the Cabinet on 31 May 1978, and the Board's recommendation to proceed with the project was accepted by the Government. Negotiations between the Department of Northern Saskatchewan and Amok proved satisfactory and in September 1978, the Department of Northern Saskatchewan approved a surface lease giving Amok the right to begin project development. The D orebody was mined out by September 1981, well ahead of the original schedule.

4.5.1.2. Geology

Uranium mineralization at Cluff Lake is found primarily in the quartzitic gneisses, red granitoid gneisses and sandstones of the Athabasca Formation. The first ore deposit found by Amok was the unusually high grade D deposit. This deposit contained massive pitchblende up to nine metres thick along the unconformity of an overturned block of the Athabasca Formation. Mineralization was concentrated in the Athabasca pelites and quartzitic gneisses of the regolith zones. The ore was confined to a zone with dimensions of about 140 m × 24 m. Uraninite was the major ore mineral and was associated with carbonaceous organic matter in the sediments. The deposit also contained several accessory minerals, but primarily native gold and gold tellurides, selenium and nickel. The reserves in the D orebody are given in Table 20 [11].

TABLE 20. CLUFF LAKE D OREBODY RESERVES

Cut-off grade (% U)	Ore reserves (t)	Average grade (% U)	Reserves (tU)
0.3	73 660	7.0	5 156
5	26 400	16.8	4 420
20	8 702	29.3	2 550

4.5.1.3. Mining

The D orebody was mined by open pit methods. The first stage involved stripping off about 6 m of overburden which consisted mostly of loose sand and gravel. Only a few isolated remnants of bedrock covering the ore had to be drilled and blasted. All waste material was stacked in two piles away from the mine site and was used for backfilling the pit on completion of mining.

As ore was removed from the pit, it was graded for uranium content and placed on one of three stockpiles. Ore grading 0.25% U or less was deposited on stockpile A, material grading 0.25–16.9% U went onto stockpile B and all ore grading above 16.9% U went onto stockpile C. The ore in stockpile C was held in a concrete structure and covered with a metre of sand to minimize the effects of radiation. The sand was removed sequentially as ore was required for mill feed.

Special precautions were developed to protect employees during the mining process which were unique to the D orebody. High grade portions of unmined ore were covered over with a metre of sand until that portion was scheduled for mining. Then, ore was continuously mined in 2.4 m × 2.4 m blocks, the direction of excavation moving from north to south. A 2.4 m × 2.4 m × 2.4 m cube of ore was mined at a time. The sand shield was removed immediately ahead of the mining face and replaced over the exposed ore layer after the cube was mined out. All mining equipment and personnel worked off the sand cover above and behind the active ore face, thereby minimizing direct contact with the ore. Through careful synchronization of the mining and shielding procedures, only the ore cube being mined was exposed at any given time.

4.5.1.4. Milling

The Phase I processing plant was designed to process medium to high grade ore from the D orebody. The capacity of the mill was about 90 t/d of ore and the average processing rate was about 82 t/d of ore at an average grade of 5.76% U.

Ore delivered to the mill was handled according to grade. High grade (>20% U) ore went to the primary crusher where it was reduced to fragments <100 mm in size. It was then transferred to the high grade storage bins. Medium grade ore which averaged 4% U was crushed separately to produce particles of less than <12 mm diameter, which was passed through a gravity concentration process to produce a concentrate containing about 45% U and a tailings fraction containing about 0.3% U. The concentrate was transferred to storage bins and the tailings fraction, or gravity residue, was transferred to a stockpile for Phase II processing. Gravity concentrate and high grade ore were crushed and stored under two storage domes.

High grade ore and concentrate were passed to a conventional rod mill for further comminution. Well blended ores were then subjected to acid leaching during which all minerals contained in the ore were dissolved. The solution was then treated in a two-stage precipitation and filtration process to remove so-called ‘nuisance’ minerals containing iron, aluminium, molybdenum and thorium. The uranium-bearing solution was then treated with magnesium oxide which precipitated magnesium diuranate (yellowcake), which was then washed and dried for packaging.

Leach tailings from the initial milling process amounted to 6500 t and contained 0.25–0.85% U and 58 g/t gold. The tailings were stored in concrete bins for future processing to recover both uranium and gold.

Gold recovery

Gold recovery was affected by conventional cyanide leaching using a 72 h retention time at a rate of about 20 t/d of feed. A carbon-in-pulp process utilizing activated coconut husk concentrated the soluble gold to about 8000 g/t of carbon. Cyanide in the gold leached pulp still containing uranium was volatilized, recovered and recycled. Any remaining uranium was recovered in the Phase II uranium plant. Gold was stripped from carbon using a hot (95°C) solution of strong caustic and sodium cyanide and loaded onto stainless steel wool by electro-winning. Doré was produced by mixing flux with the stainless-steel wool followed heating in an electric furnace. The plant operated for about two years (1988–1990) and produced about 350 kg of gold.

Capital costs for the gold recovery circuit were C $2.6 million. Considering an average gold price for the period of operation of about US $12 860/kg, the total value of recovered gold amounted to some US $4.5 million.

4.6. COLLAPSE BRECCIA PIPE

Collapse breccia pipes are sub-vertical, chimney-like structures, 30–175 m in diameter and up to 1000 m in length [79]. They are formed by progressive collapse, upwards, of overlying strata into a cavity which has been typically formed by dissolution of limestone. Such pipes are numerous, and in the Grand Canyon region of Arizona, in the USA, they could number in the thousands. Only a few of these pipes, several dozen perhaps, contain economic or potentially economic concentrations of uranium. At least two of these pipes, Orphan Lode and Canyon, also contain economic concentrations of copper.

4.6.1. Orphan Lode (USA)

Discovered in the late 1800s, Orphan Lode was first mined sporadically for copper. The black sandstone accompanying the copper ore was discarded. In 1951, however, this black sandstone was determined to be high grade uranium ore. Notably, the Orphan Lode was located on an old patented mining claim (private property) within the Grand Canyon National park. Reasonably good relations were maintained between the mining company and the US National Park Service throughout the life of the mine.

Uranium ore from the Orphan Lode was first produced in 1956 and the mine operated continuously through 1969, producing a total of 454 000 t of ore, from which 1656 t U were recovered at a grade of 0.36% U. Copper (3006 t) and silver (3425 kg) were also recovered as by-products [80].

4.6.1.1. Geology

The Orphan Lode breccia pipe is located within the south wall of the Grand Canyon, cropping out approximately 300 m below the canyon rim. The mined and explored portion of the pipe extends vertically through the Coconino Sandstone (Permian) and the Hermit Shale (Permian) and into the Supai Formation (Permian/Pennsylvanian) for a total distance of 180 m. Over this distance, the diameter of the pipe varies from 46 m to 113 m. As with most pipes in the region, it is assumed to bottom–out in the Redwall Limestone, which is Mississippian in age [80].

Uranium–copper–silver ores occurred both in the centre of the pipe as well as in annular rings adjacent to, and external to, the pipe–wall contact. Mineralization within these ores included uraninite, coffinite, pyrite (by far the most abundant sulphide), argentiferous chalcocite, tennantite–tetrahedrite, arsenopyrite with minor chalcopyrite and bornite. Most minerals are extremely fine grained. High grade uraninite was reported to be associated with earthy haematite. Ores from the annular rings were characterized by their high uranium (up to 35–60% U) and high lime (>25%) contents. The more central ores averaged about 0.50% U and 3% lime. Ores were blended to maintain a shipping grade of about 0.34% U and 6% lime.

4.6.1.2. Mining

Mining of the Orphan Lode was complicated by its location in the Canyon wall. Initially, ore was transported to the Canyon rim by a 500 m long aerial tramway. This tramway was later replaced by a 460-m vertical shaft. All ore was mined by conventional underground methods. So-called ‘annular ore’ was mined primarily with jackleg drills and air-powered slushers and ‘central ores’ were mined within a ‘glory hole’ stope that could be drilled from the perimeter. This central stope was 130 m high, 23 m wide and 115 m in length. Target output for the mine was 225 t/d.

4.6.1.3. Milling

Uranium–copper–silver ore from the Orphan Lode was milled at the between 1957 and 1966 at the Tuba City mill, also in Arizona, the ore being trucked 160 km to the mill. Originally an acid leach mill, it was converted to include an alkaline process in 1962 to accomodate the high sulphide and high lime ores from the Orphan Lode. After grinding, ore was fed to a flotation circuit; flotation concentrate (sulphide) was directed to an acid leach complex and flotation tails directed to an alkaline leaching process. Sulphide leaching incorporated a three-stage counter-current decantation process followed by rake classifiers and filters to produce a sulphide concentrate for shipment to a copper smelter where silver could also be recovered. Copper recovery was reported to be 75%. Solution from the acid leach process was treated by solvent extraction to produce a pregnant strip solution suitable for combining with a similar solution from the alkaline leach process. The combined solution was treated with sodium hydroxide to precipitate uranium [80]

In 1967–1969, Orphan Lode ore was shipped to the Cotter Corporation mill in Cañon City, Colorado. This mill also incorporated a flotation circuit to remove and concentrate sulphides.

4.6.2. Canyon (USA)

The Canyon breccia pipe is located on the south side of the Grand Canyon, south of Grand Canyon National Park and about 11 km south of Tusayan. Gulf Mineral Resources Company drilled eight exploration holes at the site from 1978 through May 1982 but only encountered low grade uranium mineralization. Additional drilling completed by Energy Fuels in 1983 identified a major deposit and subsequent drilling through 1994 provided a sufficient basis for feasibility studies and stock exchange qualifying reports.

The project has been subject to intense opposition from local native Americans as well as from numerous anti-nuclear and anti-development organizations.

Surface facilities, including shaft headframe, are in place and the Canyon Mine is currently (2017) under development with shaft sinking and exploration drilling activities being undertaken [81].

4.6.2.1. Geology

The Canyon breccia pipe is essentially vertical and has an average diameter of less than 60 m. It extends for at least 700 m vertically from the Toroweap limestone to the upper Redwall horizons. Uranium mineralization occurs both within the pipe and within an annular ring outside of the pipe extending over a vertical distance of some 520 m.

Sulphide mineralization occurs throughout the pipe but is concentrated in a 6-m thick sulphide cap near the Toroweap–Coconino contact. This cap consists of massive pyrite and bravoite, an iron–nickel sulphide.

Ore mineralization is mainly a uranium–pyrite–haematite mix with massive copper sulphide mineralization common within and near the ore zone. The annular fracture zone in the lower Hermit upper Esplanade horizons seems to contain the highest grade of mineralization. In October 2016, Energy Fuels, Inc., the current owner/operator, announced that exploration results to date indicated high grade copper ore averaging 8.75% Cu.

In 2007, inferred uranium resources for the Canyon breccia pipe were estimated, based on limited surface drilling, to total 83 000 t of ore at an average grade of 0.83% tU, which equates to approximately 627 tU [82-83].

4.6.2.2. Mining

While not yet determined in detail, it is anticipated that mining methods will likely be centred on long hole stoping for extracting centre–core mineralization and jackleg/slusher stoping for extracting ore within the annular fracture zone [82].

4.6.2.3. Milling

It is planned that ore from the Canyon mine be hauled by truck 520 km to the White Mesa mill located near Blanding, Utah.

White Mesa is an acid leach mill with a licensed capacity of over 3000 tU/year. It has treated a very wide variety of uranium-bearing mineralization, including substantial quantities of high grade ores from breccia pipes in the Grand Canyon region. A more complete description of the milling process at White Mesa can be found in the chapter below dealing with uranium–vanadium.

4.7. SANDSTONE DEPOSITS

4.7.1. Uranium–vanadium: Northern Colorado Plateau (USA)

4.7.1.1. History

The history of uranium mining in the tabular sandstone deposits of the Northern Colorado Plateau (‘the Plateau’) district dates to 1898, when about 9 t of ore were mined and sold for $2600 to Charles Poulot, a French chemist. The ore came from the Rock Creek area, which is now included in the Uravan Mining District in Colorado. The ore was exceedingly high grade, averaging 15.43% U and 11.4% V_2O_5. Although prospecting increased and several new uranium occurrences were discovered, a viable mining industry did not develop until about 1910. It was then that Plateau ores became a major source of radium. World demand for radium was satisfied primarily by ore mined in the Plateau, until the discovery of high grade pitchblende in the former Belgian Congo in 1922. As a result, uranium mining on the Plateau declined sharply [84].

Limited mining, largely due to industrial demand for vanadium, persisted from the late 1920s until about 1944. By then, eight vanadium mills, with a combined capacity of about 720 t/d of ore, were operating throughout the district. The grade of the ore averaged 1.14% V and 0.21% U. Usually, uranium was discharged into the mill tailings. During the period 1945–1947, the US Government sponsored the recovery of uranium from some tailings sites to supplement its wartime procurement programme to supply the Manhattan Project.

The US Atomic Energy Commission (AEC) was established in late 1946 and initiated an ore buying programme in 1948, which immediately gave impetus to a new episode of uranium prospecting and mining. Twelve ore buying stations were built in the Plateau district to accommodate miners and the second mining boom in 40 years was in full progress.

During the 1950s, five of the eight vanadium mills were modified to recover uranium, and four new uranium mills began operations. Vanadium became an important by-product. Production details are presented in Table 21. The search for uranium on a national scale was so successful that in 1962 the AEC was forced to curtail ore purchases. As a result, production of uranium began a steady decline, and when the AEC buying programme was officially terminated in December 1970, mining in the Plateau district was limited to supplying ore for continuing operation of the Moab and Uravan mills.

Resurgent uranium prices in the mid-1970s encouraged new uranium development in the form of two new mills: Shootering Canyon and White Mesa. Shootering Canyon operated only for a brief test run and White Mesa became the only operating mill on the Plateau when both the Moab and Uravan facilities closed in 1984. White Mesa continues to operate on a campaign basis, with intermittent runs of Plateau ore, Arizona strip ore and alternative feed.

TABLE 21. URANIUM–VANADIUM OPERATIONS: NORTHERN COLORADO PLATEAU

Facility	Owner/operator	Period of operation	Ore processed (t)	Ore grade (%)		Production		Recovery (%)	
				U	V	tU	tV	U	V
Durango	Vanadium Corp.	1949–1963	1 456 244	0.25	1.18	3 020	12 794	80	70
Durita heap leach	Ranchers Exploration & Development	1977–1979	571 526	0.03	n.a.	146	634	72	n.a.
Grand Junction	Climax Uranium Co.	1951–1970	2 069 845	0.24	1.07	4 496	15 896	93	72
Moab	Atlas Corp.	1956–1984	17 690 099	~0.25	n.a.	24 769	7 079	~96	n.a.
Montecello	AEC	1949–1960	819 458	0.26	n.a.	1 763	808	83	n.a.
Naturita	Vanadium Corp.	1947–1958	539 383	0.25	1.28	1 046	4 669	77	68
Rifle (old mill)	Union Carbide	1947–1958	629 128	0.30	1.21	1 644	5 545	85	69
Rifle (new mill)	Union Carbide	1958–1972	1 816 329	0.19	n.a.	4 718	n.a.	94	58
Shiprock	Kerr-McGee	1954–1968	1 385 441	0.22	0.88	2 855	7 062	94	58
Slick Rock	North Continent	1931–1943	33 566	0.22	2.28	58	518	n.a.	n.a.
Uravan	Union Carbide	1949–1984	8 981 127	0.17	0.88	15 462	71 659	91	n.a.
White Mesa	Energy Fuels	1980–present	n.a.[a]	n.a.	n.a.	7 348	17 236	n.a.	~80
Total						**67 325**	**143 900**		

[a] n.a.: not available.

4.7.1.2. Geology

The Plateau physiographic province covers an area of about 363 000 km^2. It is bounded to the east by the Colorado Rockies, to the north by the Uinta Uplift, to the west by the high plateau and to the south by the Basin and Range fault block belt of Arizona and New Mexico. The Plateau encompasses, generally, western Colorado, southern Utah, northern Arizona and north-western New Mexico (Fig. 40).

The Plateau geology consists, principally, of predominantly crystalline Precambrian complexes and variably bedded, sedimentary rocks of the Palaeozoic, Mesozoic and Cenozoic eras; Palaeozoic and Mesozoic rocks being the most predominant on the Plateau.

Within the Plateau, the host strata to uranium ores are predominantly sedimentary, ranging in age from Precambrian to Tertiary (Fig. 41). Sedimentary rocks of Mesozoic age are the dominant host rocks of uranium and vanadium deposits on the Plateau. Sedimentary rocks are exposed over large areas and include strata of fluvial, lacustrine, aeolian, volcanic and marine origins. Strata include such lithological units as conglomerates, sandstones, siltstones, mudstones and limestones.

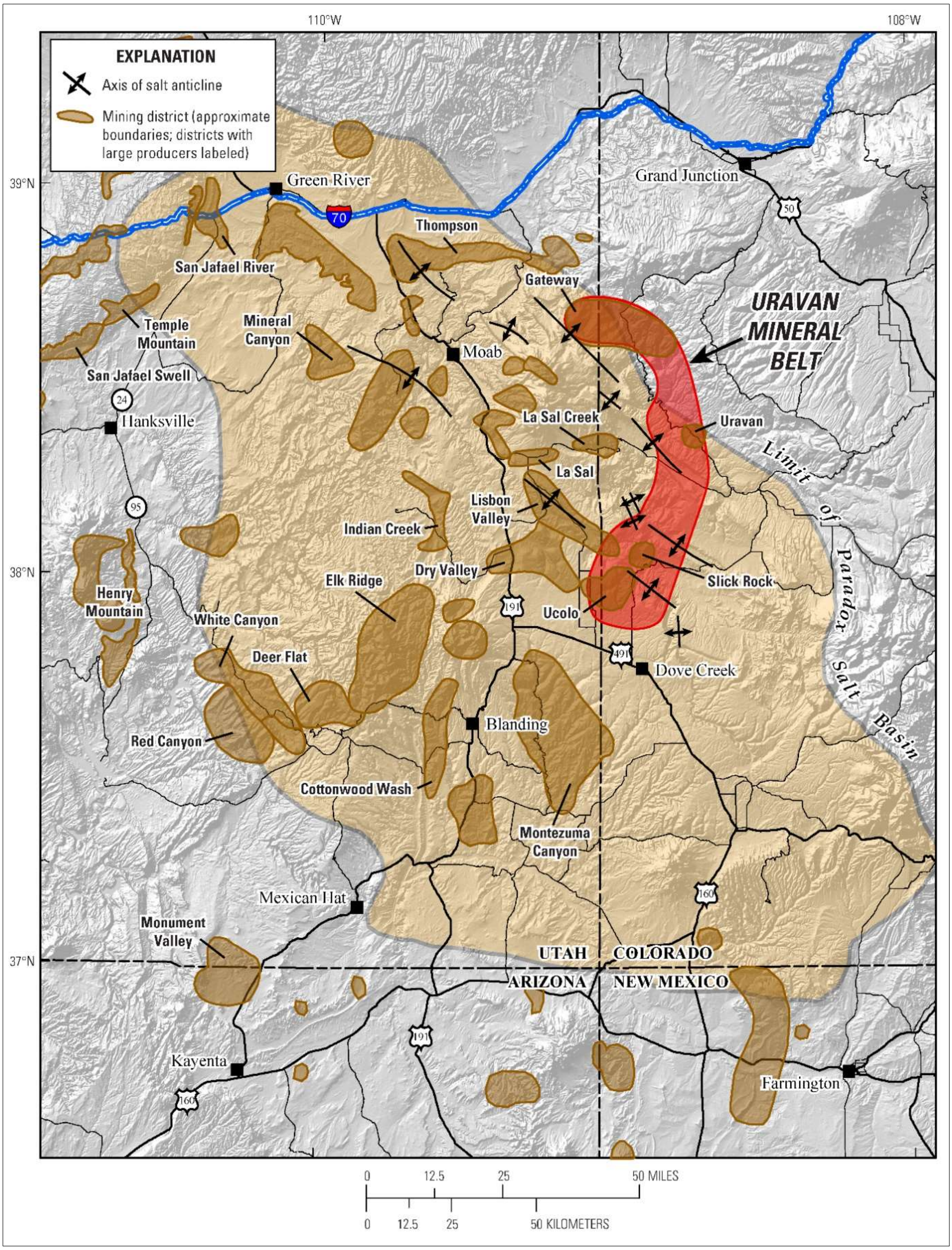

FIG. 40. Map of the Uravan Mineral Belt, showing major uranium districts (adapted and redrafted from Ref [84]).

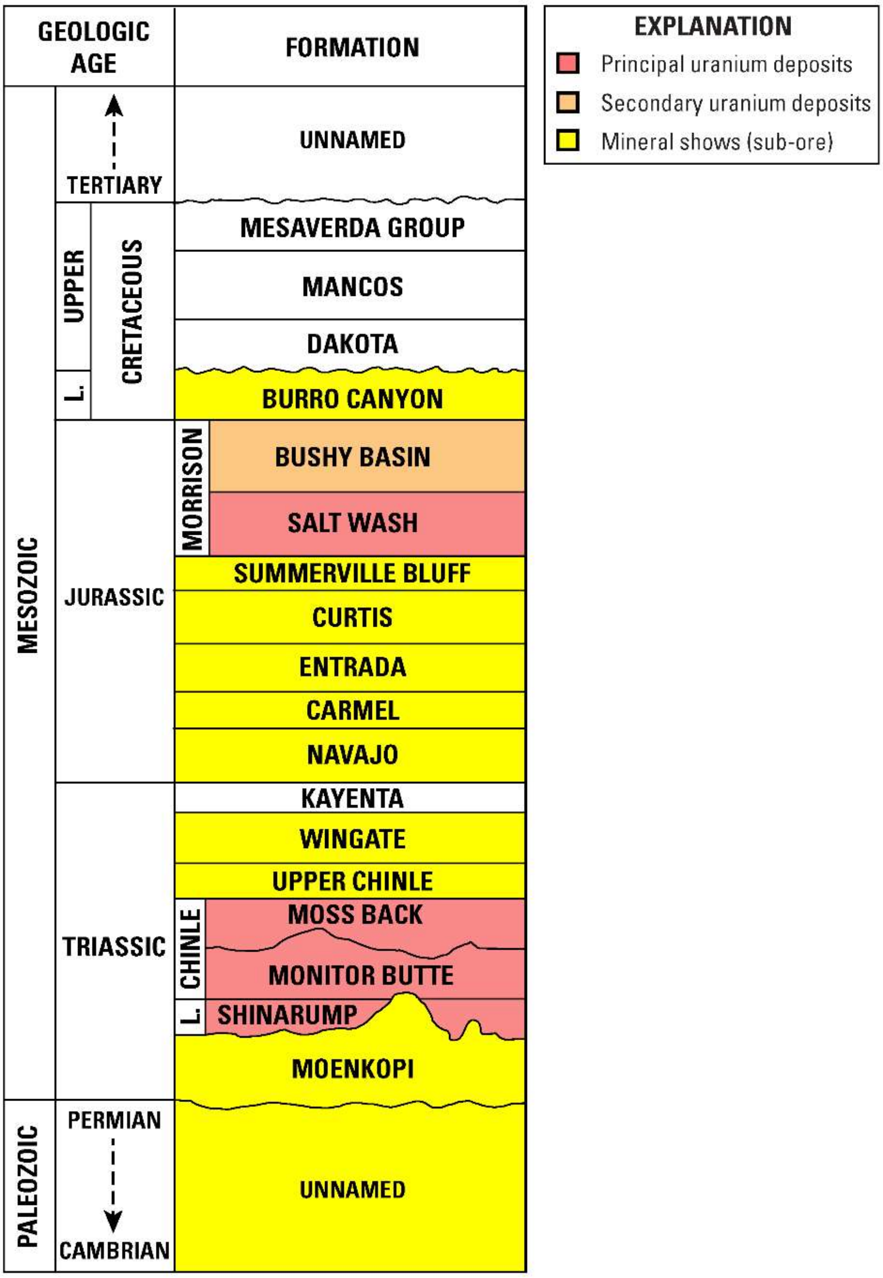

FIG. 41. Generalized stratigraphic section of the Northern Colorado Plateau Uranium District, showing the distribution of uranium occurrences (adapted and redrafted from Ref. [84]).

Structurally, the Plateau has been a region of broad stability, but tectonic activity has occurred at various times. Precambrian exposure is predominantly to the north-east. During the Pennsylvanian and Permian periods, north-west trending uplifts and accompanying downfolds developed across the Plateau. These structures appear to have exercised some control over the later Laramide tectonic events. Indeed, Laramide igneous intrusions tend to align themselves along a north-west trending lineament. The principal Laramide structural features of the Plateau consist of several uplifts and basins; basins constitute one-third of the total area of the Plateau. Most deformation within the Plateau has occurred along monoclines which, generally, lie between the larger uplifts and their adjacent basins. Regional tectonic activity or patterns are considered to have had little direct effect on the distribution of uranium on the Plateau. However, indirectly, tectonic activity has provided favourable conditions, such as the variety of sediments, their depositional environments and the hydrological conditions prevailing which favour uranium deposition.

The Uravan Mineral Belt is an area within the Colorado Plateau in south-western Colorado within which uranium–vanadium deposits in the Salt Wash Member of the Morrison Formation are closer spaced, larger in size and of higher grade than those in adjacent areas and throughout the region (Fig. 41). On the Plateau, the Morrison Formation is divisible into four members. In ascending order, they are: Salt Wash, Recapture, Westwater Canyon and Brushy Basin. The two middle members are absent in the Uravan area. Thus, the Brushy Basin disconformity overlies the Salt Wash Member. The strata, in general, are flat lying. The Morrison Formation in this area comprises light coloured, lens shaped, cross-stratified sandstones that are red–green–grey in colour, and variegated claystones, siltstones and mudstones.

The Brushy Basin is comprised primarily of variegated, red, green and purple bentonitic mudstone strata and lesser amounts of pebbly sandstone. The Salt Wash is characterized by light grey and light brown sandstones interbedded with red and green siltstone and mudstone beds. Sandstones are, on average, fine grained and well sorted to moderately well sorted, although channel sandstones are coarse-grained and poorly sorted. The Salt Wash is considered to have been formed by braided streams emanating from south–central Utah or north–central Arizona.

With few exceptions, orebodies share many of the same characteristics, regardless of their metallic elements, lithological or structural environment, or stratigraphic position. Most of the deposits occurring in clastic fluvial environments are roughly tabular in shape and irregular in outline. Their longer dimensions are parallel to the bedding planes of the enclosing rocks, and the longest dimension of many deposits is approximately aligned with the long axes of the lenses in which they occur. Many of the deposits cut across the beds. The upper and lower limits of most deposits are quite sharply defined, whereas lateral margins may be sharp or indistinct.

Deposits range from small masses only a few metres across to large masses hundreds of metres in length and tens of metres in width. Thicknesses may range from centimetres to metres. A size analysis of 666 deposits in the Plateau resulted in the size distribution seen in Table 22 [85]. Ore grade commonly ranges up to 0.4% U, although smaller amounts of higher grade ore are present in some deposits.

Throughout the Plateau region, orebodies tend to be clustered in groups. Clusters range from only a few hundred metres across to a few kilometres across, and distances between clusters may range from a few hundred metres to several kilometres. The terrain between clusters may contain a few deposits, but most of these tend to be small.

A variety of metallic elements are associated with sandstone type uranium deposits on the Plateau. In certain areas, vanadium and/or copper are of such a grade and quantity as to be of economic importance. In addition, anomalous but generally trace amounts of lead, cobalt, nickel, molybdenum, chromium, arsenic and selenium occur within the sandstone type uranium deposits.

TABLE 22. DISTRIBUTION OF DEPOSIT SIZE: COLORADO PLATEAU [85]

Size of deposit (tons of ore)	Proportion of total (%)
0–100	29
100–3000	41
3000–10 000	16
10 000–25 000	8
25 000–50 000	3
50 000–150 000	3

With one exception in Utah, all uranium produced in the Uravan region has come from underground mines of varying depths. One open pit mine was operated near La Sal in the 1970s. Laccolithic stocks of Laramide age, known as the La Sal Mountains, intrude the area near the small town of La Sal. Their influence on the uranium deposits has been debated over the years, but no final conclusions have been reached.

Most Salt Wash ore production within the Uravan Mineral Belt prior to the 1970s was derived from relatively shallow deposits of carnotite, usually at a depth of 100 m or less. At this time, generally, adits were more common than shafts. However, since that time, production of non-oxidized ore has far exceeded that of oxidized ore. Depths of mines may exceed 200 m and production from shafts is more common. Non-oxidized uranium minerals are uraninite and coffinite, although uraninite is the dominant mineral. Vanadium occurs in several forms, but most commonly as the mineral montrosite.

Ore minerals are typically fine grained and fill the pore spaces of sandstone, impregnate clay pellets and fossil wood, or replace the cell walls of the wood. Commonly, interstitial clay has been replaced or recrystallized and, locally, quartz grains have been corroded. Calcite is probably the most common cementing material in sandstones.

In the Uravan area, three major sandstone units within the Salt Wash are recognized. These are known locally, from bottom to top, as the first rim, second rim and third rim. All larger mines of the area are in the upper sandstone lens (3rd rim) of the Salt Wash. Small mines have been developed in the lower part of the Salt Wash, or in the base of the overlying Brushy Basin.

Deposits are tabular, elongate, or pod-like and, in general, parallel to the bedding plane. Locally, uranium ore crosses the bedding in steeply curved surfaces referred to as ‘rolls’ by the miners. In length, orebodies range from a metre to hundreds of metres, and in thickness from several centimetres to 3–4 metres. Ore may follow the bottom of a channel or sandstone lens, the top of a lens, or it may appear to ‘float’ mid-way in a permeable layer. Sedimentary control of mineralization is common. Vein filling or fracture control is rare in these orebodies. The Lisbon Fault, located south of La Sal in Lisbon Valley, Utah, is a major north trending fault affecting Triassic rocks and uranium deposits therein. It is a normal fault and vertical movement has been calculated as being over 100 m. Salt Wash deposits lying to the north but along the strike of this fault do not appear to be affected. Thickness of ore in the area ranges up to 6 m but is more commonly of the order of 2–3 m. Ore grade may range up to 0.4% U or above, but the average production grade commonly ranges between 0.17 and 0.25% U.

Salt Wash vanadium–uranium deposits in the Uravan area commonly record a V:U ratio of between 5:1 and 10:1.

4.7.1.3. Mining

Individual orebodies in the Plateau are quite variable in terms of size and geometry and may be small and irregular meandering deposits or they may form large lenticular bodies varying in size from a few tonnes to as much as 10 000 t. The mining method requires flexibility as well as selectivity and can be best generalized as an ‘open room with random pillar’ method. Following delineation of an orebody or a mining area, a mode of entry was selected:

(a) *Shaft*: At depths greater than 150 m, vertical shafts tend to be the favoured way of gaining access to the deposit, assuming that no canyon wall is near enough to provide access via an adit. In addition, as mining proceeds, additional shafts for ventilation must be drilled. The shaft entry method provides the shortest distance to the orebodies, although, it has the disadvantage of incurring high, fixed maintenance and operating costs. Haulage might be by rail or trackless rubber-tyred equipment;

(b) *Decline*: As most haulage from the mine face was done with rubber-tyred equipment, a trackless decline provides a non-stop haulage cycle. The gradient capability of haulage units (5–10 t units) limits the decline to a maximum slope of about 12.5%, and consequently this limitation can necessitate long haulage routes thereby increasing entry development costs, although, such costs are offset by increased productivity per person-shift and lower surface capital costs;

(c) *Adit*: If a canyon wall were deep enough to expose the Salt Wash unit hosting the ore and were it to lie in reasonable proximity to the orebody itself, a level adit entry would be the prefered means of access. Development would then be conducted within the ore horizon and this may encounter ore not previously identified. Haulage can be by rail or trackless rubber-tyred equipment.

Regardless of the mode of entry, a separate escape route must be provided for the miners. Ventilation raises, as required, were generally drilled by a raise borer and back reamed to 2.13 m diameter.

Jackleg drills were the main tool for drilling ore. Single unit jumbo drills were used to produce deeper rounds in development headings. An ammonium nitrate–fuel oil (ANFO) mixture was the explosive agent employed and charges were detonated by dynamite cartridge. During the early history of mining on the Plateau, most haulage was done by rail. By the 1950s, trackless rubber-tyred haulage was being developed and increasingly utilized to enhance productivity.

Small units with capacities of 1–3 t such as the ‘Young Buggy’ (Fig. 42) or ‘Getman Scootcretes’ were loaded by front-end loaders and hauled ore from the face to the surface or to a shaft station.

As time progressed, broken ground was loaded and trammed by small (1–2 m^3) load–haul–dump (LHD) units (Fig. 43). As tramming distances lengthened, the LHD units functioned as front-end loaders, dumping their loads into 4–8 t articulated haulage trucks designed for underground use. Split set rock bolts with landing mats and/or chain link fencing provided most of the necessary roof support. Timber or steel sets were rarely used.

FIG. 42. ‘Young buggy’ (courtesy T. Pool).

FIG. 43. LHD machine (courtesy T. Pool).

Data from Deremo mine operations provide an insight into typical productivity. In 1976, some 93 000 t of ore and 82 000 t of waste were removed from the Deremo mine. Mine personnel totalled 115, including 59 miners and their assistants. Production crews were reported to average about 22 t of ore and/or waste per person-shift and overall productivity amounted to 6.3 t per person-shift. Figure 44 is a plan of the underground workings at the Deremo mine, which illustrates the challenge posed by many of the deposits in the Uravan Mineral Belt.

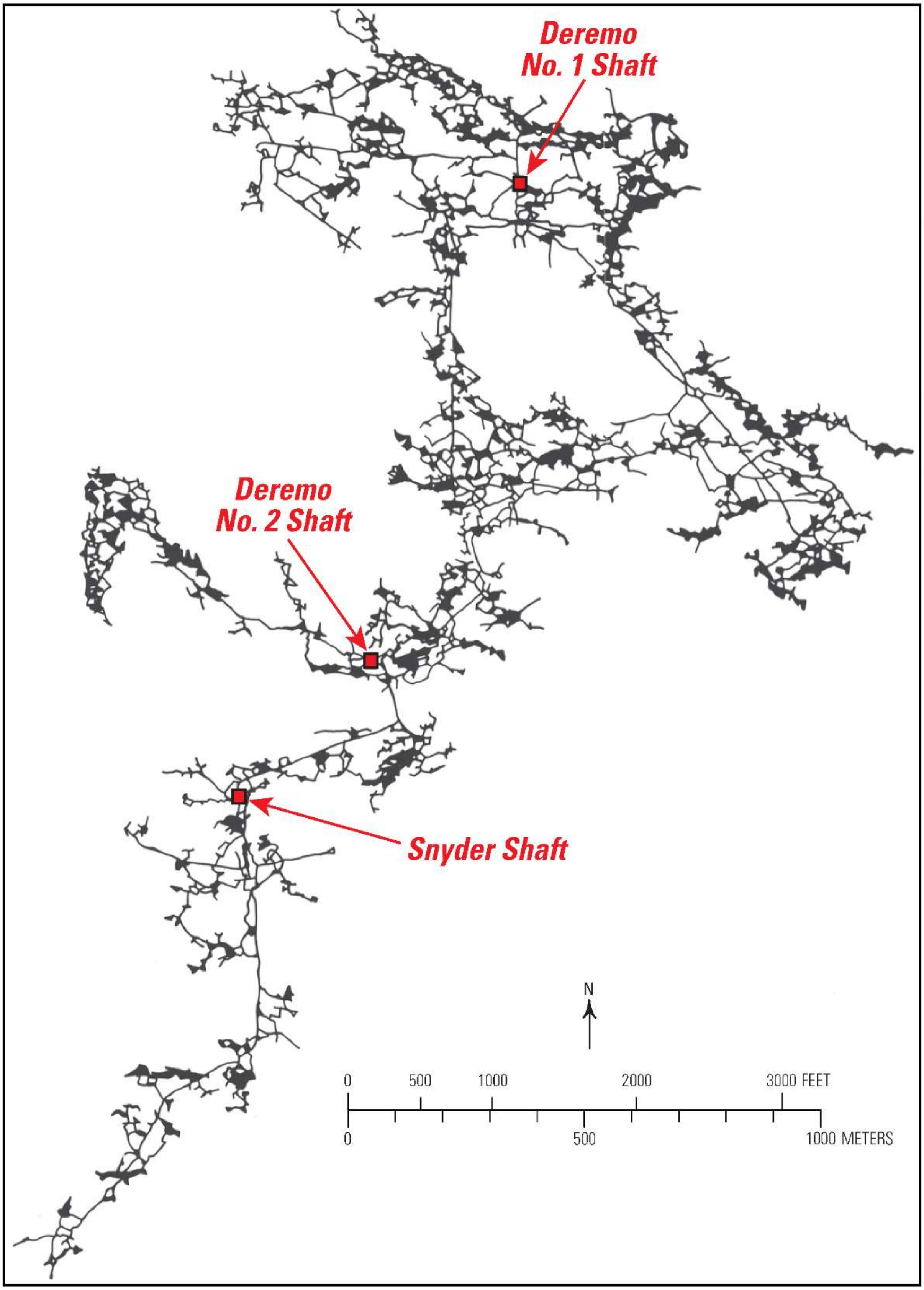

FIG. 44. Plan view of the Deremo mine, Slick Rock area, Colorado (adapted and redrafted from Ref. [86]).

4.7.1.4. Uranium–vanadium price/value relationship

Historical and constant dollar prices for uranium and vanadium are shown in Figs 45 and 46, respectively. By combining the two constant dollar values and applying them to typical ore grades for the Plateau, a perspective on ore value over time can be gained, as shown in Fig. 47.

4.6.1.5. Uravan

The Uravan site was used almost continuously from 1915 through 1984 for the recovery of various metals. Radium was recovered during the period 1915–1923 by the Standard Chemical Company. The Union Carbide Corporation purchased Standard in 1936 and recovered vanadium from 1936 to 1945. Additionally, from 1942 to 1945, under contracts with the Manhattan Engineer District, Union Carbide processed vanadium tailings at the Uravan site to produce a 'green sludge', containing about 17% U. This product was shipped to Grand Junction, Colorado, where Union Carbide operated a Government owned (Manhattan Engineer District) plant which processed the sludge to further concentrate the uranium. These operations ceased in 1945 with the drop-in demand for vanadium and uranium at the end of World War II.

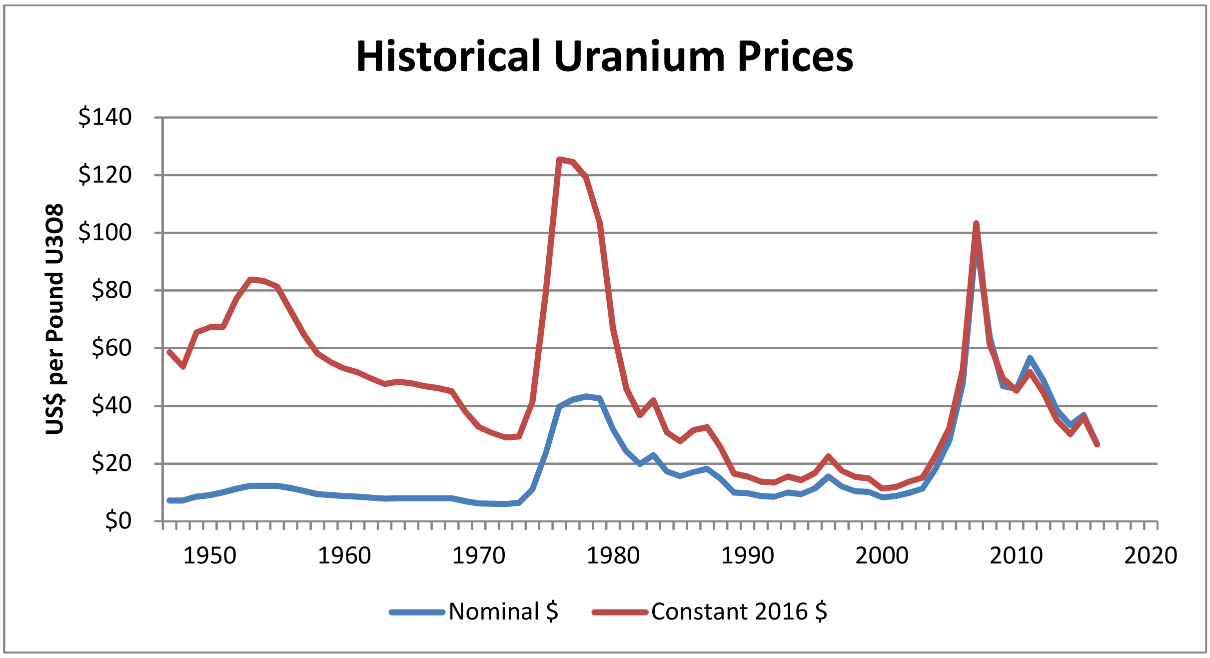

FIG. 45. Historical uranium prices (courtesy T. Pool).

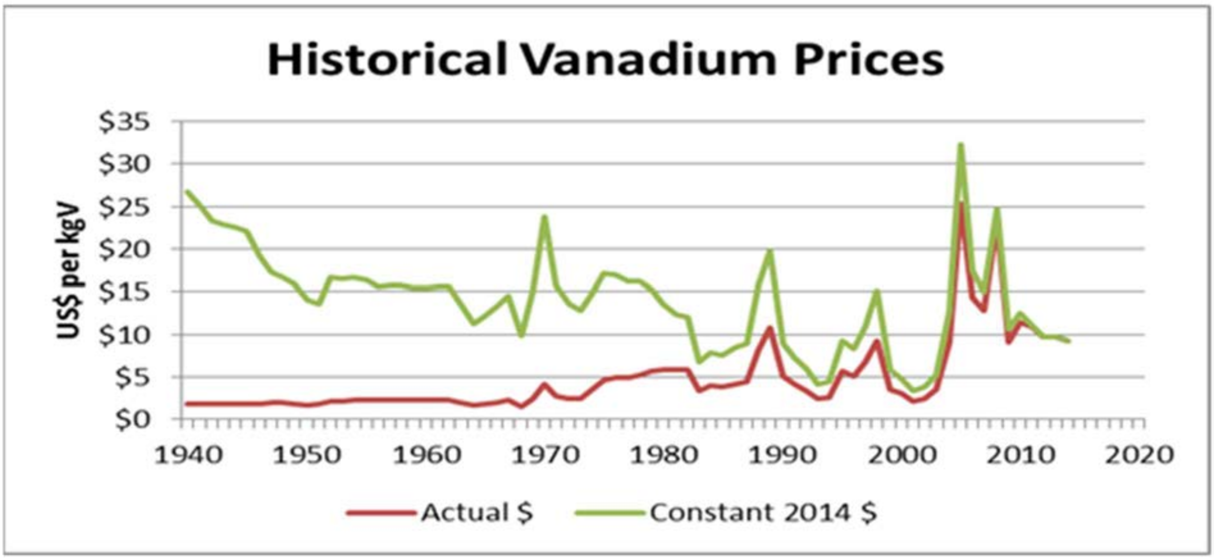

FIG. 46. Historical vanadium prices (courtesy T. Pool).

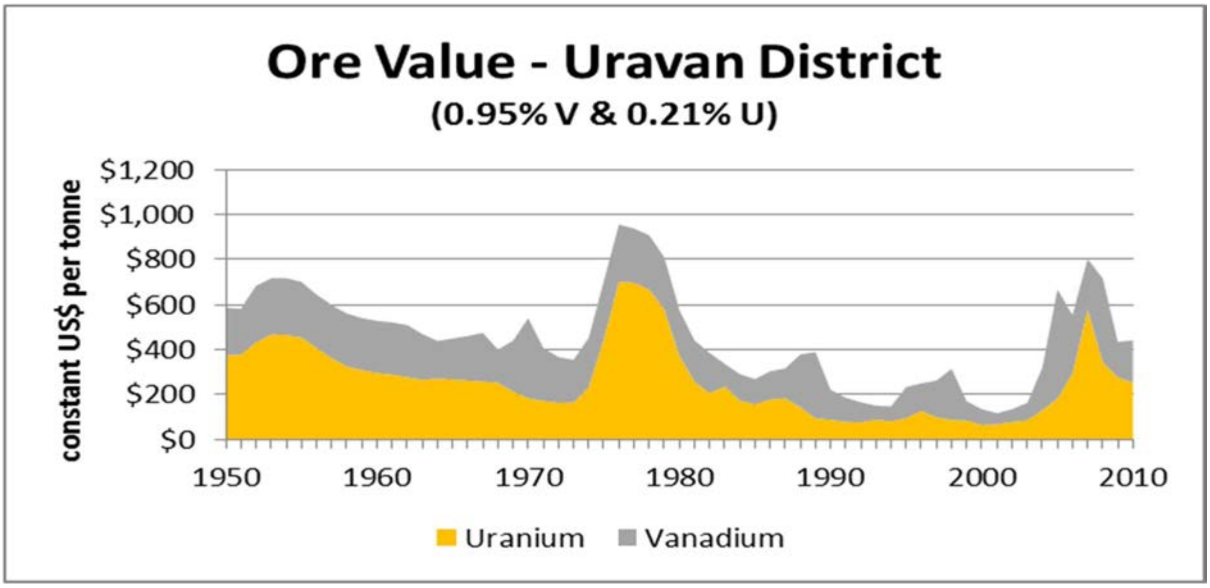

FIG. 47. Ore value of the Uravan District (courtesy T. Pool).

In 1949, after signing a new uranium supply contract with AEC, the Uravan mill was reactivated, using a salt roast, water leach process for vanadium recovery and neutralization and chemical upgrading for uranium recovery, attaining a capacity of about 450 t/d in 1950. In 1955, the new B plant was constructed on the canyon rim at Club Mesa above Uravan, expanding mill capacity to 900 t/d. The B plant used hot acid leaching rather than the salt roast process employed in the old mill. Both uranium and vanadium are taken into solution during the highly oxidizing two-stage acid leach process. Figure 48 is a schematic of the Uravan mill flowsheet.

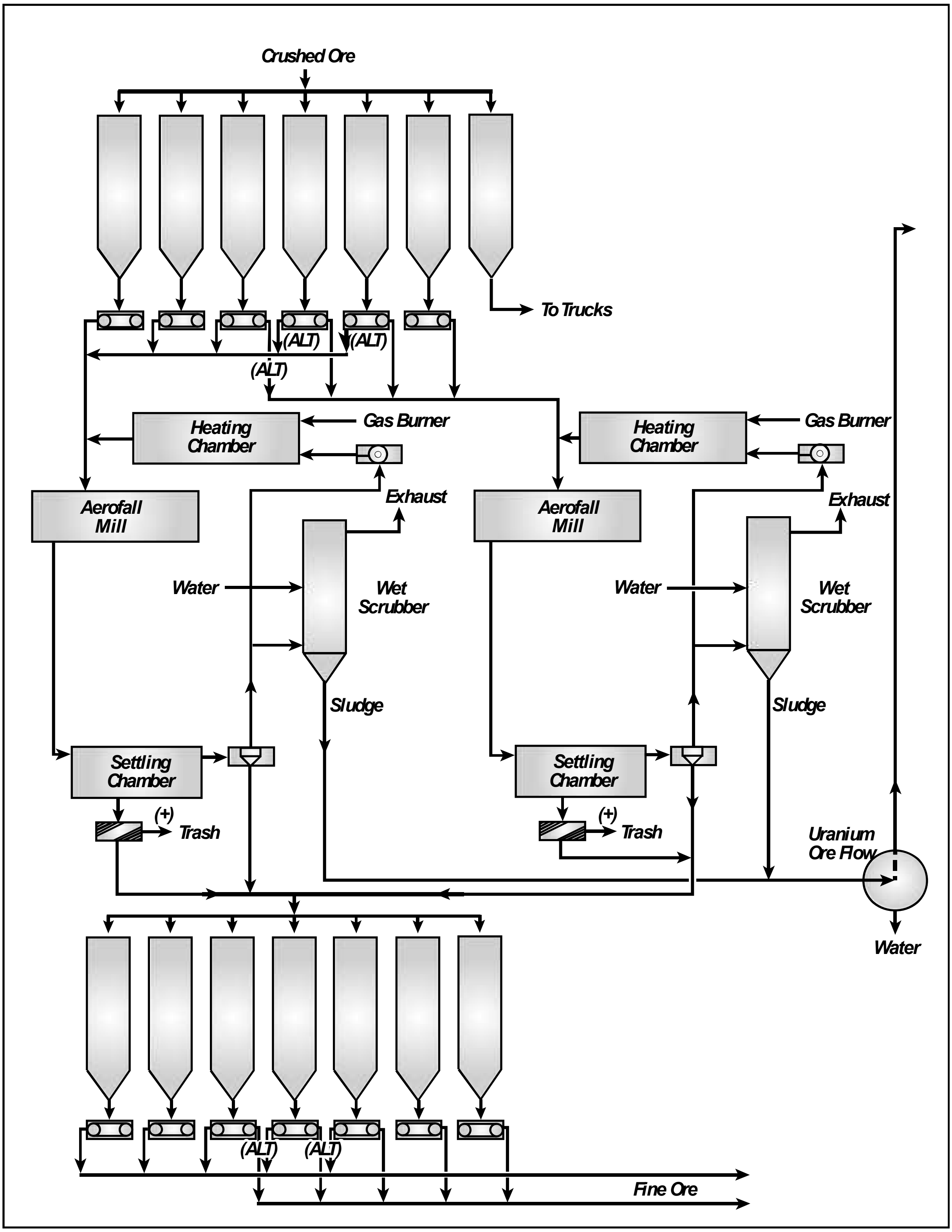

FIG. 48(a). Flowsheet of Union Carbide Corporation's Uravan Mill (adapted from Ref. [87]).

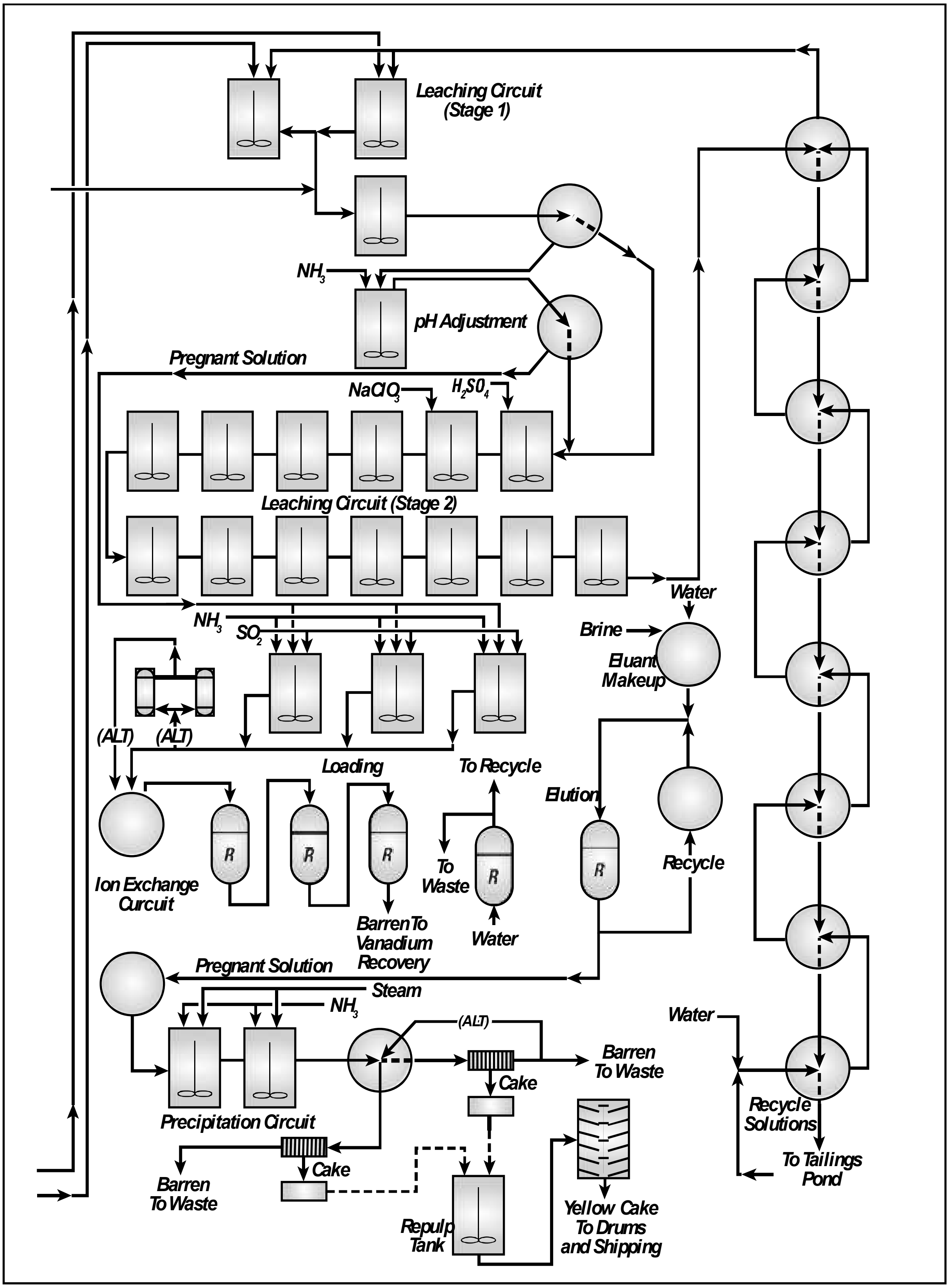

FIG. 48(b). Flowsheet of Union Carbide Corporation's Uravan Mill (adapted from Ref. [87]).

The Uravan mill was closed permanently in 1984 and incurred significant cleanup liabilities, mostly from old tailings. UMETCO (the successor company to Union Carbide) replaced the Uravan mill capacity in 1984 by acquiring a 70% interest in Energy Fuels' White Mesa mill.

Metal recovery

Run-of-mine ore was crushed in a single stage to a minus 50 mm product and stored in six ore bins. Discharge from the bins was done using belt feeders and conveyors to two autogenous grinding circuits. The dry grinding process generatedd a product containing about 3% plus 35-mesh (0.595 mm) and this size fraction was stored in seven fine ore bins.

The two-stage leaching process featured, in the second stage, a hot, strong sulphuric acid leach to improve extraction of vanadium. Strong acid leach liquors from this step were separated from leach residue and returned for the first stage leach at a pH of 0.5–0.8.

The second stage leach was characterized by a residence time of 15–16 h, 50–60% solids, acid addition of about 118 kg/t of ore, sodium chlorate addition of approximately 1 kg/t of ore, a temperature of 79°C and an initial pH of 0.15–0.25. Solid–liquid separation was achieved in eight stages of counter-current decantation. Solids were discharged at 58% to the tailings area. A pregnant solution containing about 1.7 g/L U was transferred to the ion exchange circuit.

The ion exchange circuit consisted of five 2.44 m diameter × 4.7 m high fixed bed columns half filled with anion exchange resin. The breakthrough point was established at 212 ppm U. Resin was stripped in two stages, first with recycled eluant and then with barren eluate containing brine and H_2SO_4. Pregnant eluate containing about 20 g/L was transferred to the precipitation circuit.

Precipitation was carried out in two stages of ammonia addition, first to raise the pH to between 3.0 and 3.3 and then to increase it further, to between 7.3 and 7.4. Incoming pregnant solution was heated to 52°C by direct steam injection. The precipitate was thickened, passed through filter presses and dried in a multiple hearth dryer.

Vanadium in the ion exchange effluent was recovered by SX to produce various vanadium products, including fused vanadium oxide and ferric vanadate. Fused vanadium oxide production at Uravan ceased in 1962, after which all vanadium precipitate was shipped to Union Carbide's Rifle mill for further treatment.

In 1967, recoveries were reported to be 95–96% for uranium and 75% for crude vanadium precipitate.

Production data

Overall mine production tributary to the Uravan mill from 1949 through 1984 is estimated to have been 9.3 million t of ore grading 0.187% U and 0.959% V as weighted averages. From this ore feed, some 17 126 tU and 89 tV are estimated to have been recovered. Historical reserve and production details are summarized in Tables 23 and 24, respectively, and production details are shown in Fig. 49.

TABLE 23. MINEABLE RESERVE SUMMARY: URAVAN
(UNION CARBIDE operations) [88]

Date	Ore (t)	% V	% U	tV	tU	V:U ratio
12/31/1951	832 334	-	-	-	-	-
12/31/1952	901 558	-	-	-	-	-
12/31/1953	986 816	-	-	-	-	-
12/31/1954	1 149 695	-	-	-	-	-
12/31/1955	1 208 913	-	-	-	-	-
12/31/1956	1 618 496	-	-	-	-	-
12/31/1957	2 033 794	-	-	-	-	-
12/31/1958	1 942 367	1.11	0.22	21 585	4 283	5.0
12/31/1959	1 858 106	1.07	0.21	19 800	3 940	5.0
12/31/1960	2 292 760	1.00	0.21	23 036	4 862	4.7
12/31/1961	1 911 428	1.00	0.21	19 204	4 053	4.7
12/31/1962	1 727 095	1.01	0.21	17 484	3 662	4.8
12/31/1963	1 562 286	1.00	0.21	15 578	3 313	4.7
12/31/1964	1 275 939	0.95	0.21	12 140	2 706	4.5
12/31/1965	1 233 275	1.01	0.18	12 485	2 197	5.7
12/31/1966	1 346 949	1.04	0.20	14 046	2 628	5.3
12/31/1967	1 400 126	1.00	0.19	13 961	2 613	5.3
12/31/1968	913 530	1.00	0.19	9 178	1 705	5.4
12/31/1969	780 836	1.00	0.18	7 845	1 391	5.6
12/31/1970	710 090	1.02	0.19	7 242	1 325	5.5
12/31/1971	580 117	1.14	0.18	6 623	1 033	6.4
12/31/1972	667 824	1.09	0.16	7 269	1 076	6.8
12/31/1973	946 259	1.05	0.16	9 939	1 525	6.5
12/31/1974	1 098 261	1.02	0.16	11 202	1 770	6.3
12/31/1975	1 572 557	0.94	0.16	14 842	2 534	5.9
12/31/1976	2 063 108	0.94	0.16	19 472	3 325	5.9
12/31/1977	2 207 230	0.94	0.17	20 832	3 744	5.6
12/31/1978	2 257 636	0.88	0.17	19 762	3 830	5.2
12/31/1979	2 662 544	0.81	0.16	21 482	4 291	5.0
12/31/1980	2 811 337	0.81	0.16	22 896	4 531	5.1
12/31/1981	2 709 753	0.81	0.16	22 069	4 367	5.1
12/31/1982	2 675 197	0.81	0.16	21 788	4 311	5.1
12/31/1983	2 577 966	0.81	0.16	20 996	4 154	5.1
12/31/1984	3 143 662	0.81	0.16	25 603	5 066	5.1
12/31/1985	3 147 970	0.81	0.16	25 638	5 073	5.1
12/31/1986	3 166 836	0.81	0.16	25 792	5 103	5.1
12/31/1987	3 192 300	0.81	0.16	25 763	5 132	5.0
12/31/1988	4,036,399	0.84	0.17	34 150	6 814	5.0
12/31/1989	3 563 603	0.90	0.19	32 061	6 542	4.9
12/31/1990	3 444 967	0.88	0.19	30 472	6 320	4.8
12/31/1991	3 622 286	0.89	0.19	32 238	6 611	4.9
12/31/1992	3 433 539	0.88	0.19	30 392	6 306	4.8
12/31/1993	3 243 251	0.88	0.19	28 630	5 977	4.8
08/23/1994	2 483 094	0.91	0.19	22 663	4 569	5.0

TABLE 24. HISTORICAL MINE PRODUCTION: URAVAN
(UNION CARBIDE operations) [88]

Year	Dry ore (t)	Uranium		Vanadium	
		(tU)	(%U)	(tV)	(% V)
1949	48 932	149	0.30%	669	1.37%
1950	67 498	217	0.32%	930	1.38%
1951	57 930	197	0.34%	882	1.52%
1952	82 581	266	0.32%	1 194	1.45%
1953	92 973	276	0.30%	1 238	1.33%
1954	78 867	241	0.31%	1 081	1.37%
1955	79 290	222	0.28%	996	1.26%
1956	146 651	398	0.27%	1 786	1.22%
1957	298 561	633	0.21%	2 909	0.97%
1958	336 660	771	0.23%	3 511	1.04%
1959	411 107	872	0.21%	4 318	1.05%
1960	491 277	1 000	0.20%	4 749	0.97%
1961	441 627	936	0.21%	4 471	1.01%
1962	433 292	882	0.20%	4 090	0.94%
1963	325 767	636	0.20%	3 273	1.00%
1964	245 253	478	0.20%	2 669	1.09%
1965	236 047	481	0.20%	2 785	1.18%
1966	322 030	601	0.19%	3 309	1.03%
1967	259 611	440	0.17%	2 411	0.93%
1968	347 145	618	0.18%	2 986	0.86%
1969	321 532	545	0.17%	2 888	0.90%
1970	355 220	603	0.17%	3 407	0.96%
1971	312 052	529	0.17%	2 731	0.88%
1972	261 917	400	0.15%	2 592	0.99%
1973	128 302	207	0.16%	1 289	1.00%
1974	248 641	316	0.13%	2 139	0.86%
1975	269 840	389	0.14%	2 506	0.93%
1976	315 432	482	0.15%	2 833	0.90%
1977	375 486	541	0.14%	3 115	0.83%
1978	369 466	533	0.14%	3 065	0.83%
1979	399 294	542	0.14%	3 039	0.76%
1980	406 400	586	0.14%	3 186	0.78%
1981	295 016	450	0.15%	2 425	0.82%
1982	271 772	415	0.15%	2 110	0.78%
1983	151 705	232	0.15%	1 178	0.78%
1984	29 024	42	0.14%	247	0.85%
Total/average	9 314 198	17 126	0.20 %	89 007	1.02 %

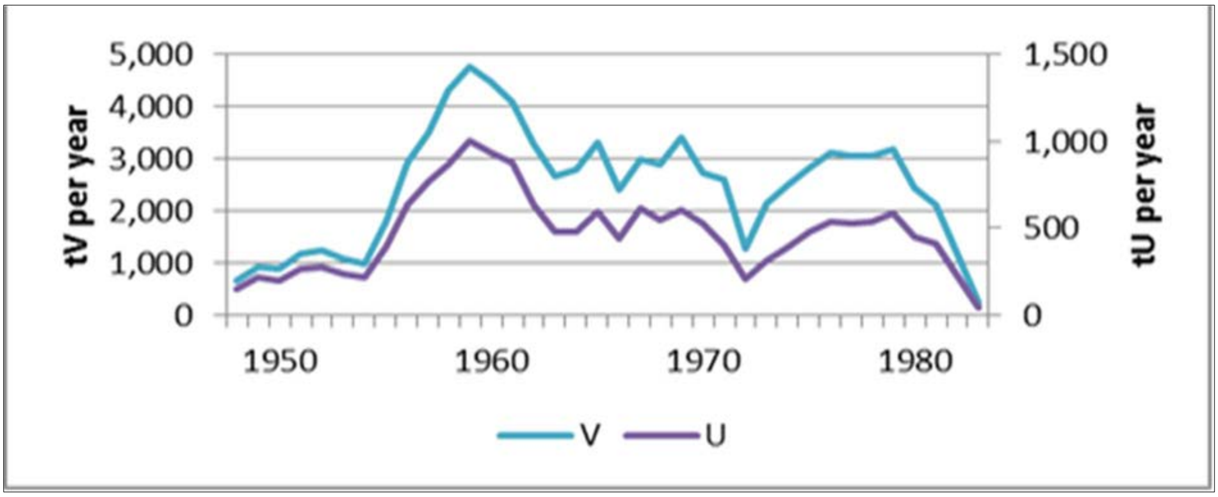

FIG. 49. Uravan mine production (courtesy T. Pool).

More than 200 individual mines were tributary to the Uravan mill during its operating life. Some of the more important ones were: La Sal, Deremo, Sunday, King Solomon, Wilson–Silverbell and Snowball.

Economic analysis

Using nominal annual prices for uranium and vanadium, Fig. 50 depicts the changing nominal annual value of uranium and vanadium production and indicates that uranium and vanadium could realistically be viewed as co-products.

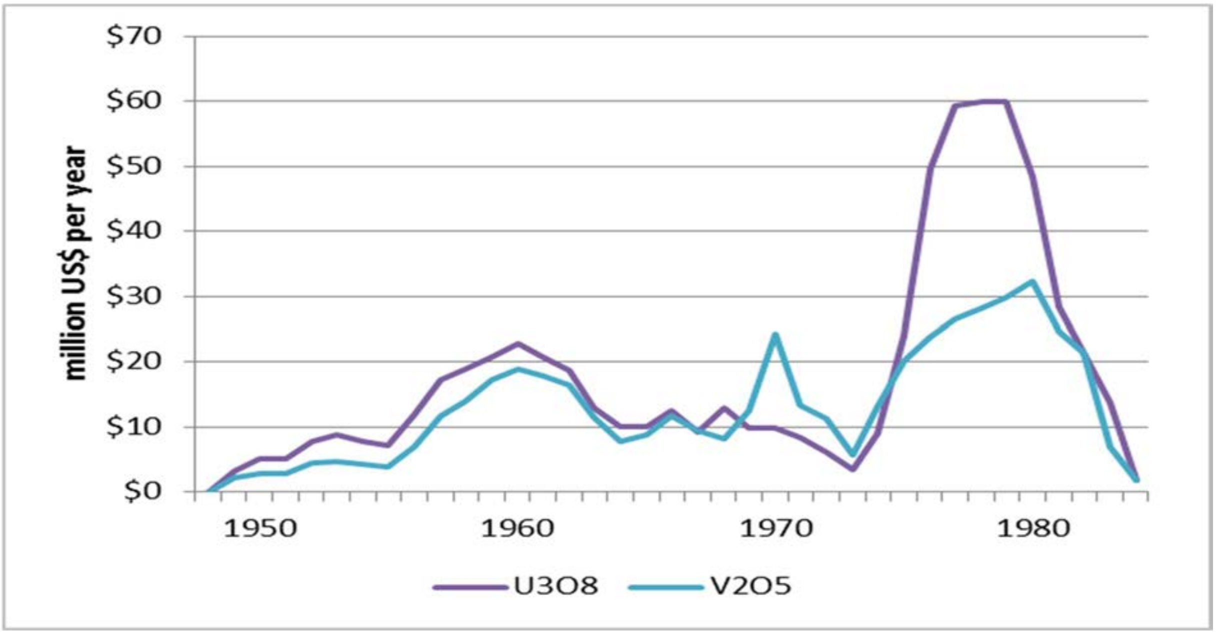

FIG. 50. Uravan nominal annual revenue (courtesy T. Pool).

On an overall basis, at nominal prices, distribution of revenue is estimated to have been 58% attributable to uranium and 42% attributable to vanadium.

Certain economic data pertaining to Uravan operations in 1982 are shown in Table 25 [89]. By updating these costs to 2014, through relative producer price indices, a current perspective on production costs can be gained.

Productivity at the Uravan operations, shown in Table 26, is derived by assessing production and Mine Safety and Health Administration records.

TABLE 25. URAVAN OPERATING COSTS [89]

Item	$/t ore (1982)	$/kgU (1982)	$/t ore (2014)	$/kgU (2014)
Mine development	11.27	7.12	23.01	14.56
Mining	42.07	26.65	85.95	54.42
Haulage	12.06	7.64	24.64	15.60
Milling	42.72	27.07	87.30	55.28
Other	18.96	12.01	38.75	19.27
Total	**127.09**	**80.47**	**259.66**	**159.12**
Vanadium credit	(70.92)	(44.90)	(144.88)	(44.90)
Net cost	56.17	35.57	114.77	114.22

TABLE 26. PRODUCTIVITY: URAVAN OPERATIONS

Item	1978	1979	1980
Ore mined and processed (t)	369 466	399 294	406 400
Person-hours expended at:			
Mine	612 782	774 655	787 347
Mill	376 434	404 884	435 911
Productivity at:			
Mine (person-hour/t)	1.66	1.94	1.94
Mill (person-hour/t)	1.02	1.01	1.07

4.7.1.6. White Mesa

The White Mesa uranium–vanadium mill is located 10 km south of Blanding in south-eastern Utah. It was constructed by Energy Fuels Nuclear Inc. and was based on the anticipated reopening of many of the small low-grade mines on the Plateau. It was originally designed to process 1800 t/d of ore. Construction began in 1979 and the mill started up in 1980. Capital costs totalled approximately US $40 million. Owing to changing economics, design capacity was not reached and the mill was subsequently modified to process higher grade ores from the company's Arizona Strip mines.

In 1983, UMETCO purchased a 70% operating interest in the White Mesa mill to replace its ageing Uravan facility. The capacity of the White Mesa mill exceeded the mine production capabilities of both partners and, consequently, mine production was stockpiled and the ore processed at intermittent intervals, the mill being closed at other times. UMETCO's interest was relinquished in 1990. Following Energy Fuels Nuclear's bankruptcy in 1995, International Uranium Corporation (IUC) acquired the assets, including the White Mesa mill, in 1996. In 2006, IUC and Denison Mines (Canada) completed a merger and began operating under the Denison name. Denison's US assets were acquired in 2012 by Energy Fuels Inc. (a new company not associated with the original Energy Fuels Nuclear).

In the early 1980s, most of Energy Fuels Nuclear's Plateau ore was derived from the following mines: Betty (Utah), C-SR-10 (Colorado), Hillside (Utah), Repeat (Utah), Sahara (Utah) and Sugar Daddy (Utah). In 2006, Denison began reopening several mines on the Plateau, including Rim, Beaver and Pandora. The Daneros mine was acquired in 2011 from Utah Energy Corp. All Plateau mines were on standby as of December 2012 and remain so as of May 2015.

Metal recovery

The White Mesa mill incorporated a sulphuric acid leach combined with an SX process containing separate recovery circuits for uranium and vanadium extraction. Grinding of the mill feed to minus-28 mesh (0.65 mm) was undertaken in a 1.8 m × 5.5 m diameter SAG mill. Leaching of the ore was done in two stages.

The first or pre-leach circuit, consisting of two mechanically agitated tanks, utilizes pregnant (high grade) strong acid solution from the counter-current decantation circuit, thus initiating leaching as well as neutralizing excess acid. The pre-leach circuit discharges to a 38-m thickener; underflow solids are pumped to the second stage leach and thickener overflow is pumped to clarification, filtration and the SX circuits.

A hot strong acid leach is used in the second stage leach unit consisting of seven mechanically agitated tanks and employing a retention time of 24 h. Free acid is controlled at 70 g/L and the temperature is maintained at 75°C.

Leached pulp is washed and thickened in the counter-current decantation circuit consisting of seven high capacity thickeners. The final thickener underflow at 50% solids is discharged to the tailings area (a 26-ha synthetic lined pond); overflow from the first thickener (pregnant solution) is returned to the pre-leach tanks.

The SX circuit consists of four extraction stages in which uranium in the pregnant solution is transferred to the organic phase, which is a mixture consisting of 2.5% amine, 2.5% isodecanol and 95% kerosene. Loaded organic is then pumped to six stages of stripping by a 1.5M sodium chloride solution and subsequently to a continuous anhydrous ammonia precipitation circuit. Precipitated uranium is settled, thickened, centrifuged and dried at 593°C. The 80% U_3O_8 product is packed into 200 L drums for shipment (Fig. 51).

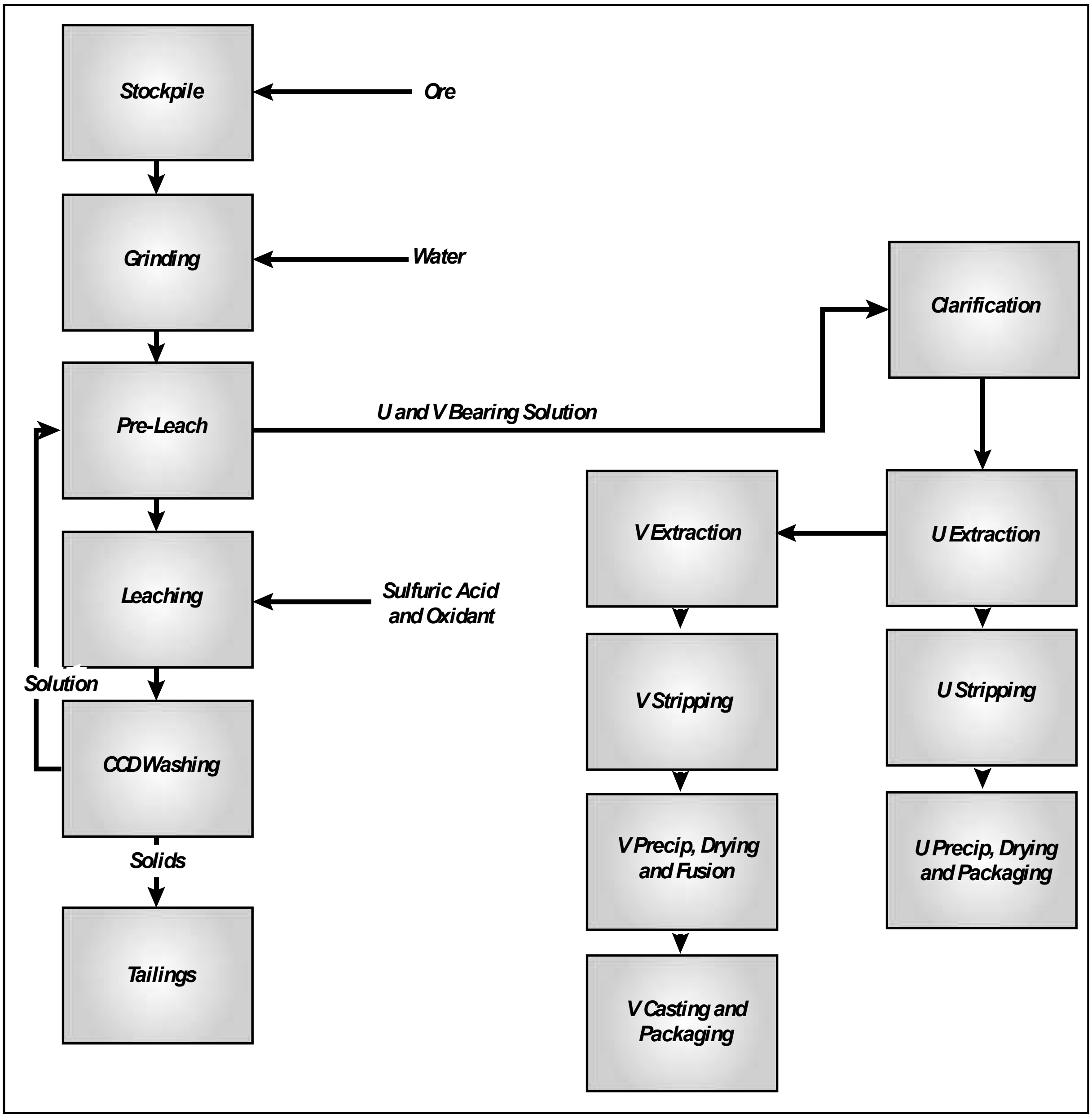

FIG. 51. Process flowsheet for White Mesa mill (adapted from Ref. [90]).

From 1980 through 1995, the White Mesa mill processed 3.75 million t of ore at an average grade of 0.31% U, producing 10.68 million kgU (10 680 tU). In 1996, mill operating costs amounted to approximately US $50/t of ore. When vanadium-bearing ores were being processed, the uranium tailings solution (raffinates) containing soluble vanadium was passed through a series of pH and oxidation adjustment tanks and fed to an SX circuit like the uranium SX circuit. Following SX, loaded vanadium strip solution is then batch precipitated, dried, fused to V_2O_5 and packaged in the same manner as yellowcake (U_3O_8).

Production data

From 1980 through 1990, the White Mesa mill processed 1.6 million t of vanadium-bearing ore at an average grade of 0.92% V, producing 11 512 tV. Overall recovery was 75.7% and mill operating costs amounted to US $3.13/kg V.
Table 27 provides a history of uranium and vanadium production at White Mesa for the account of Energy Fuels Nuclear/International Uranium/Denison mines. Production for the Union Carbide (UMETCO) account is not included.

TABLE 27. PRODUCTION AT WHITE MESA MILL [91]

Year	Alternative feed (tU)	Arizona Strip (tU)	Colorado Plateau	
			Uranium (tU)	Vanadium (tV)
1980	0	243	104	404
1981	0	477	1 044	757
1982	0	603	1 680	487
1983	0	142	160	231
1984	0	0	0	146
1985	0	324	0	0
1986	0	1 698	610	449
1987	0	1 641	147	263
1988	0	791	177	306
1989	0	967	914	1 180
1990	0	608	502	850
1991	0	0	0	0
1992	0	0	0	0
1993	0	0	0	0
1994	0	0	0	0
1995	0	567	0	0
1996	58	202	0	0
1997	231	0	0	0
1998	31	0	0	0
1999	64	0	187	413
2000	0	0	0	0
2001	0	0	0	0
2002	0	0	0	0
2003	0	0	0	0
2004	0	0	0	0
2005	0	0	0	0
2006	93	0	0	0
2007	98	0	0	0
2008	36	0	304	191
2009	73	0	236	78
2010	115	0	405	367
2011	77	0	389	202
2012	167	0	488	37
2013	135	387	0	204
2014	150	212	0	0
Total	1 328	8 862	7 347	6 565

Production cost

During the period 2008–2010, Denison Mines published production cost information in company reports filed quarterly and annually with the US Securities and Exchange Commission. These data are given in Table 28.

TABLE 28. WHITE MESA PRODUCTION AND COSTS DURING OPERATING TENURE OF DENISON MINES [92]

Period	Conventional (tU)	Alternative feed (tU)	Production (tV)	Production cost ($/kgU)	DDA[a] ($/kgU)	Total cost ($/kgU)
YE 31/12/2008	304	36	422	$99.16	$72.07	$171.24
YE 31/12/2009	236	73	173	$97.81	$84.86	$182.68
YE 31/12/2010	405	115	810	$100.00	n.a.[b]	n.a.
YE 31/12/2011	389	77	445	$123.76	n.a.	n.a.

[a] DDA: Depreciation, depletion and amortization.

[b] n.a.: not available.

4.7.2. Uranium–copper: Mexican Hat and Moab (USA)

Uranium–copper ores occur in the White Canyon district of south-eastern Utah, hosted within tabular sandstones of the Triassic Shinarump Member of the Chinle Formation. These sandstone ores are located mainly in palaeochannels and exhibit grades of 0.16–0.25% U and, in some deposits, copper grades of up to 2%. The Happy Jack mine was one of the more prolific producers of uranium–copper ore [93].

From 1957 through 1965, uranium–copper ores were milled at the Mexican Hat mill, which was owned and operated by Texas Zinc Minerals Corporation. The mill had an initial capacity of 638 t/d , which was raised to 823 t/d in 1958. The mill process consisted of crushing, grinding, sampling, flotation of a copper sulphide concentrate, leaching with sulphuric acid and manganese dioxide in separate copper and uranium circuits, filtration of leached copper concentrates, five-stage counter-current decantation of the main leach pulp which contained the bulk of uranium in solution, clarification of the pregnant liquor in pressure type filters, uranium recovery by SX, precipitation of uranium oxide by neutralizing the loaded acidified nitrate with magnesium hydroxide, thickening, washing, filtering, drying and packaging the uranium concentrate. Copper concentrates were shipped to a smelter [93].

Atlas Minerals purchased the Mexican Hat mill in 1963 and then closed it in 1965 and relocated the copper circuit to its Moab mill where it processed ore containing sufficient copper to justify recovery intermittently in its alkaline circuit. Approximately 10% of the alkaline plant's total operating time (at a feed rate of 25 t/h) was required to process available copper-bearing ores. Assuming 90% availability, annual copper ore production was in the region of 20 000 t/year.

The copper recovery process at Moab was in keeping with that at Mexican Hat. Flotation concentrates containing about 20% Cu and up to 0.85% U were leached with sulphuric acid. Residence time was 12 h. After leaching, the residue was separated from the solution by filtration and water washing of the cake. The residue was stored as final copper concentrate. Dissolved uranium in the pregnant leach solution was recovered in a special solvent extraction unit [94].

4.7.3. Uranium–molybdenum: Akouta, Niger

4.7.3.1. History

The Akouta deposit was discovered by the France's Commissariat à l'Energie Atomique during a surface drilling programme carried out in 1971–1972 to investigate uranium showings detected in exploration programmes conducted during the 1960s. Follow-up work in 1974–1975 resulted in the delineation of a uranium deposit containing 40 000 tU [95].

COMINAK, La Compagnie Minière d'Akouta, was established on 12 June 1974 to develop and mine the deposit. Mining began in April 1978 and the first uranium concentrates were produced on 31 August 1978. On 24 December 1979, the first molybdenum concentrates were produced.

The project has operated continuously since that time and was operating at capacity as of 2014. Total uranium production since inception is 67 631 tU. Molybdenum production reached 150 t in 1983, but such production data have been unavailable since 1985 [96].

4.7.3.2. Geology

Akouta is a tabular sandstone type deposit. Mineralization occurs in basal channel units of the Guezouman Formation, which is Carboniferous in age and has a thickness of up to 45 m. The original ore reserves were estimated at 40 000 tU and the average grade was reported to be 0.29–0.39% U. Coffinite and pitchblende are the predominant uranium minerals. An amorphous molybdenum sulphide (jordisite) is also present and occurs in association with very high uranium concentrations in highly reduced zones. Typical molybdenum: uranium grade ratios vary in the range 0.2–0.3.

4.7.3.3. Mining

Underground access to the mining levels at 250 m is provided by two declines, each 1300 m in length and having a 20% slope. One decline is used for personnel, materials and equipment access and is paved. The other decline houses the main conveyor belt that brings ore and waste to the surface [96].

The mining method employed is room and pillar and the sequence of extraction is similar to that used in coal mines. Access to the pillars is developed in ore; the pillars are then removed on retreat. Pillars are replaced with a cementitous fill that allows for 100% extraction of the Akouta orebody. In the new Ebba orebodies, pillar extraction will only be 75% because of the thinner ore horizon, lower ore grade and weaker rock [97].

Mining is accomplished with rubber-tired equipment for drilling, bolting, cleaning (scaling), loading and transport. Total annual mine production amounts to approximately 500 000 t of material, including low grade ore and waste [98].

4.7.3.4. Milling

Run-of-mine ore is crushed underground and transported to the surface by conveyor where it undergoes semi-autogenous grinding, pugging with a mix of sulphuric acid/sodium nitrate/nitric acid and water, solid–liquid separation effected with belt filters, concentration by solvent extraction and precipitation with milk of magnesia to produce a magnesium diuranate product. Following uranium SX, molybdenum was removed by SX to provide a molybdenum-bearing pregnant solution (containing 12.5 g/L Mo). This solution is initially heated to 70°C and its pH lowered to 2 by the addition of sulphuric acid. The purpose of this treatment is to break down any carbonates present and ensure that they are not precipitated at the same time as the molybdate. This operation is carried out in a 10 m^3 tank equipped with a degassing cover. Retention time is one hour. Figure 52 shows a schematic of the flowsheet [13].

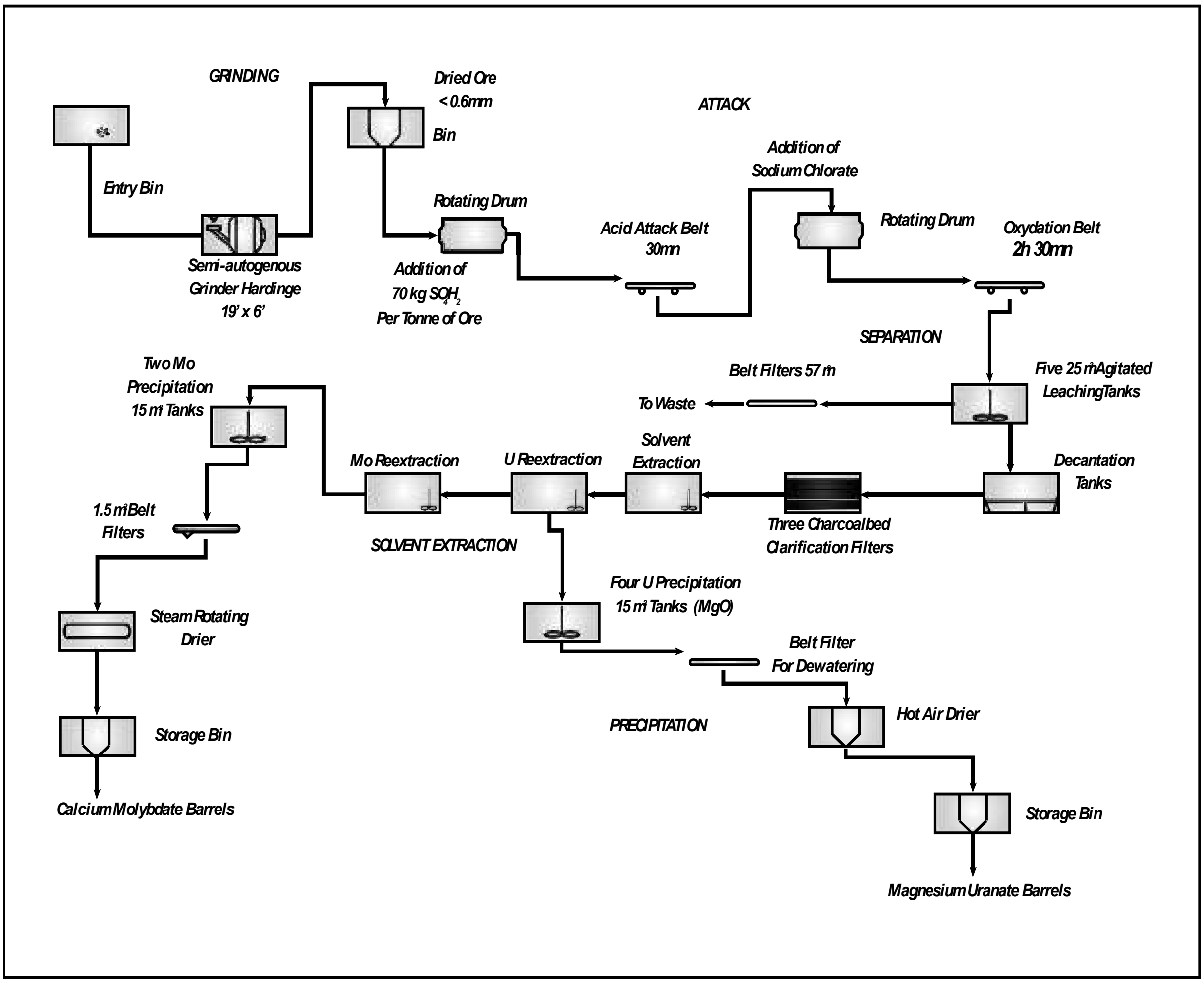

FIG. 52. Ore processing schematic flowsheet at Akouta (adapted from Ref. [13]).

Molybdenum in the pregnant solution is precipitated in a 15 m^3 tank as calcium molybdate by the addition of milk of lime (100 g/L) at a pH of 9–9.7 and a temperature of 50–55°C [13].

The pulp obtained is then thickened in a 3.6 m^3 settling tank with the addition of a flocculant. Underflow from the settling tank is filtered and then washed in water. The cake is then dried to a residual moisture content of less than 5%. After drying, the calcium molybdate contains around 38% molybdenum [13].

A major reason for undertaking molybdenum recovery is due to concerns over high levels of molybdenum remaining in the uranium concentrates. Generally, high levels of molybdenum (>0.3% Mo) incur penalties at uranium conversion facilities. It is not clear whether Akouta’s molybdenum concentrates are of commercial quality, nor is it clear when molybdenum production might have ceased. Current operations do not include molybdenum recovery.

4.7.4. Uranium in siltstones: Egypt

Low grade uranium–copper mineralization has been recorded in Palaeozoic rocks of west central Sinai in Egypt. Mineralogical data have indicated that the lithologies of west central Sinai that host uraniferous mineralization are essentially composed of quartz, clay, calcite and/or dolomite, as well as iron oxide and gypsum. Several secondary uranium and copper minerals have also been identified, including, for uranium: uranophane, beta-uranophane, clarkeite, meta-torbernite, kasolite and carnotite, and for copper: atacamite, paratacamite, langite, spangolite, chalcanthite, turquoise and chalconatronite.

Leaching studies performed on a sample from the Abu Thor area assayed at 0.31% U and 0.22% Cu. A sulphate leach liquor was prepared by using the optimum conditions determined for the pug leaching method (200 kg acid/t ore, curing time of 1.5 h at 110°C followed by water leach for 6 h at a solid/liquid ratio of 1:3 at room temperature). The pregnant solution had a pH of 1.1 and assayed 1.125 g/L U and 0.45 g/L Cu. Both copper and uranium were extracted from this solution by SX. Metal recovery from solution exceeded 95% for both elements [99].

There is no recent information on the status of this project.

4.8. PALAEO QUARTZ-PEBBLE CONGLOMERATE DEPOSITS

4.8.1. Gold–uranium (South Africa)

The occurrence of uranium in association with gold ores in South African quartz pebble conglomerates was first determined through evidence of radioactivity in the late 1880s. However, it was not until 1915 that the extent of the association with reef systems was established, and in 1923 the radioactive source was identified as uraninite, one of the most important uranium ore minerals. The Government Metallurgical Laboratory initiated studies focusing on the recovery of uranium from gold ores in the early 1940s. The first pilot recovery plant was successfully producing uranium concentrate in 1945. A second and larger pilot plant installed in the milling circuit of the Blyvooruitzicht gold mine in 1949 proved that commercial extraction was practical. A full-scale recovery plant began operating in late 1952. Table 29 provides a summary of historical uranium production in South Africa as a by- or co-product of gold mining. Total historical production through 2014 is estimated to be 158 934 tU (Figs 53 and 54).

TABLE 29. SUMMARY OF SOUTH AFRICAN URANIUM PRODUCTION AS A BY-PRODUCT OF GOLD MINING (tU)

Production facility	Output (tU)	First year of operation	Last year of operation
Beisa	901	1982	1984
Blyvooruitzicht	5 555	1953	1984
Buffelsfontein	15 465	1957	1991
Chemwes	3 998	1979	1988
Daggafontein	1833	1953	1962
Dominion (Riet Kuil)	293	2007	2008
Dominion Reefs	1 546	1955	1963
Doornfontein	226	1957	1962
East Cham d'Or	469	1953	1964
Ellaton	280	1953	1961
ERGO	2 150	1978	1991
Ezulwini/Cooke No. 4	141	2009	Present
Freddies	493	1955	1960
Free State Saaiplaas	39	1953	1955
Harmony	7 992	1955	1988
Hartebeestfontein	13 959	1957	2000
Lorraine	281	1956	1960
Luipaards Vlei	2 983	1955	1964
Merriespruit	436	1980	1984
President Brand	481	1955	1960
President Brand (JMS)	7 316	1977	1990
President Steyn	624	1955	1960
Randfontein Cooke	3 835	1978	1988
Randfontein mill site	6 138	1954	1981
Stilfontein	838	1953	1961
Vaal Reefs East	10 993	1956	1990
Vaal Reefs South	21 995	1979	Present
Vaal Reefs West	17 906	1953	1997
Virginia	6 648	1955	1986
Vogelstruisbult	717	1955	1964
Welkom	416	1957	1961
West Driefontein	3 321	1956	1988
West Rand Consolidated	11 054	1952	1981
Western Areas	3 430	1982	1997
Western Deep Levels	2 164	1970	1985
Zandpan	1 203	1966	1972
Others	782	1953	1961

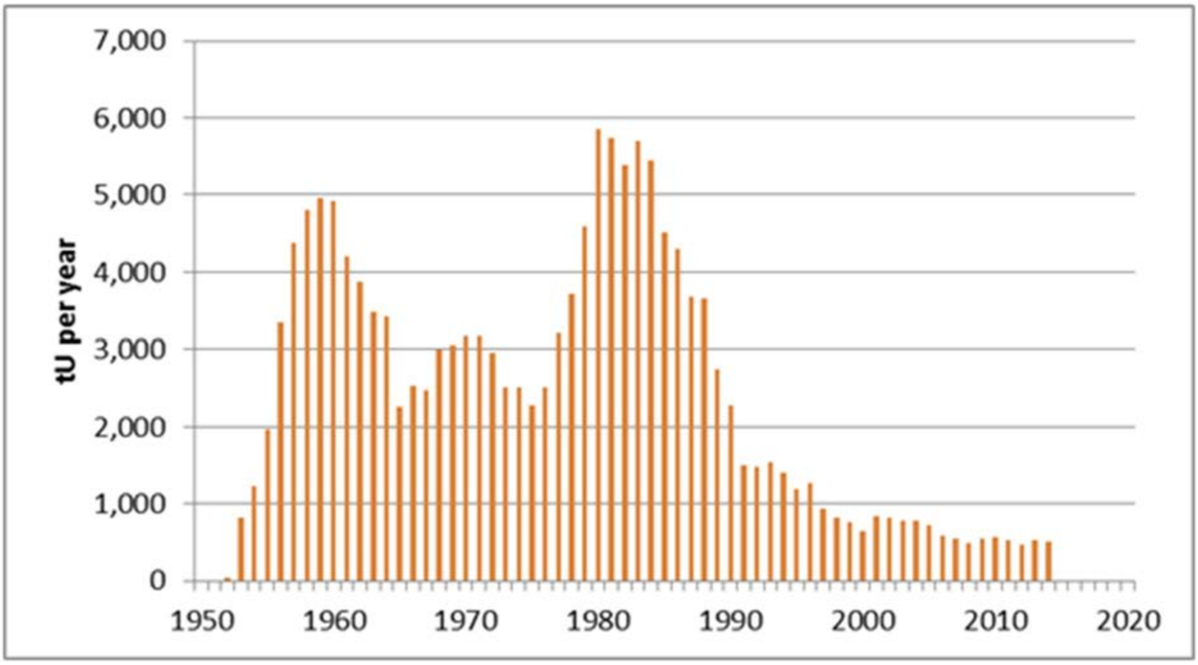

FIG. 53. South African uranium production as a by- or co-product of gold mining (courtesy T. Pool).

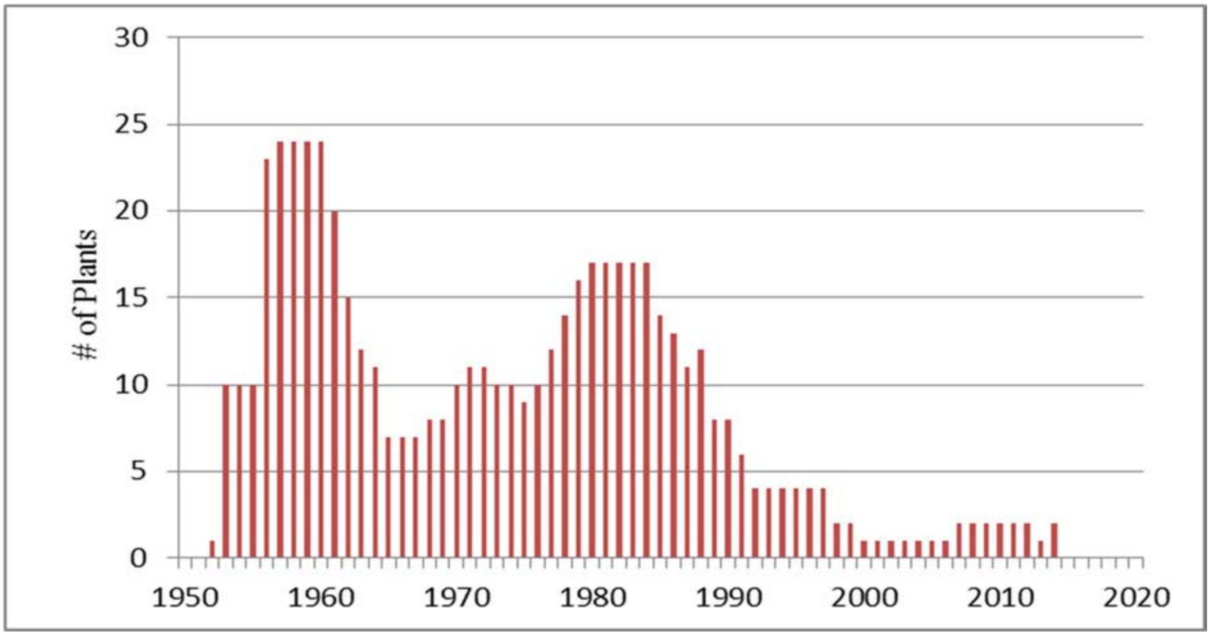

FIG. 54. Historical gold/uranium operations in South Africa (courtesy T. Pool).

4.8.1.1. Geology

The Witwatersrand sedimentary system of Meso-Archaean age is one of the world's major sources of gold and uranium. Sediments of the Witwatersrand system were deposited in a structural basin bounded on the north-west and south-west by faulting. Downwarping towards the basin centre and to the south-east created the deepest section of the basin where sediments are approximately 7600 m thick. The important mines are located along the shallower north-western and southern tip of the basin. Current mining depths vary from 600 m to over 3700 m.

The Dominion Group strata, followed by the Witwatersrand Supergroup, were deposited unconformably on Archaean basement rocks as a series of coarse sandstone units interbedded with siltstones and conglomerates. The irregular depositional sequence formed as fluvial fans, deltas and channel fill. Uranium mineralization is hosted in conglomerate beds of the Dominion Group, whereas gold and uranium mineralization are associated with the conglomerate beds (reefs) of the 4000-m thick section of the middle portion of Witwatersrand Supergroup.

Important mineralization is found in the matrix of conglomerates, in sand stringers containing moderate to high concentrations of pyrite, along the contact zones of shale and quartzitic beds and in carbon/shale seams related to planes of unconformity. The minerals were transported as detrital particles and deposited during the sedimentary sequence and are therefore syngenetic. However, there is evidence that certain occurrences, primarily gold in the shale and carbon environment, formed later because of chloride solutions and biochemically through interaction between the gold (and uranium) and the algal or lichen colonies that developed along the edges of channels and in deltaic deposits. Reef systems containing gold and uranium mineralization in sufficient quantities and of high enough grade to be economically mined are confined to only 10–12 conglomeratic sequences. The lowest reefs include the Ada May (Beisa Reef) in the Orange Free State; the Carbon Leader of the Main Conglomerate Formation in the Central Rand; the White, Monarch and Bird Reefs on the West Rand; the Vaal Reef in the Klerksdorp Goldfield; the Basal and Leader Reefs in the Orange Free State Goldfield; the Elsburg Reef in the Central Rand; and their equivalents, the A and B Reefs, in the Orange Free State.

Uraninite generally occurs as minute crystals, usually less than 0.08 mm in diameter, frequently showing signs of rounding. Secondary uraninite sometimes encloses and even replaces primary uraninite grains. In the case of the Bird Reefs in the West Rand and Klerksdorp areas, secondary uraninite is a dominant constituent.

A variety of quartz pebbles are found in the conglomerate reefs, with those of vein quartz predominating. Pebbles comprising tourmalinized rock or argillaceous or schistose types are rare. The pebble matrix material consists mainly of quartz of sand grain size, accompanied by varying amounts of pyrite and other sulphides. Heavy minerals are represented by chromite, zircon and leucoxene. Apart from gold and uranium minerals, other valuable constituents occur in minute traces including iridosmine and osmiridium.

Phyllosilicates are present in the form of muscovite, pyrophyllite and chlorite. Carbon (thucholite) occurs as granules, about 2 mm in diameter, as well as forming seams. The carbon seams can be up to 15 mm thick and often mark the lower bedding planes of the so-called ‘carbon seam reefs’ in the West Rand, Klerksdorp and Odendaalsrus areas.

Thucholite is of economic importance owing to abundant inclusions of uraninite, as well as pyrite and gold. Such uranium may not respond to conventional ore processing techniques, however, and significant losses can result from the presence of such material in a process feed stream.

Economic conglomerate beds are quite thin, generally less than 1.0 m, which is the minimum thickness for economic exploitation. Thus, in many circumstances, mining dilution is a major economic factor.

Structure is another important variable. Most conglomerate beds are faulted and continuity is sometimes difficult to establish.

4.8.1.2. Resources

Kenan [100] reports current uranium resources within the Witwatersrand Basin as follows:

— Measured: 47 523 tU at an average grade of 0.097 kgU/t ore;
— Indicated: 219 607 tU at an average grade of 0.094 kgU/t ore;
— Inferred: 300 855 tU at an average grade of 0.302 kgU/t ore.

4.8.1.3. Mining

Mine access

Some of the world’s deepest shafts are situated in mines located in the Witwatersrand Basin. Shaft No. 9 at Vaal River provides an example. The sinking of the No. 9 shaft system began in 1978, the initial hoisting of ore began in 1984 and full capacity (280 000 t/month) was reached in 1986. The shaft system consists of two shafts: the main hoisting shaft, 10.6 m diameter and a ventilation shaft, 7.5 m diameter. The hoisting shaft reaches a depth of 2342 m, the deepest in existence for a single hoist. Two Blair multi-rope hoists are used; one hoisting two 20 t skips in balance, and the other hoisting 15 t skips. In addition, the shaft is equipped with two double-drum men and material hoists capable of carrying 50 workers per deck or 10.5 t of material per conveyance. The rock hoists use two rope (cable) skips. Each rope has a diameter of 48 mm. The man and material conveyances use a single rope 54 mm in diameter.

The No. 9 system is connected with the No. 8 and No. 3 shafts each over a distance of 4 km. The shaft system is intended, eventually, to provide access to 16 levels. Ventilation is provided by three fans generating a total upcast exhaust capacity of 1005 m^3/s of air. The refrigeration system installed by 1991 includes four 12 000 kW units providing chilled water and/or air. It is planned that 84% of the downcast air volume will be cooled to 7°C. The combined planned refrigeration system is believed to be the world’s largest.

Mine development and stoping

All major development occurs in the footwall in order to avoid the problems of subsidence. Mine development consists of:

— Establishing a series of shaft stations during the shaft sinking process;
— Driving cross-cuts to access the orebodies;
— Installing services.

Shaft stations are situated at vertical intervals of 50–100 m within the ore profile. Cross-cuts from the shaft stations to the ore horizon are driven using drill jumbos and mechanical loaders. Progress is typically 3–4 m per 8-hour shift. Diamond drilling is used to probe ahead of development to detect water-bearing fissures. If detected, such fissures would be sealed off by cement grouting. Mining services, including track, water, compressed air, ventilation and communications, are installed within the cross-cuts.

Stope development is by means of footwall drives along the strike of the orebody and cross-cuts into the orebody. Within the orebody, low to medium angle (20–45°) raises are driven to block out ore and define the stoping blocks. Stopes might be of the order of 200 m in length and are divided into a series of panels, perhaps 40 m in width. Stoping heights are very low, usually about 1 m, but can range up 3 m in some instances. Ore is drilled to 1 m depths with air-leg or hand-held drills and blasted with conventional explosives. Mucking is generally by slusher, perhaps double or triple, from the face to boxhole raises connected to the haulage level. Mine roof support was traditionally affected by a series rock filled timber cribs. Newer methods include hydraulic props and backfill with tailings.

Mine characterization

The South African Chamber of Mines developed the following profile of a ‘typical’ mine [101]:

— Depth: 1600 m;
— Output: 250 000 t/month;
— Stope height: 1.3 m;
— Underground workers: 8800;
— Surface workers: 2600;

- Area mined: 50 000 m^2/month;
- Face available for production: 10 000 linear m;
- Face advance: 5 m/month;
- Development required: 30 km/year;
- Virgin rock temperature: 38°C;
- Stope wet bulb temperature: 29°C.

4.8.1.4. Vaal River

The Vaal River (formerly Vaal Reefs) division of AngloGold Ashanti consists of three underground mines, one surface operation which reclaims tailings (slimes) and a combined processing plant with a total ore treatment capacity of over 800 000 t/month. Gold is the principal product. Uranium and sulphuric acid are minor by-products.

History

Vaal River has operated three uranium recovery plants: East, South and West. The East plant was commissioned in May 1956 and operated periodically through 1990. The West plant, which was originally built and operated by Western Reefs, was in continuous service from October 1953 through 1997. This plant was acquired by Vaal Reefs in September 1971. The South plant entered service in 1979 and has operated continuously ever since.

Included is an adjacent property, the Afrikander lease, which was leased by Vaal Reefs and brought into production at the rate of 50 000 t/month in 1981. Vaal River acquired the Mine Waste Solutions project from First Uranium in 2012 and commissioned the uranium recovery plant in late 2014. In total, the Vaal River plants produced 50 894 tU through 2014.

Resources

As of 31 December 2014, Vaal River reported a mineral resource of 1051.27 million t of potentially economic material grading 0.1 kgU/t and containing 108 612 tU. Included within this resource is an ore reserve of 506.88 million t of economic material grading 0.09 kgU/t and containing 47 146 tU. Both the resource and the reserve reflect the recent addition of material at the Mine Waste Solutions project.

Milling

Vaal Reefs began uranium extraction in 1956 using standard acid leach in Pachuca tanks with solid–liquid separation effected by drum filter. IX was used for uranium recovery. SX was introduced in 1970. As the operation of all plants is similar, only the South Lease Area Plant is described in detail in the following paragraphs.

Run-of-mine ore is milled in two parallel units, 4.88 m diameter × 9.14 m long, followed by single stage classification in a pump-fed 1050 mm cyclone. The grinding mills operate autogenously but may use steel balls as a grinding medium if necessary.

Mill product is screened to remove wood chips prior to thickening in two 60 m diameter swing-lift thickeners. To prevent contamination in the gold recovery circuit, the thickened pulp is again screened prior to leaching in the gold plant.

Gold recovery

The gold leach section consists of ten 2000 m^3 mechanically agitated leach tanks having a retention time of 48 h. Sodium cyanide is added at the head of the circuit. The leached slurry flows under gravity to the carbon-in-pulp adsorption circuit through eight mechanically agitated tanks (each of 350 m^3 capacity) arranged in series. Activated carbon is transferred from tank to tank by screens situated counter-current to the pulp flow. The loaded carbon from No. 1 tank is deslimed and washed and sent to a 15 m^3 elution column. The eluant is a 1.0% caustic soda/0.2% sodium cyanide solution. This is fed into the elution column at a temperature of 120°C and dissolves the adsorbed gold. The gold-bearing solution flows through heat exchangers to six electrowinning cells where gold is recovered on steel wool cathodes. Periodically, the gold loaded cathodes are removed for calcining and smelting. The carbon is reactivated in a rotary kiln at 700°C and recycled.

Uranium recovery

After removal of the charcoal from the pulp in the carbon-in-pulp circuit, the slurry is filtered in three-disc filters. The filtered pulp, at a density of 1.61g/cm^3 is fed to fourteen sulphuric acid leach Pachuca tanks, using manganese dioxide as an oxidant. Retention time at 60°C is 16 h.

Leached pulp is washed and thickened in six counter-current decantation thickeners. Washed pulp from the No. 6 thickener is returned to the gold recovery plant to scavenge any remaining gold. Pregnant uranium solution (overflow No. 1 thickener) is treated in the counter-current IX section. The IX section consists of five units, each having one resin adsorption column, one measuring chamber, one rinse chamber and one elution chamber. Loaded resin is eluted to produce a concentrated uranium solution enhanced by a factor of 50 compared with the pregnant solution fed to the IX circuit.

Concentrated eluate is pumped to the SX plant, which comprises three each of solvent extractors, scrubbers, strippers and precipitation units. The uranium content is first transferred to the organic phase, scrubbed and stripped by ammonium sulphate back to the aqueous phase, with the now barren organic phase cycled to the SX units. Aqueous strip solution is treated with liquid ammonia to produce an ammonium diuranate precipitate, which is thickened, centrifuged and shipped by tank truck to Nuclear Fuels Corporation (NUFCOR) for calcining and packaging. Fig. 55 illustrates the uranium recovery process in operation at Vaal Reefs South.

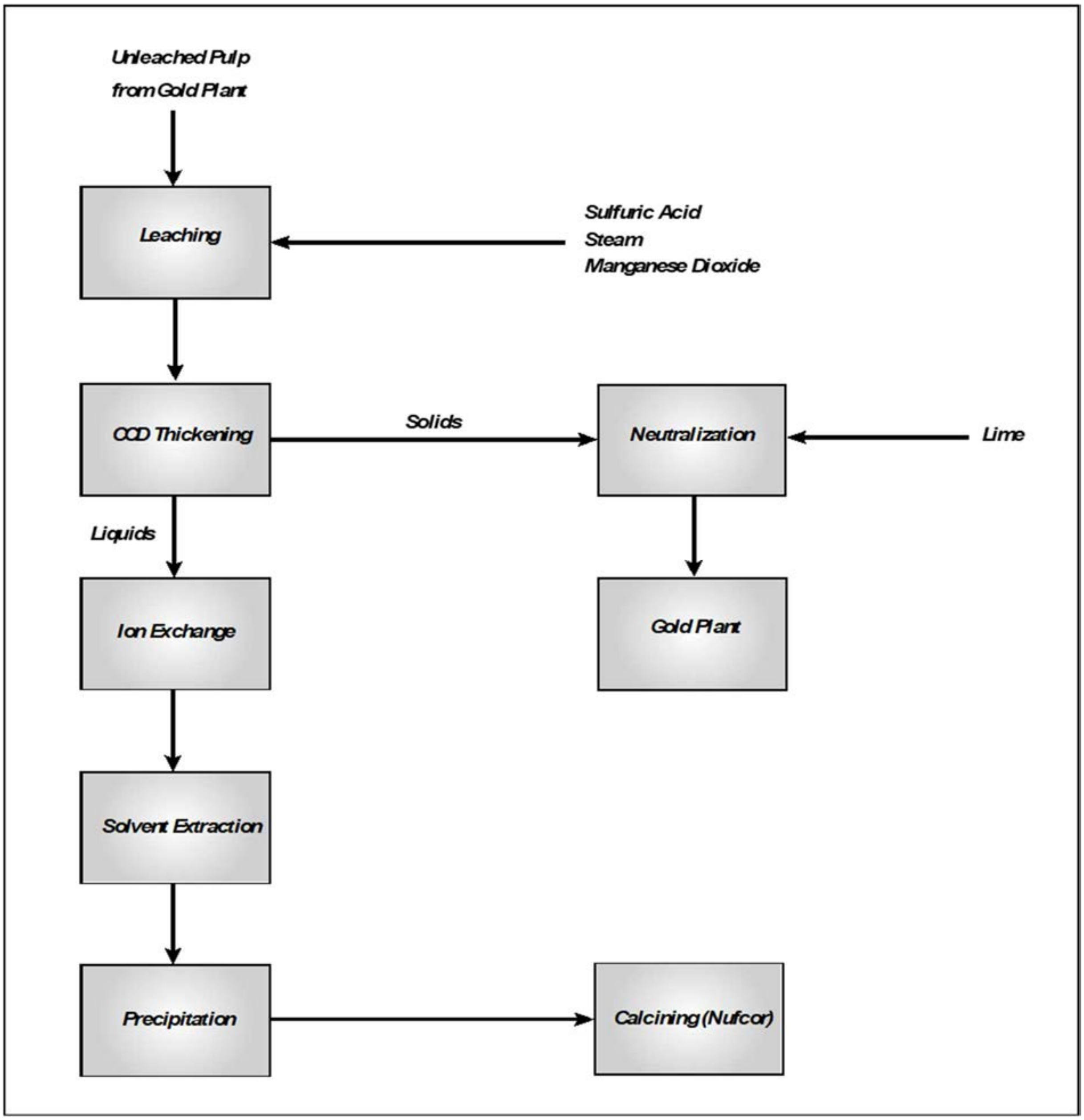

FIG. 55. Flow diagram for uranium recovery at Vaal River (courtesy T. Pool).

Production data

During its production history from 1953 through 2014, the Vaal Reefs complex has produced over 50 894 tU. Annual production from each of the three plants is shown in Fig. 56.

Table 30 lists the uranium production statistics for Vaal Reefs during the 1980s, which indicate an average yield of about 0.17 kgU/t of slimes treated with profitability ranging from a low of about US $8.41/kgU (US $3.81/lb U) to a high of over US $83.95/kgU (US $38/lb U). Average profitability during this period amounted to roughly US $27.98/kgU (US $12.72/lb U).

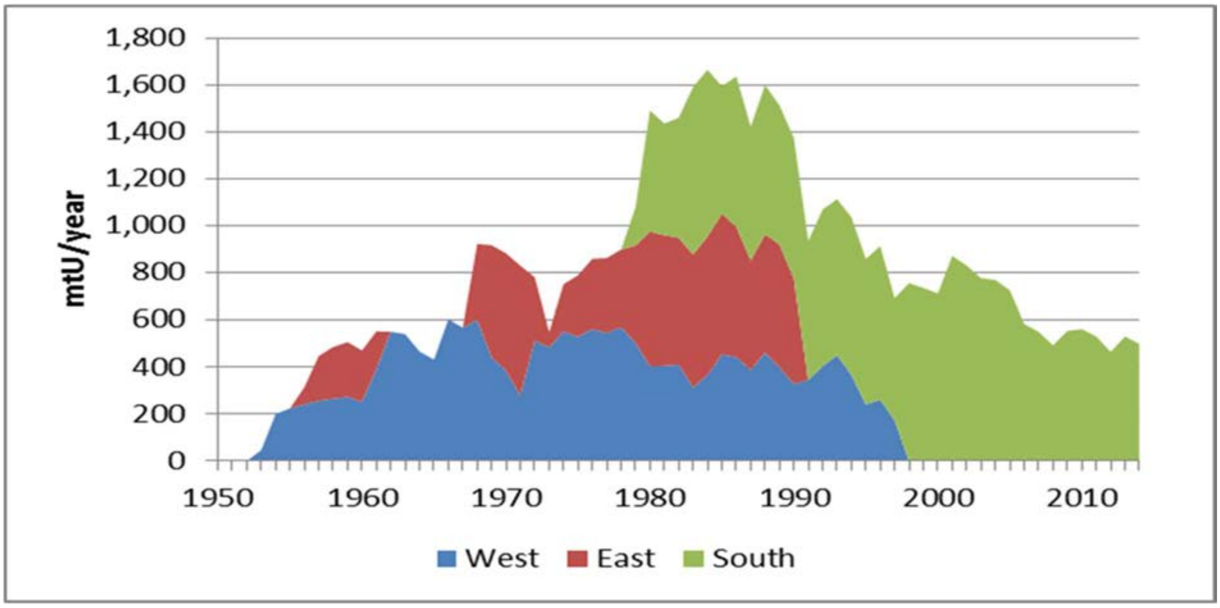

FIG. 56. Historical uranium production at Vaal Reefs (courtesy T. Pool).

TABLE 30. OPERATING STATISTICS FOR VAAL REEFS (1984–1990)

Year	Qtr	Slimes treated (1000 t)	U production (kg)	U yield (kg/t)	U profit (R 1000)	Exchange rate (R/US$)	U profit (US$/lb U)	U Profit (US$/kgU)
1984	1	2 149	411 149	0.20	11 115	1.475	7.05	15.54
	2	2 155	434 128	0.20	8 316	1.475	4.99	11.02
	3	2 131	414 617	0.20	8 033	1.475	5.05	11.14
	4	2 106	404 711	0.20	6 089	1.475	3.93	8.66
1985	1	2 272	444 794	0.20	38 079	2.229	14.77	32.56
	2	1 772	307 486	0.17	37 819	2.229	21.23	46.79
	3	2 283	436 338	0.20	33 237	2.229	13.14	28.98
	4	2 246	407 172	0.18	28 657	2.229	12.14	26.77
1986	1	2 276	416 050	0.19	18 490	2.285	7.48	16.49
	2	2 328	411 038	0.18	35 573	2.285	14.57	32.12
	3	2 346	425 933	0.18	34 107	2.285	13.48	29.72
	4	2 205	383 656	0.18	22 652	2.285	9.94	21.91
1987	1	2 303	394 184	0.17	58 582	2.036	28.08	61.9
	2	2 264	375 934	0.17	76 155	2.036	38.27	84.37
	3	1 899	295 282	0.15	63 447	2.036	40.59	89.49
	4	2 144	356 541	0.17	20 565	2.036	10.9	24.02
1988	1	2 246	375 910	0.17	8 565	2.273	3.86	8.51
	2	2 319	404 708	0.18	10 091	2.273	4.21	9.29
	3	2 323	408 312	0.18	21 370	2.273	8.85	19.52
	4	2 278	410 602	0.18	29 256	2.273	12.06	26.58
1989	1	2 238	369 940	0.16	15 179	2.623	6.02	13.27
	2	2 298	361 859	0.16	n.a.[a]	2.623	n.a.	n.a.
	3	2 328	388 915	0.17	n.a.	2.623	n.a.	n.a.
	4	2 331	392 770	0.17	n.a.	2.623	n.a.	n.a.
1990	1	2 275	364 587	0.16	n.a.	2.587	n.a.	n.a.
	2	2 284	355 822	0.15	13 097	2.587	5.47	12.06
	3	1 944	354 378	0.18	n.a.	2.587	n.a.	n.a.
	4	n.a.	n.a.	n.a.	n.a.	2.587	n.a.	n.a.

[a] n.a.: not available.

4.8.1.5. Buffelsfontein

The Buffelsfontein gold/uranium operation located in north-central South Africa, 170 km south-west of Johannesburg and 10 km south-east of Klerksdorp. The Vaal River, which forms the boundary between Transvaal Province to the north and Orange Free State Province to the south, also forms the southern boundary of the Buffelsfontein mining lease.

The Buffelsfontein uranium circuit was commissioned in July 1957 and operated continuously through 1991, at which time production was suspended due to the depressed market price for uranium. During the period 1957–1988, the Buffelsfontein uranium circuit treated 77.4 million t of feedstock and recovered 14.65 million kg of U, giving an average yield of about 0.192 kgU/t of ore treated. Total uranium production at Buffelsfontein from 1957 through 1991 amounted to 15.46 million kg of U. Maximum annual production of 653 809 kgU was achieved in 1970 (Fig. 57).

Uranium reserves remaining at Buffelsfontein in 1990 were reported to be 5.1 million t of gold/uranium ore at an average in situ grade of 0.28 kgU/t, or 1.44 million kgU. At an average recovery rate of 75%, the recoverable uranium content would be 1.081 million kgU.

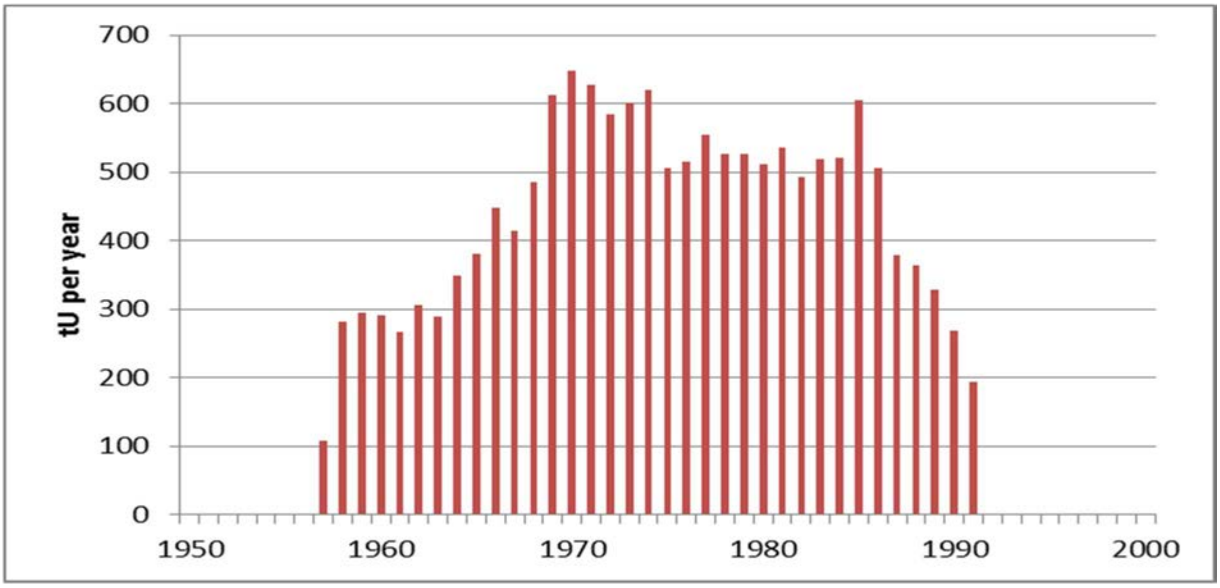

FIG. 57. Buffelsfontein production history (courtesy T. Pool).

Mining

The Buffelsfontein mine was operated by the Gencor Group and is located in the Witwatersrand Basin. As of 1990, the mine had been in operation for over thirty years and was accessed by four vertical shafts from the surface. By the mid-1980s, the Strathmore shaft system, consisting of two new subvertical shafts, was completed. The upcast shaft, 6.7 m in diameter, extends from a depth of 2173 m below the surface (on level 23) to level 37 at a depth of 3178 m. The downcast shaft, 9 m diameter extends from level 13 (1500 m below the surface) to level 37 (3178 m below the surface). The subshafts are interconnected. Ore was hoisted to level 13 and then trammed by trolley locomotive some 2.1 km to the Eastern shaft for hoisting to the surface. Mining was conducted at the rate of approximately 220 000 t/month [102].

The Vaal Reefs area of the Buffelsfontein mine was accessed by the two subvertical shafts and was ventilated by fans at the Eastern shaft (two 4100 kW motors providing 600 m^3/s air at 8.3 kPa). By the time the air reached level 28 the temperatures were 30.5°C wet bulb and 43.5°C dry bulb. Eight 3500 kW refrigeration plants located underground provided 28 MW of cooling and an additional 20 MW of cooling was provided for mid-shaft cooling from a surface refrigeration plant installation providing chilled water [103].

The area mined in the late 1980s had an average dip of 45° and a thickness of 0.8–0.9 m. Stoping width was approximately 1.40 m and mining continued from level 26 downwards, some 2353 m below the surface.

Milling

A generalized flowsheet of the milling process performed at Buffelsfontein is shown in Fig. 58. After primary crushing to 15 cm, a Sortex RM191 radiometric ore sorter was employed to reject waste rock. The Buffelsfontein plant had a total ore throughput of about 260 000 t/month, after sorting. Prior to the installation of a sorting plant, hand sorting was utilized to reject about 20% of the run-of-mine ore. The sorting plant consisted of four six-channel sorters which were designed to reject material grading less than 0.0021% U. The primary purpose was to enhance the gold grade, which is generally roughly proportional to the uranium grade.

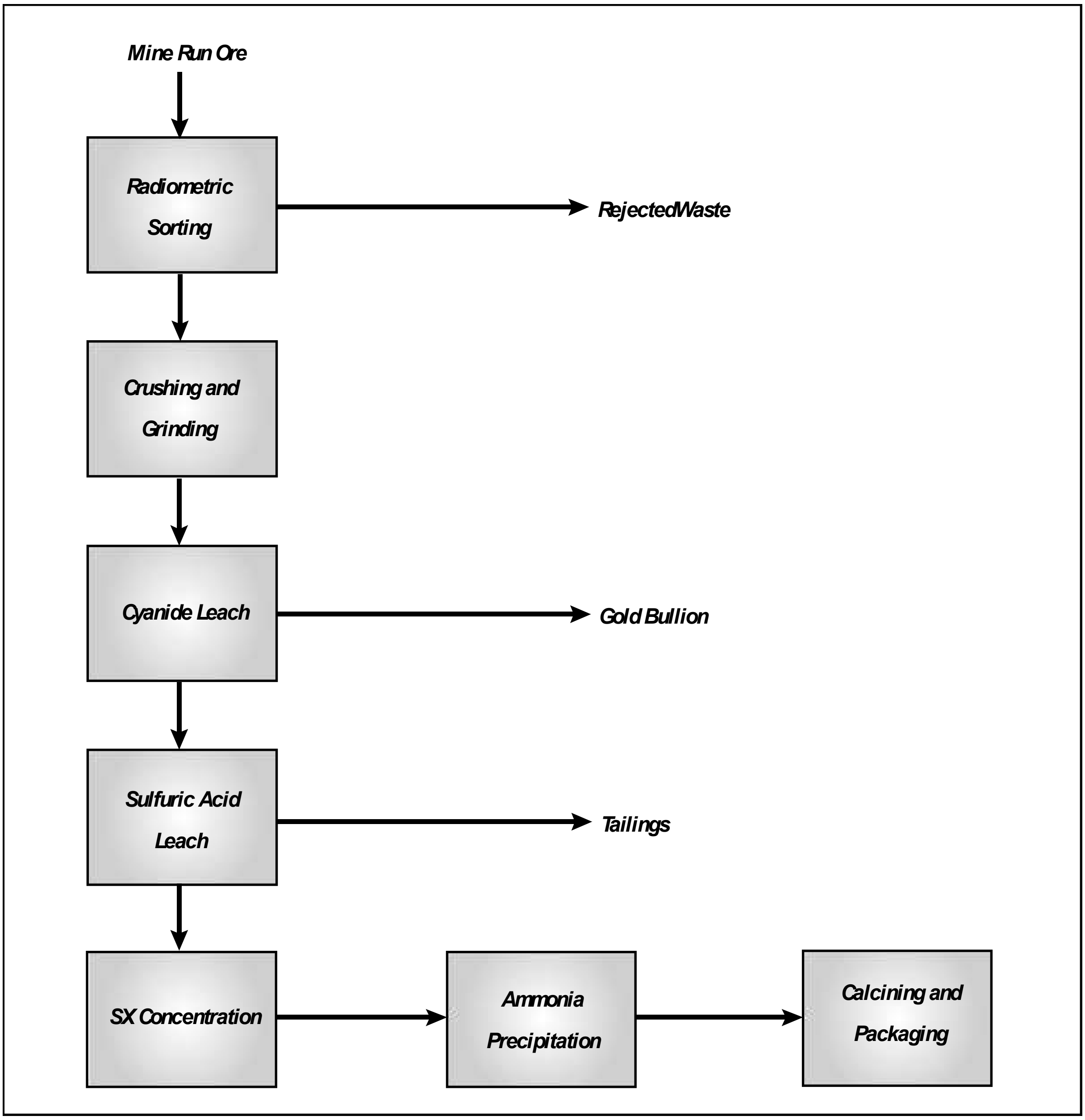

FIG. 58. Buffelsfontein mineral processing flowsheet (courtesy T. Pool).

After secondary and tertiary crushing, ore was ground in both rod and tube mills to a pulp having a particle size in range 100–200 mesh. Pulp was thickened to about 50–60% solids and pumped to agitators where it was aerated and leached with an alkaline cyanide solution (calcium cyanide and lime). Retention time was 24–30 h. The pulp underwent two stages of filtering; the first producing a pregnant gold solution, the second a feedstock for the uranium extraction circuit. The pregnant solution was clarified and de-aerated. The gold content was precipitated with zinc dust and the resultant precipitate was filtered, dried, calcined and then fluxed with a mixture of borax and silica and melted in a furnace to produce doré bullion (88–90% gold, 8–10% silver) and slag. Doré was refined to gold and silver bars in the Rand refinery. Gold production was approximately 1300 kg/month [104].

Washed tailings from the gold circuit were leached in Pachuca tanks with sulphuric acid and manganese dioxide as an oxidant at a temperature of 50°C. Bacteria was used in the leaching circuit for the oxidation of ferrous iron. This Bacfox system reduced the amount of sulphuric acid and manganese dioxide used and improved uranium recovery [105].

Leached pulp was filtered by rotary drum filters with the solids becoming the final plant tailings. Filtrate (pregnant uranium solution) was clarified in an Aquazur circulator classifier (of French design), yielding a clarified effluent containing approximately 80 ppm of suspended solids; an improvement compared with the previously used sand filters and resulted in a reduction of losses of organic solvent [106].

In 1968, SX of the uranium replaced the original IX system previously used. A tertiary amine solvent and controlled ammonia strip (with ammonium sulphate recycle) were used in four extractions, five scrubbing and four stripping stages. After precipitation with ammonia, ammonium diuranate was thickened, filtered and calcined. The resulting yellowcake was crushed and packed into steel drums for shipping. Approximately 31.38 tU were produced monthly.

Production data

Operating statistics for the period 1984–1991 are summarized in Table 31 and typify the processing results at Buffelsfontein. Table 32 shows the distribution of revenue over the same time period.

TABLE 31. OPERATING STATISTICS FOR BUFFELSFONTEIN (1984–1991)

Year	Quarter	Slimes treated (t)	U produced (kg)	U yield (kg/t)
1984	1	777 000	120 441	0.155
	2	781 000	127 226	0.163
	3	788 000	129 771	0.165
	4	769 000	142 918	0.186
1985	1	806 000	154 368	0.192
	2	817 000	145 462	0.178
	3	792 000	140 797	0.178
	4	823 000	164 546	0.200
1986	1	704 000	135 708	0.193
	2	751 000	115 352	0.154
	3	733 000	139 101	0.190
	4	710 000	116 200	0.164
1987	1	615 000	104 326	0.170
	2	676 000	94 148	0.139
	3	723 000	92 451	0.128
	4	732 000	88 210	0.121
1988	1	707 000	82 273	0.116
	2	662 000	94 996	0.143
	3	638 000	94 996	0.149
	4	584 000	91 603	0.157
1989	1	562 000	89 059	0.158
	2	582 000	83 121	0.143
	3	585 000	84 818	0.145
	4	596 000	72 095	0.121
1990	1	587 000	72 943	0.124
	2	596 000	67 006	0.112
	3	569 000	67 006	0.118
	4	556 000	61 306	0.110
1991	1	536 000	54 555	0.102
	2	512 000	52 112	0.102
	3	518 000	51 739	0.100
	4	249 000	37 320	0.150

TABLE 32. DISTRIBUTION OF REVENUE AT BUFFELSFONTEIN

Year	U (%)	Gold (%)
1987	50	50
1988	44	56
1989	41	59
1990	34	66
1991	31	69

4.8.1.6. Ergo

ERGO was a publicly traded company managed and administered by Anglo American Corporation. ERGO's operations located in north-eastern South Africa in the eastern suburbs of Johannesburg, within 50 km of the city centre. ERGO recovered gold, uranium and sulphuric acid from previously untreated tailings of gold mining and milling operations in the Witwatersrand mining area, near the cities of Benoni, Brakpan and Springs.

History

The ERGO project was conceived in the early 1970s following a sampling programme conducted on the tailings piles of Witwatersrand gold producers. As a result of this programme, Anglo American constructed a 1.0 t/h flotation pilot plant to determine the technical and economic feasibility of extracting uranium, pyrite and additional gold from these tailings. The pilot plant commenced operation in September 1974 and operated until May 1979 on samples acquired from various tailings piles [107].

Construction of the main recovery plant began in 1976 and commissioning was completed in 1978, with a tailings treatment capacity of 18 million t/year. Capital costs were R 125 million (US $115 million) [108].

In September 1981, ERGO built a second 1.0 t/h pilot plant to investigate the possibility of additional gold recovery using a carbon-in-leach process. The results of this test work were also positive and in December 1984 ERGO began construction of a 24 million t/year carbon-in-leach plant. This plant was commissioned in April 1985 [109]. Capital costs for this plant were R 63 million (US $43 million) [108].

In its 1990 Annual Report, ERGO announced that a recently completed strategic planning exercise indicated that certain elements of the present configuration would no longer be economically viable at prevailing product prices. As a result, the uranium plant was closed in April 1991. Total uranium production at ERGO from 1978 through 1991 amounted to 2578 tU. Maximum output of 302 tU was achieved in 1981.

Resources

ERGO acquired the rights to a series of slimes (tailings) dams in the East Rand. These dams and their contents were the result of historical gold mining operations. Initial project resources were contained in nineteen separate areas totalling 404.1 million t averaging 0.54 g/t gold, 46 g/t uranium and 0.98% sulphur [110]. Resources remaining as of 31 March 1981 are summarized in Table 4.5 and consist of 298 million t averaging 0.044 kg/t U3O8 (0.05 kg/t U)

According to Dahlkamp, uranium mineralization in the Witwatersrand Basin consists mainly of “uraninite with local concentrations of uranothorite, brannerite or other U–Ti phases (leucoxene), thucholite, coffinite and minor amounts of hexavalent uranium minerals” [111]. Resources remaining as of 31 March 1981 are summarized in Table 33.

TABLE 33. REMAINING SLIMES DAMS SCHEDULED FOR MINING BY ERGO
(as of 31 March 1981)

Slimes dams	Available material (10^6 t)	Grade			Content		
		Gold (g/t)	U (kg/t)	Sulphur (%)	Gold (kg)	U (tU)	Sulphur (10^6 t)
Springs 1	12.3	0.63	0.041	1.01	7 507.8	424	0.12
Springs 3	2.6	0.84	0.058	0.99	2 116.0	127	0.03
Springs 4	6.3	0.54	0.036	0.77	3 296.1	190	0.05
Springs 5	4	0.52	0.053	1.24	2 015.3	178	0.05
Springs 6	3.2	0.47	0.043	1.14	1 457.2	117	0.04
Brakpan 1	37.5	0.66	0.029	1.16	23 979.6	915	0.44
Brakpan 2	6.9	0.53	0.031	1.21	3 543.2	178	0.08
New State Areas	11.5	0.62	0.040	0.99	6 908.1	388	0.11
GGMA 1	24.2	0.62	0.029	1.02	14 537.0	590	0.25
GGMA lower apron	3.2	0.60	0.031	0.48	1 860.2	82	0.02
GGMA 2	14.6	0.43	0.047	0.80	6 082.6	576	0.12
GGMA 3	13.6	0.39	0.046	0.69	5 138.9	527	0.09
Geduld 1	36	0.57	0.022	0.98	19 881.3	672	0.35
Geduld 2	1.6	0.54	0.037	0.49	837.1	51	0.01
Modderfontein East	42.7	0.52	0.031	0.89	21 512.9	1 134	0.38
SA Land 1	33.7	0.39	0.050	1.28	12 733.9	1 427	0.43
SA Land 2	13.3	0.37	0.042	1.14	4 767.8	478	0.15
East Geduld 2	8.5	0.20	0.039	0.65	1 647.1	281	0.06
Can Dyk	22.4	0.26	0.054	0.89	5 642.7	1 029	0.20
Total/average	298.1	0.51	0.040	0.94	145 464.8	9 364	2.98

Mining

In 1978, ERGO began retreatment of the East Witwatersrand tailings just east of Johannesburg. At a processing rate of 1.5 million t/month, the project had an estimated life of 20 years.

The tailings were reclaimed by hydraulic monitors. Normally, three sites were being reclaimed simultaneously by three or four monitors at each site. The monitors directed high pressure (2.0 MPa) water jets at the working face to produce a water/tailings mixture (slurry). Water requirements corresponded to about 1.0 m^3 of water per tonne of tailings.

The resultant slurry was screened and pumped to the flotation plant. Two main pipelines serviced an area, one for recycle and make-up water, the other for slurry. Water for each monitor (40–60 mm nozzles) was furnished by pumps at the rate of 800 m^3/h. Slurry was pumped a distance of 10–20 km depending on the location being reclaimed.

Milling

Reclaimed tailings were pumped to a flotation plant where a pyrite concentrate was recovered. This concentrate, representing only 3% of the original feed, was then leached for its uranium content. Barren flotation tailings were pumped to a disposal area. After uranium was recovered, pyrite concentrate was roasted to produce sulphuric acid and calcines, which were then treated in a cyanidation plant to recover gold [112].

At the flotation plant, slurry was treated in three recovery lines. Major items of equipment in each line included three air agitated Pachuca tanks, a rotary pulp distributor, two banks of 18 flotation rougher cells and two banks of flotation cleaner cells. The slurry was acidified to a pH of 3.5–4.0. Reagents used include: sodium mercaptobenzothiazole (90 g/t), dithiophosphate (collectors) (10 g/t), Dowfroth 200 (frother) (17 g/t) and copper sulphate (20–25 g/t).

Typical concentrate averaged:

- 28–30% sulphide/sulphur (recovery 78–82%);
- 8–10 g/t gold (recovery 40–45%);
- 254–339 g/t U (recovery 20–30%).

The concentrate (3% of the original feed) was thickened to 56% solids and pumped to the uranium plant. Flotation tailings (97% of the feed) were neutralized with lime, thickened to 50% solids and pumped about 10 km to a disposal site.

Uranium recovery

The uranium recovery plant consisted of one pyrite storage Pachuca tank, three leach Pachuca tanks (each 10.6 m diameter × 17.72 m high) and a residue Pachuca tank (8 m diameter × 13.72 m high). The acid leach (requiring 45 kg sulphuric acid per tonne of feedstock) at a temperature of 55–60°C had a retention time of 26 h. Leach efficiency was about 85%. Liquid–solid separation was affected by eight drum filters. Solids were sent to roasters; the resulting pregnant uranium solution (350 mg/L U) was clarified prior to SX.

The SX plant consisted of three extraction stages, four scrubber stages and four stripper stages in tank type mixer–settler units. The organic phase consisted of 5% Alamine and 3% Isodeconal dissolved in a paraffin solvent of Pegasol 3445. The loaded organic phase contained about 3.5 g/L U. During scrubbing, ammonium hydroxide was added for pH adjustment. In the stripping phase (containing 130–140 g/L ammonium sulphate), uranium was stripped from the organic phase into the aqueous phase to a concentration of 5.9–6.7 g/L U.

Following pH adjustment, uranium was precipitated from the aqueous solution to produce 97–98% ammonium diuranate. The precipitate was thickened and dewatered in centrifuges. Uranium recovery was very low, averaging approximately 20%, which was mainly attributable to uranium minerals not being recovered by the flotation process.

The ammonium diuranate was shipped by tank trucks to NUFCOR, a calcining facility operated by the Chamber of Mines on behalf of its member companies.

Wet residue (pyrite concentrates) from the uranium plant was fed at 67% solids to three Lurgi fluid bed roasters. Off-gas was treated in two double catalytic/double contact Lurgi acid plants; one a nominal 1000 t/d unit, the other a 500 t/d plant with an oleum (fuming sulphuric acid) facility.

Gold recovery

Roaster calcines were treated in the gold recovery plant. Originally, the plant was a standard gold cyanidation plant using zinc dust to precipitate gold from pregnant solution.

Roaster calcines were ground in two ball mills (each 2.74 m × 3.05 m), then leached in five Pachuca tanks (654 m^3 capacity each) followed by treatment in eight rotary drum filters. Filtered solids were repulped and pumped to a separate waste disposal area. The filtrate (pregnant solution) was clarified and the gold content precipitated. The precipitate was acid washed, filtered, calcined and smelted to produce doré bullion.

In 1985, the plant was converted to a carbon-in-leach plant using existing plant facilities except for precipitation. Activated carbon made from coconut shells was added to the cyanide leach pulp. As the gold was being leached by the cyanide solution, it was being simultaneously adsorbed by carbon particles. Loaded carbon was recovered by screens, and then eluted by a strong cyanide solution. The solution was passed through electrowinning cells where gold was precipitated onto steel wool. The resulting precipitate and steel wool were smelted to produce doré bullion. It was reported that this process reduced dissolved losses and resulted in an improvement in gold recovery. A simplified flowsheet of the ERGO mineral recovery process is presented in Fig. 59.

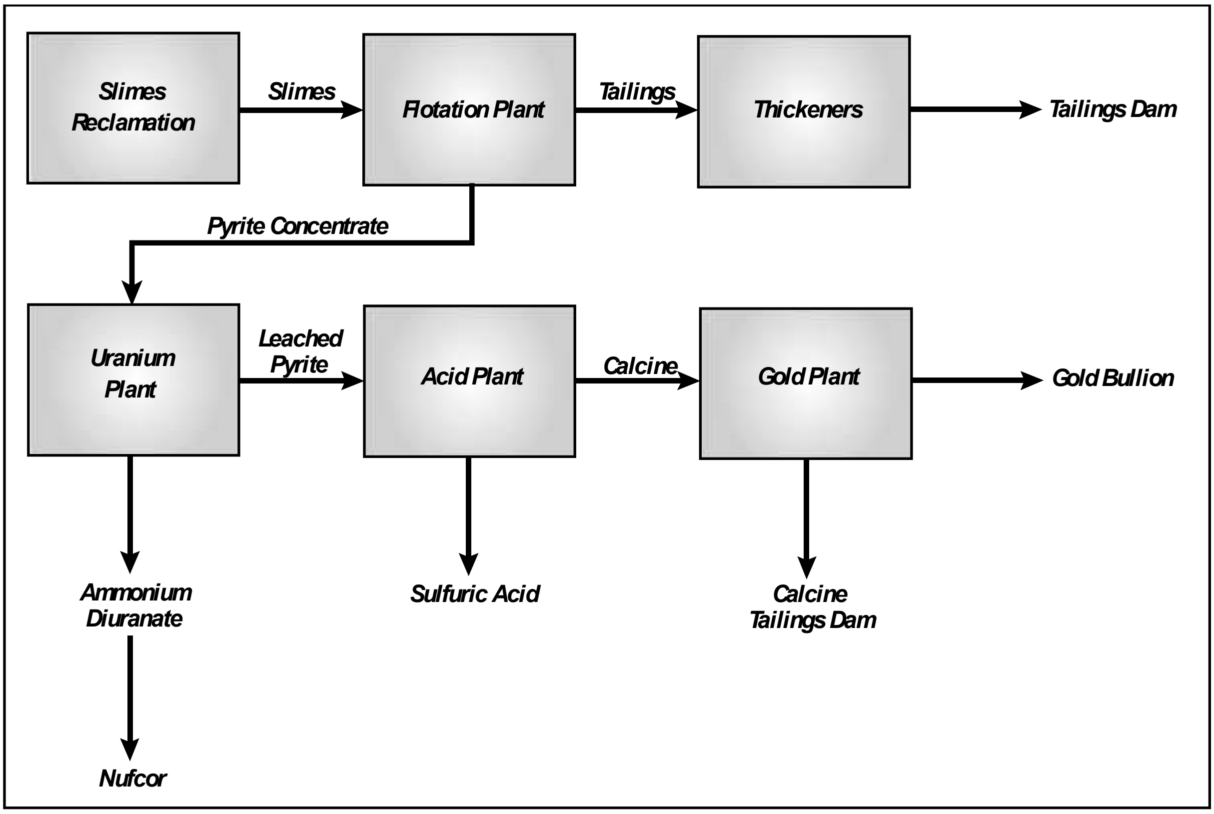

FIG. 59. ERGO simplified process flowsheet (courtesy T. Pool).

Production data

Uranium production at ERGO totalled 992 tU. Annual production is shown in Fig. 60. Detailed production information on uranium, gold and sulphuric acid for the period 1984–1990 is presented in Table 34.

Economic analysis

Financial information for the period 1984–1990 is presented in Table 35 [113]. Assuming nominal uranium prices for the period 1984–1990, distribution of revenue between the various products can be calculated, as shown in Table 36.

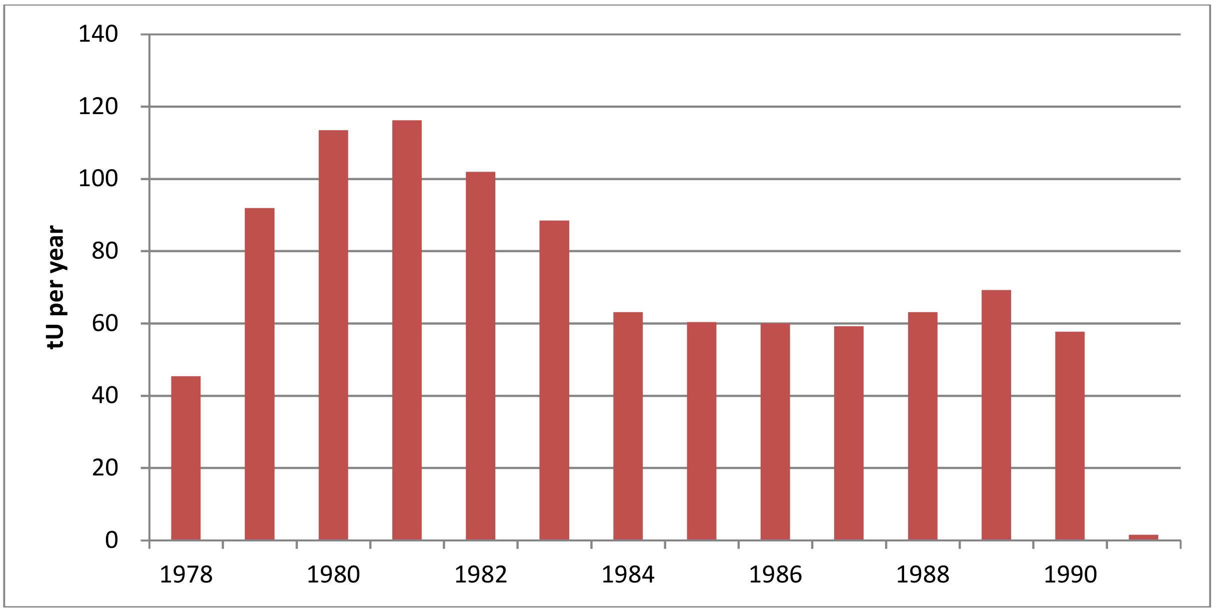

FIG. 60. Uranium production at ERGO (courtesy T. Pool).

TABLE 34. PRODUCTION DETAILS AT ERGO (1984–1990)

Year	Throughput (1000 t)	U (t)	Gold (kg)	Sulphuric acid (t)
1984	19 014	63	5 068	456 584
1985	18 231	60	6 522	452 665
1986	20 022	60	6 835	471 998
1987	22 350	59	8 708	503 441
1988	34 949	63	11 026	500 507
1989	37 437	69	11 992	541 614
1990	37 092	58	11 732	491 453

TABLE 35. ECONOMIC SUMMARY FOR OPERATIONS AT ERGO (1984–1990)

Year	Throughput (1000 t)	Total cost (R 1000)	Unit cost (US $/t)	Operating profit (R 1000)	Revenue	
					Gold/silver (R 1000)	Sulphuric acid/U (R 1000)
1984	19 014	59 097	2.11	57 572	85 943	26 055
1985	18 231	94 189	2.32	87 967	149 512	27 682
1986	20 022	125 568	2.74	103 369	184 825	37 406
1987	22 350	171 344	3.77	116 730	240 385	40 440
1988	34 949	264 259	3.33	109 136	327 887	38 590
1989	37 437	317 886	3.24	133 393	386 286	55 114
1990	37 092	340 027	3.54	90 701	382 697	40 730

TABLE 36. NOMINAL REVENUE DISTRIBUTION AT ERGO

Year	U_3O_8 (US $/lb)	U_3O_8 (US $1000)	Gold (US $1000)	Sulphuric acid (US $1000)	Revenue distribution		
					U (%)	Gold (%)	Sulphuric acid (%)
1984	17.27	2 837	58 255	17 991	3.6	73.7	22.7
1985	15.60	2 448	67 085	12 200	3.0	82.1	14.9
1986	17.12	2 667	80 885	16 638	2.7	80.7	16.6
1987	18.15	2 806	118 066	20 616	2.0	83.4	14.6
1988	14.74	2 416	144 223	17 601	1.5	87.8	10.7
1989	10.00	1 803	147 287	22 978	1.0	85.6	13.4
1990	9.76	1 461	147 913	17 103	0.9	88.8	10.3

4.8.1.7. Joint Metallurgical Scheme (JMS)

The JMS gold/uranium recovery project was established in east-central South Africa, near the city of Welkom, 235 km south-west of Johannesburg, in Orange Free State Province. Mining leases which contained tailings dams processed at JMS extended some 25 km to the north and south of Welkom. These mining leases include Free State Geduld, Free State Saaiplaas, President Brand, President Steyn, Welkom and Western Holdings.

The JMS, at the time of implementation, was owned and operated by Free State Consolidated Gold Mines Limited (‘Freegold’) for the benefit of several of the company’s mining projects. Freegold was managed, administered and 30.5% owned by Anglo American.

History

The JMS was commissioned in 1977 and was designed to treat a combination of pyrite concentrates, old tailings and current tailings from a series of six mines: President Brand, President Steyn, Welkom, Western Holdings, Free State Geduld and Free State Saaiplaas. Capital costs were R 88 million [114].

This enterprise was an expansion of the existing President Brand uranium plant which was completed in 1971 at an initial capital cost of R 9.0 million. This plant had a treatment capacity of 180 000 t/month but did not commence operations until 1976 owing to unfavourable market conditions. By 1980, this capacity had been expanded to 500 000 t/month.

Deteriorating market conditions forced a decrease in production in April 1990 and production was suspended indefinitely in September 1990.

Resources

Resources available to JMS, as of 30 June 1980, are summarized in Table 37.

TABLE 37. JMS RESOURCES
(as of 30 June 1980)

Company	Ore reserves							
	Quantity (10^6 t)	Gold (g/t)	U(kg/t)	Reserves (tU)	Quantity (10^6 t)	Gold (g/t)	U (kg/t)	Resource (tU)
Free State Geduld	10.327	13.38	0.08	786	53.649	0.403	0.07	3 632
Free State Saaiplaas	2.917	5.04	0.16	469	16.97	0.289	0.09	1 580
President Brand	11.458	10.67	0.08	873	33.085	0.64	0.1	3 359
President Steyn	17.286	8.9	0.09	1 609	48.961	0.555	0.079	3 853
Welkom	8.523	6.84	0.08	721	44.208	0.358	0.089	3 928
Western Holdings	12.583	10.51	0.06	745	46.84	0.428	0.07	3 171
Total/average	63.094	9.82	0.082	5 203	243.713	0.481	0.08	19 522

Mining

Mining operations related to Anglo American's JMS project include slimes recovery from previously deposited tailings and slimes treatment of current tailings. Both types of tailings are the result of the mining and processing of gold ore and both are obtained, ultimately, from conventional, deep, underground mining operations conducted on thin 'reefs' of quartz pebble conglomerate containing both gold and uranium.

Currently, generated tailings (or slimes) are pumped directly from the gold processing circuit to the flotation plant.

Deposited tailings were recovered from existing tailings dams by sluicing with remotely controlled high-pressure water monitors. The slimes were washed away from the working face as a slime–water slurry which gravitated to a primary pumping station adjacent to the dam. The slurry was screened and pumped to the flotation plant.

Milling

The JMS came into production in 1977 primarily to reprocess the old tailings of six mining companies of the Anglo-American Group in Orange Free State [115].

Historical and current low grade tailings are treated in three pyrite flotation plants: (i) President Brand (350 000 t/month), President Steyn (570 000 t/month) and Free State Geduld (570 000 t/month). The three plants have an aggregate treatment rate of 1.49 million t/month.

Pyrite flotation concentrates and the highest grade uranium-bearing historical and current tailings were treated in the President Brand uranium plant at the rate of 500 000 t/month.

Uranium production was centralized in the President Brand uranium plants, which consisted of a tailings leach plant and a separate pyrite concentrate leach plant. As previously noted, pyrite concentrates are produced in three separate flotation plants.

Feed to the flotation plants consisted of low grade slimes from old tailings piles and from current tailings from existing gold recovery plants. Feed to each flotation plant was acidified to pH4 in Pachuca tanks for 5–6 h conditioning to counteract the depressant effect of residual lime and cyanide in gold plant residues.

Sodium mercaptobenzothiazole was used as a collector and triethoxybutane as a frother. Copper sulphate was used as an activator for the flotation of pyrite. Pyrite concentrates represented only 2–3% of flotation plant feed. Concentrates from President Steyn and Free State Geduld were trucked to the President Brand plant. Tailings were thickened through water recovery in 105 m diameter thickeners and the solids sent to new tailings sites.

The uranium plant at President Brand treated combined pyrite concentrates from three flotation plants and also high grade uranium tailings. Tailings (300 000 t/month) were washed and filtered to remove impurities from the gold plants, repulped and fed to air agitated Pachuca tanks. Sulphuric acid, manganese dioxide and steam were added during the 24 h retention period. Leached pulp was filtered and washed in drum filters. Filtrate (pregnant solution) was clarified and sent to the SX unit. Solids were repulped and sent to the flotation circuit for recovery of pyrite.

Pyrite concentrates (40 000 t/month) were treated in a separate circuit in air agitated Pachuca tanks similar to the tailings treatment section. Owing to the higher uranium content and lower tonnages involved, a longer, more intensive leach was justified. Leached pulp was filtered on belt filters; the filtrate joined pregnant solution from the tailings leach plant. Solids (pyrite) were sent to two fluid bed roasters operated at 800–850°C where they reacted with oxygen to form sulphur dioxide and iron oxide calcine. Cooled and cleaned sulphur dioxide was processed in a double catalysis contact section to produce sulphuric acid at the rate of 1000 t/d, providing acid for the uranium recovery plants.

The combined pregnant solution was clarified and processed by SX in four extraction units using a tertiary amine organic solvent, four scrubbing units and four stripping units where uranium was separated from the organic phase by an aqueous solution of ammonium hydroxide/ammonium sulphate under controlled pH conditions.

Purified uranium was precipitated with gaseous ammonia to form ammonium diuranate or yellowcake. Yellowcake was thickened and centrifuged and sent for calcining to a central plant operated by NUFCOR.

Calcined pyrite (now iron oxide) was reground in a ball mill and then leached in a carbon-in-pulp cyanide process for recovery of residual gold remaining from the original gold mining and milling operations. Leached calcine was disposed of along with President Brand flotation tailings. During the leaching process, the gold content was adsorbed by carbon. Loaded carbon was eluted in a strong cyanide solution at 90–110°C. Carbon was recovered and regenerated at 700°C for reuse. Eluted solution was treated in an electrowinning plant comprising four cells of Anglo American Research Laboratory design. Gold was precipitated onto steel wool and smelted to produce doré bullion [115].

The JMS developed a complex process maximizing uranium recovery with production of sulphuric acid and the retrieval of by-product gold which had remained unrecovered. The process is illustrated in Fig. 61.

Production data

Total uranium production at the JMS for the period 1977–1990 amounted to 7317 tU. Maximum output of 927 tU was recorded in 1981. Annual uranium production is illustrated in Fig. 62. Operating statistics for the period 1984–1991, as shown in Table 38, are examples of the processing results obtained at the JMS.

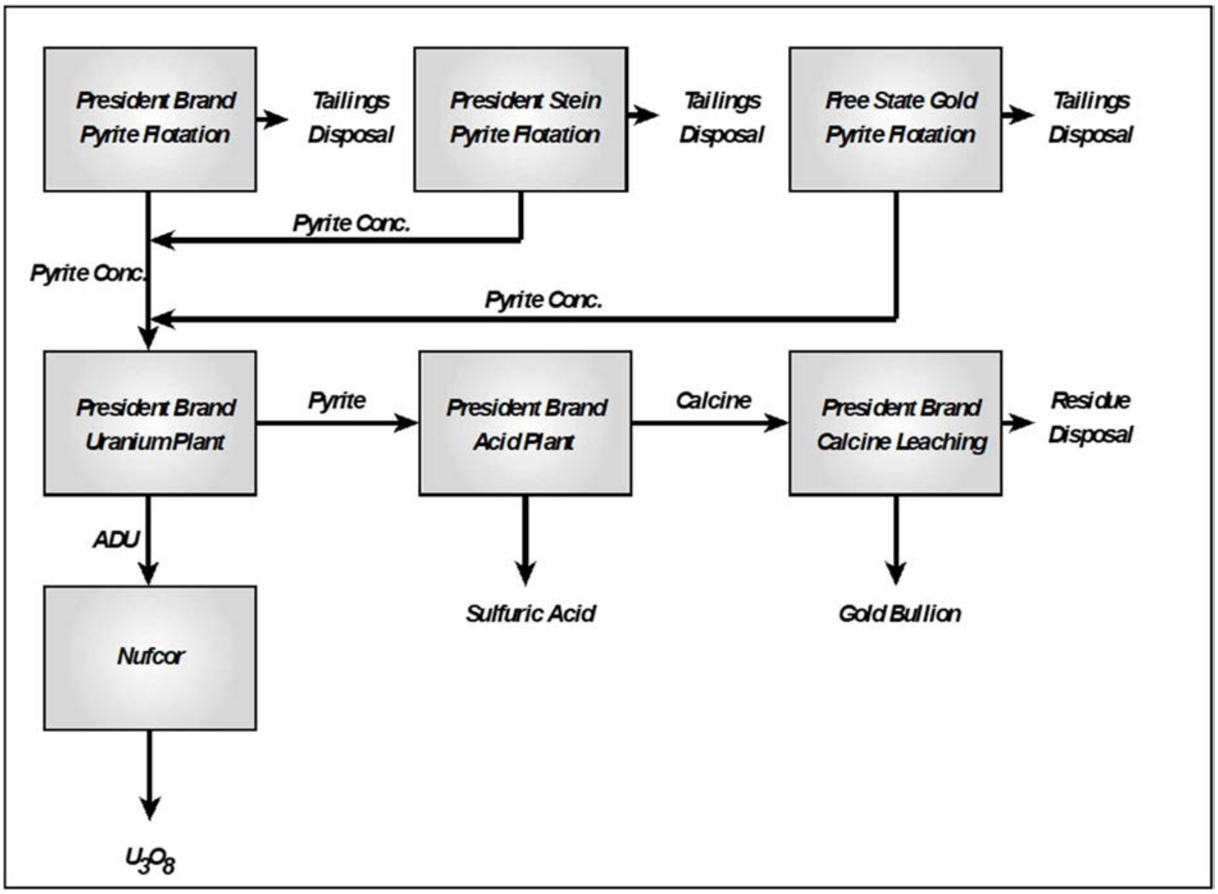

FIG. 61. JMS process flowsheet (courtesy T. Pool).

Economic analysis

Operating and financial results for the JMS for the fiscal year ending 1988 are shown in Table 39. Considering overall slimes treated and total costs of production, the unit cost of treating slimes as of 1988 was R 9.4/t, or about US $4.14/t.

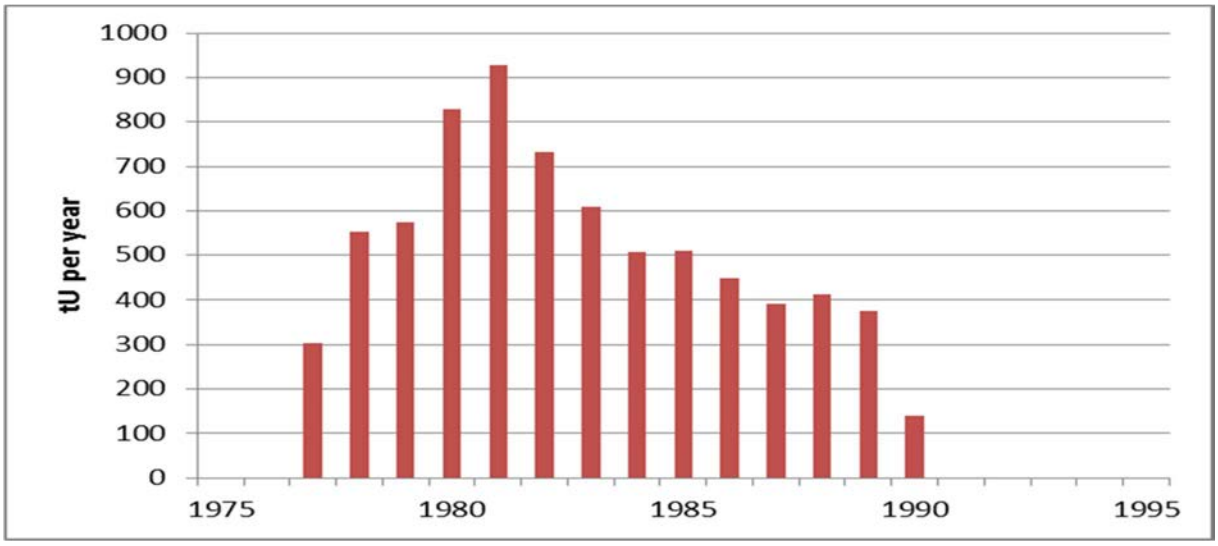

FIG. 62. JMS production history (courtesy T. Pool).

TABLE 38. JMS PRODUCTION STATISTICS (1984–1990)

Year	Qtr	Slimes treated (t)	U production (kg)	Uranium yield (kg/t)
1984	1	4 803 000	136 961	0.029
	2	3 661 000	101 821	0.028
	3	3 947 000	112 697	0.029
	4	5 526 000	154 701	0.028
1985	1	5 485 000	120 734	0.022
	2	5 502 000	136 912	0.025
	3	5 518 000	133 698	0.024
	4	5 292 000	119 347	0.023
1986	1	5 464 000	122 229	0.022
	2	5 454 000	117 484	0.022
	3	5 466 000	107 276	0.020
	4	5 260 000	102 384	0.019
1987	1	5 339 000	105 724	0.020
	2	4 770 000	94 149	0.020
	3	4 503 000	85 934	0.019
	4	5 071 000	106 717	0.021
1988	1	4 826 000	98 802	0.020
	2	5 379 000	109 500	0.020
	3	4 877 000	95 405	0.020
	4	5 280 000	108 422	0.021
1989	1	5 065 000	98 405	0.019
	2	5 171 000	86 242	0.017
	3	5 406 000	100 830	0.019
	4	5 205 000	90 887	0.017
1990	1	4 729 000	82 115	0.017
	2	3 905 000	32 812	0.008
	3	3 652 000	26 782	0.007
	4	n.a.[a]	n.a.	n.a.

[a] n.a.: not available.

TABLE 39. OPERATING AND FINANCIAL RESULTS FOR THE JMS

	Quarter ended September 1987	Quarter ended June 1988	Year ended September 1988
Pyrite flotation plants:			
Slimes treated (1000 t)	3 893	4 299	16 215
Head grade:			
Uranium (kg/t)	0.08	0.08	0.08
Sulphur (%)	0.99	0.96	0.97
Gold (g/t)	0.47	0.46	0.46
Uranium plant:			
Slimes treated (1000 t)	984	1080	3938
Concentrate treated (1000 t)	102	117	432
U produced (kg)	95 384	109 477	410 338
Sulphuric acid plant:			
Sulphuric acid produced (t)	58 728	96 649	343 833
Gold plant:			
Calcine treated (t)	51 137	84 783	298 893
Pyrite treated (t)	8 058	3 141	29 622
Gold produced (kg)	401	591	2 240
Revenue (R million)	41.7	43.9	198.9
Costs (R million)	34.0	33.9	152.4
Profit (R million)	7.7	10.0	46.5
Capital expenditure (R million)	0.5	0.2	2.5

Using nominal prices for gold and uranium for 1988, the distribution of revenue for the JMS for the fiscal year ending September 1988 can be calculated to be: gold (42.7%), sulphuric acid (35.9%) and uranium (21.3%).

4.8.1.8. Mine waste solutions

Mine Waste Solutions is a recently developed project focused on recovery of gold and uranium from historical tailings. Gold production commenced in 2003, but licensing delays and financial difficulties delayed the implementation of uranium recovery. Vaal River acquired the Mine Waste Solutions project from First Uranium in 2012 and commissioned the uranium recovery plant in late 2014.

The mineral reserve estimate from First Uranium's 2010 annual information form is shown in Table 40 [116].

TABLE 40. MINERAL RESERVE ESTIMATE FOR MINE WASTE SOLUTIONS PROJECT
(as of 1 January 2010)

Category	Tonnage (10^6 t)	Gold		Uranium	
		Grade (g/t)	Content (t)	Grade (kgU/t)	Content (tU)
Measured	201.720	0.253	51.038	0.064	12 858
Indicated	130.909	0.258	33.764	0.073	9 550
Inferred	16.517	0.306	5.055	0.084	1 390
Total	349.146	0.257	89.857	0.068	23 797

The life-of-mine plan envisaged a 16-year life for excavating and processing some 22 million t of tailings annually. The average gold grade is projected to be 0.28 g/t and the average uranium grade is expected to be 0.077 kg/t. 'Reverse' leaching will be utilized to maximize gold recovery at around 65%, while uranium recovery will be quite low, at about 23%, mainly due to poor recovery in the flotation circuit. Overall, gold production should be about 6220.7 kg/year while uranium production should average about 350 tU/year.

Total capital expenditures (in South Africa Rand (ZAR)) were expected to be approximately ZAR 3700 million, about US $530 million at the average 2010 exchange rate of ZAR 7.3 per US$. Operating costs were expected to be ZAR 39.35/t (US $5.39/t) of material handled. Operating costs were calculated after a cost allocation to either gold or uranium. All variable and fixed costs pertaining to mining, carbon-in-leach processing and tailings deposition were allocated to gold, and the flotation plant and uranium leach plant costs only to uranium costs. On this basis, the gold and uranium cash costs, respectively, were US $13 085/kg gold and US $ 69.19/kg U.

4.8.1.9. Cooke No. 4

Sibanye Gold's Cooke No. 4 (formerly Ezulwini) gold/uranium project in South Africa reactivated by-product uranium production in May 2014. This reactivation followed acquisition of the project from Gold One which had only recently acquired the project from First Uranium. First Uranium is estimated to have produced approximately 71 246 kgU during the period 2009–2012. The project includes a deep underground mine where both gold and gold/uranium ores are mined, a 200 000 t/month gold plant and a 100 000 t/month uranium plant.

Geologically, the project lies within the Witwatersrand Basin and is focused on quartz pebble conglomerates of the Middle and Upper and Ellsburg reefs of which the Upper unit contains mainly gold and the Middle unit contains both gold and uranium. Resources, as of 2010, are shown in Table 41 [117].

Mining operations are conducted by conventional up-dip drift and fill mining using cemented tailings for backfill support. Transport of broken ore to the shaft is affected by a combination of slusher, LHD and rail. Gold and gold/uranium ores are mined and handled separately.

Gold recovery from auriferous ores involves crushing, grinding, gravity beneficiation, cyanidation, carbon-in-leach concentration, electrowinning and refining. Uranium recovery from gold/uranium ore involves crushing, grinding, hot acid leaching, counter-current decantation, solid–liquid separation, IX and SX concentration, and ammonia precipitation. Ammonium diuranate concentrates are shipped to NUFCOR for calcining.

TABLE 41. MINERAL RESOURCES AT EZULWINI PROJECT (COOKE No. 4)
(as of 31 December 2010)

Classification	Stratigraphic unit	Reserve (1000 t)	Au grade (g/t)	U grade (%)	Contained Au (kg)	Contained U (t)
Measured:	Upper Ellsburg	1 732	8.39		14 525	
	Middle Ellsburg	1 131	6.03	0.057	6 812	643
Indicated:	Upper Ellsburg	7 499	6.10		45 722	
	Middle Ellsburg	2 873	5.39	0.065	15 490	1 876
Inferred:	Upper Ellsburg	45 869	5.45		249 885	
	Middle Ellsburg	7 737	6.43	0.075	49 734	5 773
Below 2500 m		105 075	4.70	0.064	493 861	66 823

Production at Cooke No. 4 during 2014 amounted to 4305 kg of gold and 69 251 kgU which, at nominal 2014 prices, indicates revenue distribution of 96.7% for gold and 3.3% for uranium. A 2013 report [118] on the Cooke No. 4 operations forecast, for the period 2014–2020, uranium production of about 102 000 kgU/year at a yield of about 0.364 kgU/t, which accounts for about 16% of total revenue.

4.8.2. Uranium–yttrium–thorium: Denison mine, Elliot Lake district (Canada)

4.8.2.1. History

The Denison mine in Elliot Lake, Canada, traces its origin back to April 1954 when S.B. Roman, the founder of Denison Mines, acquired 83 mining claims in the Elliot Lake district from the original holders. The initial drilling programme quickly outlined more than 140 million kg of U (>140 000 tU), making it one of the world's largest uranium deposits. Construction commenced in 1955 and the first uranium production was recorded in May 1957. Since that time, through to its closure in 1992, the project produced 56 655 tU, mainly through conventional mining, but also with a more recent contribution from underground leaching.

4.8.2.2. Geology

Uranium resources in the Blind River–Elliot Lake area occur in fluvial arkose, in quartzite, and in pyritic, uraniferous oligomictic conglomerate. These resources were substantial and accounted for the majority of Canadian uranium production prior to high grade discoveries being made in the Athabasca Basin.

The average recovered grade was about 0.1% U. Uranium mineralization consists principally of brannerite and uraninite, although thucholite is also present. The pyrite content of the ore is 6–10%. Thorium is present in the uraninite and monazite [119].

4.8.2.3. Mining

The Denison Mine at Elliot Lake was a deep (750 m) underground mine based on a room and pillar design. Mining was highly mechanized and the production capacity was 21 000 t/d. The mine also implemented both mine water recovery and bacteria enhanced underground leaching to supplement conventional production.

4.8.2.4. Uranium recovery

The original Denison mill commenced operation in May 1957, with a design capacity of 6000 t/d and about 1.92 million kgU/year. In 1978, under the terms of its new agreement with Ontario Hydro, Denison undertook a major mine/mill expansion programme which resulted in an increase in milling capacity to 15 000 t/d and about 3.81 million kgU/year.

Crushing and grinding facilities were located adjacent to the main hoisting shaft and consisted principally of a 8.5 m × 3 m SAG mill in closed circuit with hydrocyclones. The ore was ground to 50% minus 200 mesh and pumped 1 km to the hydrometallurgical recovery plant, where pumped slurry was thickened, agitated and heated to 75°C. Sulphuric acid and sodium chlorate were added in Pachuca tanks, which provided a residence time of 40 h. Solid–liquid separation was accomplished with hydrocyclones and rotary drum filters. Uranium was extracted in a fixed bed IX system and stripped from the resin with nitric acid. Ammonia precipitation was used to produce ammonium diuranate which was dried at about 145°C. Mill recovery was 94%.

4.8.2.5. Yttrium recovery

In the 1970s, Denison recovered yttrium from the uranium mill liquid waste stream using an old centrifugal method, but a technological breakthrough in the way yttrium was used with europium to produce a more brilliant picture in colour television picture tubes increased the efficiency of this application. However, owing to the change in application, demand for the rare earth dropped significantly. The market subsequently declined and production at Elliot Lake using a centrifuge became uneconomic.

In 1986, Denison, Molycorp Inc., Shin-Etsu Chemical Co. Ltd and Mitsui & Co. Ltd, as a joint venture group, completed construction of a plant adjacent to Denison's then existing facilities to produce yttrium oxide as a by-product of uranium processing. Capital cost of the plant was about C$10 million.

The yttrium plant, with a production capacity of 150 t/year of Y_2O_3, was designed in 1985–1986 and commissioned in September 1986. Recovery was expected to be 85%. Major items of equipment included: a bulk extractor, capacity 400 m^3/h aqueous, 10 m × 11 m, fibreglass reinforced plastic construction and 12 strip units, capacity 40 m^3/h of solvent, 2 m × 4 m, fibreglass reinforced plastic construction [120].

The SX process implemented by Denison consisted of mixing IX uranium–barren water with kerosene. A DEHPA-TBP solvent was then added and the mixture allowed to settle into two phases. Owing to the density differential, kerosene floated to the top of the mixture and water sank to the bottom. By adjusting the acidity of the water, it was possible to have the yttrium pass from the water phase into the kerosene and vice versa. The solution was treated in three stages to concentrate yttrium and to remove undesirable contaminants such uranium and thorium. Ammonium difluoride was employed as the stripping agent to remove these contaminants. Kerosene was recirculated. Some uranium was returned to the uranium plant and wastewater was pumped to the tailings area. Yttrium, as Y_2O_3, was precipitated from the resulting water solution by raising the pH. It was then filtered using drum filters, dried and packed into drums. The final product graded about 40–50% yttrium.

The average feed rate of barren solution in 1988 was 8678 m^3/d. The plant operated for 336 d [121]. The yttrium plant was shut down on 15 June 1990, during its fourth year of operation and after having operated for 145 d in 1990 with an average feed rate of 8201 m^3/d of IX barren solution [122].

Back calculating from capacities and annual output suggest that the yttrium oxide feed grade to the plant was likely to be of the order of 0.0055% yttrium.

The plant was shut-down in 1990 wasat the request of the partners in response to prevailing market conditions, as well as to operational and environmental issues. Denison noted in its 1990 Annual Report that:

> *"The yttrium recovery plant was shut down in June 1990 pending installation of an after settler for solvent recovery. Soft markets for yttrium throughout the year precluded completion of this capital work and the resumption of operation of the plant."*

As a result of the subsequent closure of the Elliot Lake uranium mining operations, the plant would not be reopened, but was decommissioned along with the other Elliot Lake facilities.

4.9. PHOSPHATE DEPOSITS (U–SC–REE)

4.9.1. Aktau, Precaspian Region (Kazakhstan)

4.9.1.1. History

The Aktau area was explored for uranium in 1956 and the orebody discovered as an outcrop. An underground mine to prove the orebody was developed in 1959 and from 1964 two open pits were brought into production. Four other deposits were also discovered but these were considered to be too deep to be mined economically. Precaspian Gorno Metallurgy Combinat was the original operator, but was replaced by the Kaskor Joint Stock Company after independence.

Precaspian Combinat produced sulphuric acid and nitric acid for use in the mill. Power generation was provided by a separate entity, which supplied power from an oil-fired power station and a nuclear power station. The latter was almost totally dedicated to supplying power for the water desalination plant. Precaspian, in addition to uranium concentrates, also produced scandium, scandium–aluminium alloys, rare earth concentrates and phosphate for fertilizer.

Production levels were reduced in the late 1980s and early 1990s from what was believed to have been around 3 million t/year of ore to about 1 million t/year of ore. In January 1994, Kaskor announced the suspension of uranium production at Aktau.

4.9.1.2. Geology

The Aktau area is host to a series of phosphate deposits, one of which, the Melovoye deposit, has been extensively mined in recent years. Phosphorites occur as subhorizontal sedimentary layers of uraniferous clays which contain the phosphatized skeletons of fish. Ore zones are deposited in gentle troughs on the prehistoric ocean floor and are thin and stratified. Uranium, phosphorus and lanthanides are concentrated in the skeletal remains, but tend to be particularly rich within the crushed and fragmented finer particles. The phosphorus content of the bones themselves exceeds 30%, with the average grade of the deposit being between 5 and 12% phosphorus.

Dahlkamp [26] and the IAEA's classification scheme [1] classify the Aktau deposits as "Phosphate, organic phosphorite subtype, a unique sub-type of uranium deposits".

Reserves are extensive and are estimated to date to 25–20 Ma. The pit at No. 5 mine advanced about 300 m/year and the total advance since the start of mining in the 1960s is 7 km. It was estimated that 4 km of reserves remained in 1994 in No. 5 pit and mining at the pit could continue to 2015. The orebody dips at 1% and by the end of the lifetime of the orebody the depth of the ore would be 170 m. Total proven uranium reserves for the Aktau area in 1994 were reported to be 64 000 tU in a cost category of US $80–130/kgU. Reserves specific to the Melovoye deposit, which was being mined at Aktau, were reported to be 43 800 tU.

4.9.1.3. Mining

The Aktau operation consisted of two open pits: the No. 3 pit, which is actually two pits adjacent to each other, and the No. 5 pit. The orebody in both pits is similar, with the overburden removed by bucketwheel excavators, conventional excavators and trucks and the ore mined by excavator and truck. The stripping ratio in 1991 varied between 35:1 and 50:1. The working walls had a 12° slope.

Overburden was deposited in mined out areas on the downstream side of the pit as it advanced. It was estimated that there was a swell factor of 10% and owing to the extra volume, some overburden was deposited on the sides of the pit. Waste was moved about 2 km and waste stripping was about three months ahead of mining. Details of overburden and ore removal in No. 5 pit in 1991 are reported in Table 42.

TABLE 42. 1991 MINING SUMMARY: AKTAU No. 5 PIT

Material	Thickness (m)	Method	Quantity
Yellow sandstone and clay	17–20	Bucketwheel	6 million m^3
Blue clay	60–60	Shovel/truck	34 million m^3
Blue clay	30	Bucketwheel	12 million m^3
Ore	3	Shovel/truck	1 million t

The original plan in the 1960s envisaged mining by open pit methods down to a depth of 60 m and thereafter mining underground. The trucks used were standard Minsk produced 40 t trucks. There were two upper level bucketwheel excavators, twelve excavators feeding about 65 trucks and two bucketwheel excavators at the lower level. There was a total of 145 trucks at the pit. Trolley assist trucks were being considered for future operations.

The orebody in No. 5 pit was reported to be, at a depth of 110 m, up to 3 m thick and 3 km wide. In 1991, the ore thickness was reported as 0.8 m. Ore in No. 5 pit occurs in a distinct lens between clay layers and, on a daily basis, geologists demarcated the ore and selective mining was carried out. At the No. 3 pit, ore was interspersed between four clay layers. Consequently, careful grade control was required. The ore grade was said to average between 200 and 220 gU/t of ore, with the highest grades found being 600–700 gU/t of ore.

Water was removed from the orebody by advancing wells which dewatered the orebody prior to overburden stripping. These wells were ahead of the advance of overburden stripping by about 1000 m and the highly saline water was used for dust control on roadways.

Ore was loaded into railcars. There were thirty to thirty-five 60 t cars per train and about three journeys per day were made from No. 5 pit.

The No. 3 pit had eight smaller excavators working on waste removal and one excavator concentrating on ore excavation. About 42 trucks were operating and around 24 million m^3 of waste were removed annually. The orebody was 100 m deep and 600 m wide. Around 5 km of ore had been removed, with about 4 km of ore remaining in one direction of the orebody and 17 km of ore in the other direction. It was estimated in 1994 that the orebody could have a lifetime lasting up to 2015. The orebody was horizontal but, owing to the topography of the overlying land, the ore zone changed from surface outcrops to being buried 100 m deep.

Ore was trucked to railcars, with the same train configuration as that used for No. 5 pit. Two train journeys were made daily.

In 1991, about 2800 people worked in the mining section, 1000 people in No. 5 pit, 600 in No. 3 pit and the remainder in support services.

4.9.1.4. Milling

Mined ore, containing about 300 gU/t and 650 g/t of phosphorus, was received at the ore reception facility in side-tipping railcars. These pneumatically operated cars discharged ore into receiving bins. Ore was then sent to the milling section by conveyor.

The first stage of comminution was an autogenous milling stage in which ore was broken down to a particle size of -5 mm. Further size reduction occurred in a rod mill in closed circuit with a spiral classifier. The milling section had six milling units, each with a nominal capacity of 100 t/h. In 1991, only two of the units were operating as a result of a then recent cut-back in production.

Spiral classifier overflow was cycloned to concentrate the valuable minerals. This process produced a sixfold increase in concentration to the underflow. The overflow was routed to the 75 km^2 tailings dam and the underflow sent to the leaching section. The cascade leach section used nitric acid and sulphuric acid as lixiviants.

Slurry was filtered on rotating horizontal 24 pan vacuum filter units. There were 25 of these units installed in the filtration section. The filtering process included a washing cycle, which operated about half way through the process. Solids were routed to the tailings dam, with the cyclones overflow material and the filtrate precipitated with ammonia gas.

The ammonia precipitation and nitric acid re-dissolution steps at Aktau improved the quality of the solution prior to SX. The ammonia and nitric acid used in the process were manufactured by the Combinat using conventional technology. Ammonia, 450 000 t/year, was manufactured from natural gas and from atmospheric nitrogen, and nitric acid, 600 000 t/year, was manufactured from ammonia. Most of the sulphuric acid, 1 million t/year, was conventionally produced by roasting pyrite brought in by rail from a mine in the Urals and some acid was produced by the electrochemical method. The latter was described as a complex and expensive process. Pregnant solution was sent to the extraction section where a series of seven extractants in various suites was used for uranium, scandium and lanthanide recovery.

The uranium extraction section comprised 15 chambers of 5 m^3 mixer–settler units and treated the 4 gU/L feed liquor. Uranium was extracted first and the liquor routed to the scandium extraction section. Three units, each with eighteen 0.5 m^3 chambers, were used for scandium extraction. The liquor then gravitated to the lanthanide extraction section. Concentrates from this section were sent to Ukraine by railway for further processing. Liquor from the lanthanide recovery section was routed to the phosphate and fertilizer plant.

It was planned that future lanthanide concentrate processing would be carried out at Aktau in two stages. The first stage separated lanthanides from radioactive impurities such as thorium and actinium in 56 chambers, each of 1.2 m^3. The second stage then separated the lanthanides into heavy and light fractions, with samarium as the cut-off between the two groups. This section comprised 85 stages of extraction. In 1991, rare earth production was described as being more important than uranium production at Aktau. With a throughput of 3 million t/year of ore, that is, with six milling units running, it would be possible to produce 500 t/year of lanthanide concentrate, roughly split 60:40 (light/heavy). The future focus at Aktau was to have been on rare earths and scandium at the expense of uranium production. However, the scandium market was very small and it appeared to be difficult to market large quantities of scandium.

Uranium from the uranium extraction section was extracted using di(2-ethylhexyl) phosphoric acid and tributyl phosphate and was precipitated with ammonium carbonate as ammonium uranyl tri-carbonate. The product was filtered and the crystals recovered contained 84% uranium. Calcination of the precipitate was carried out at about 850°C. The calcination produced ammonia and carbon dioxide, which were absorbed and recycled.

Scandium from the scandium extraction section was re-extracted, filtered, re-dissolved and extracted in four repeated cycles to give a product comprising 99% scandium. The product was either converted to a scandium–aluminium master alloy, refined scandium or other scandium compound. The phosphate/fertilizer plant produced two products: a 30% phosphorus, 30% nitrogen fertilizer and a 52% phosphorus, 12% nitrogen fertilizer. Phosphate was precipitated from the SX spent liquor using ammonia.

4.9.1.5. Production data

Little information is available on actual historical uranium production except for the years 1990–1993 when production was reported to be 1000, 800, 350 and 320 tU, respectively. Dahlkamp [26] indicates a total production of about 15 000 t U and remaining resources of 64 400 t U in the US $80–130/kgU reasonably assured resource category.

4.10. SHALE DEPOSITS

Several massive black shale/schist formations containing large to very large resources of various metals at very low grades, including uranium, exist within Scandinavia. Notable exploration and development projects based on these formations include Talvivaara in Finland and Häggån and Viken MMS in Sweden.

Large resources of the order of 5.67 million t U have been estimated in the black shales from Estonia [123]. These rocks contain high concentrations of Mo, V, Zn, Pb, Ni and other metals. During the Soviet era, the black shales were mined for uranium at Sillamäe in north-east Estonia [124] and a total of 22.5 tU was produced from 272 000 t of ore from an underground mine between 1948 and 1952.

In Poland, about 560 000 t of copper is extracted annually from the black shales of the Lubin Sieroszowice Basin. Estimated uranium resources hosted in the shales are 144 000 tU at an average grade of 60 ppm U. It is estimated that about 1700 tU are deposited in the mine waste dumps every year. Studies are ongoing to recover uranium and other metals from copper flotation tailings and waste material from other mines to decrease the release of hazardous elements to the environment and to recover the valuable components [125, 126].

4.10.1. Talvivaara project (Finland)

4.10.1.1. Introduction

The Talvivaara project includes two adjacent deposits, Kuusilampi and Kolmisoppi, located in central Finland near the town of Sotkamo. These were discovered in the late 1970s by the Geological Survey of Finland. The host formation is metamorphosed black shale within the Early Proterozoic Kainu schist belt. Formation thickness ranges up to 330 m. Mineralization consists mainly of sulphides, including pyrite, pyrrhotite, chalcopyrite, sphalerite, alabandite and pentlandite [127]. Mineral resources have been reported to be in excess of 1000 million t of ore averaging 0.22% nickel, 0.13% copper, 0.5% zinc and 0.02% cobalt [128].

Uranium mineralization occurs dispersed throughout the formation as very small uraninite crystals [129]. Uranium resources are estimated at 11 880 tU for Kolmisoppi and 15 130 tU for Kuusilampi at an average grade of 17 ppm U.

In 2011, Cameco has signed two agreements to buy uranium produced at the mine by the Talvivaara Mining Company Plc. Talvivaara was expecting production of uranium to be approximately 350 tU per year once the mine ramps up to full production. The production capacity for uranium extraction could be increased if the Ni-Zn production is increased. Cameco would pay US$60 million to construct a uranium extraction circuit at the mine. Talvivaara would then pay back the initial construction costs in the form of uranium concentrate. Once the initial costs are paid Cameco would then continue to purchase the uranium concentrate at a pricing formula based on market price on the day of delivery [130].

The relatively low metal content of the ore was seen to preclude a conventional mining/milling process and a bioheap leaching process was successfully tested in the mid-2000s.

4.10.1.2. Process development

Talvivaara's application of the bioheap leach technology (Fig. 63) has its origins at the Outokumpu Research Centre, where it has been developed using the Talvivaara ore since 1987. Talvivaara is continuing the development on a laboratory scale with financing from the National Technology Agency of Finland, Tekes.

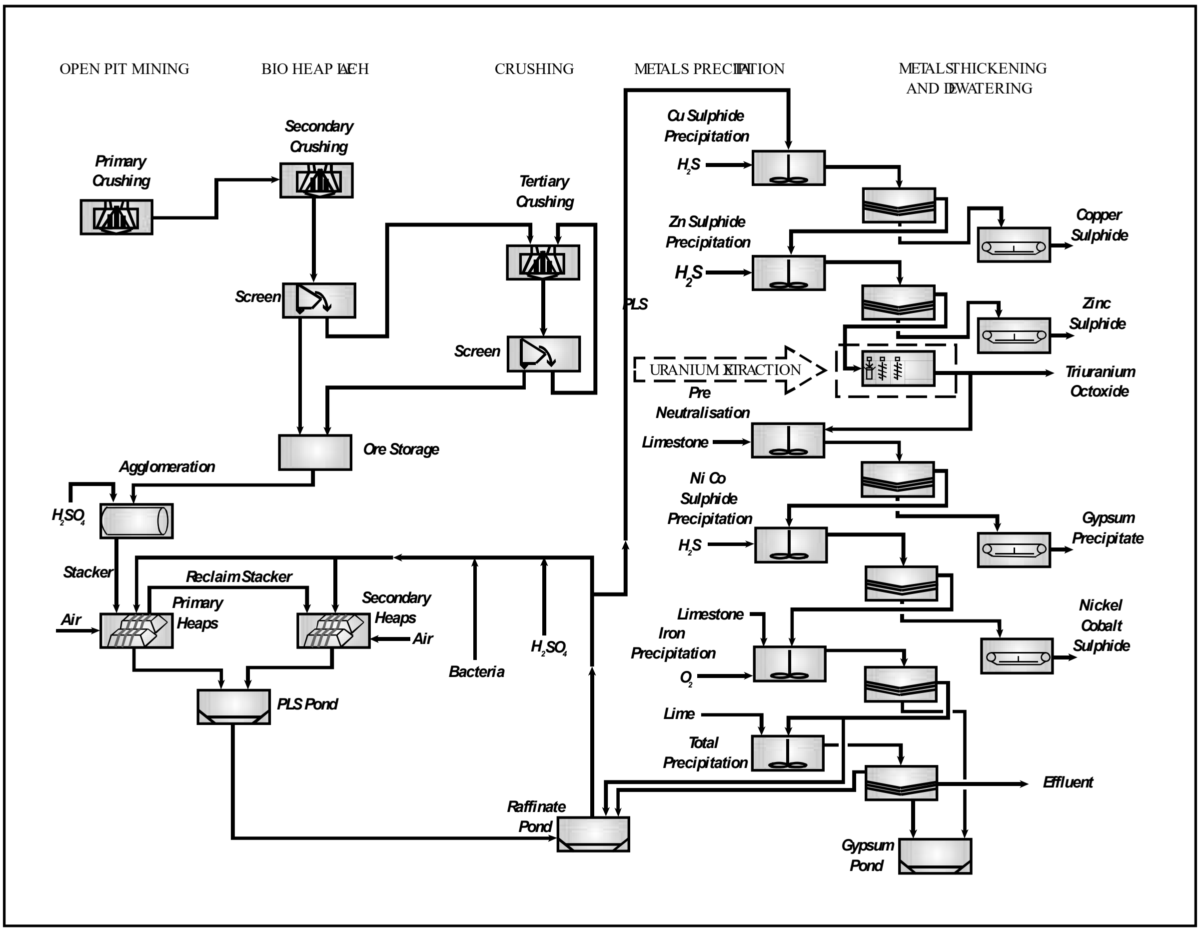

FIG. 63. Schematic of proposed black shale bioheap leach project at Talvivaara, Finland (adapted from [128]).

A pilot scale leaching trial with 110 t of ore has also been carried out as part of the European Union sponsored Bioshale Project, which studies the properties of black schist ores. This trial, run in a log column in Outokumpu, Finland, was successfully started in -20°C conditions in March 2005, thereby providing a strong indication of the applicability of the process under sub-arctic environmental conditions.

In mid-2005, a 17 000-t demonstration plant was constructed at the mine site. A representative ore sample was mined, crushed to 80% -8 mm, agglomerated and built to an 8-m high heap. Irrigation of the heap started in August 2005. The pilot heap was inoculated with indigenous bacteria collected from the site. The amount of bacteria in the pregnant leaching solution has been in the range of 106–108 cells/mL. The bacteria involved are mesophilic and thermophilic.

The startup of solution flow soon resulted in elevated temperatures of over 50°C in the pregnant leach solution. The temperature rise is due to the oxidation of the large quantity of pyrrhotite and pyrite in the ore. Elevated temperatures have also been maintained over boreal winter conditions. Metal recovery was initiated in November 2005. In winter 2007, the heap was reclaimed and the secondary leaching phase begun.

Combined trials with the bioheap leach technology at Talvivaara have provided an understanding of the key parameters of the process. One of the most important determinants of leaching rate is the particle size of the ore. Other key factors include pH, temperature and rate of aeration.
Open pit mining commenced in 2007 with an overall estimated stripping ratio of about 1.5 tonne of waste per tonne of ore [130]. The bioheap leaching process includes acidification with sulphuric acid

and the addition of air and bacteria. With a pregnant leach solution containing about 25 mg/L U, the recovery of was determined to be possible by using SX. A uranium recovery plant with a capacity of 350 tU/year was constructed but never commissioned owing to the low prevailing market price of uranium.

Talvivaara Mining Company plc, the initial operator, was a nickel mining business operating in Finland. Its mining business, Talvivaara Sotkamo, went bankrupt in November 2014, and was bound for liquidation. The mine had suffered several leaks of toxic metal tailings, which had threatened to contaminate local waterways.

There has been considerable Government involvement, as the Government of Finland is the largest single owner through its investment company Solidium. In 2015, the newly established State-owned entity Terrafame Oy bought the mining business of Talvivaara. Since February 2017, Terrafame operates the Talvivaara mine, which has new owners from Singapore Trafigura and Galena Asset Management.

In November 2017 Terrafame intended to submit a permit application to the Ministry of Employment and Economic Affairs for the production of uranium. If the government grants the permit, uranium recovery could begin towards the end of 2019 at a rate of 150-250 tU per year [131].

4.10.2. Black shale from Sweden

The so-called ‘alum shales’ of Sweden have been exploited for several centuries, initially for alum as a preservative for animal hides, then for oil and vanadium in the 1940s, and subsequently for small quantities of uranium in the 1960s. These shales are host to a very large, low grade, polymetallic resource of uranium, nickel, copper, vanadium, molybdenum and zinc which has the potential to be exploited by open pit mining and heap leach processing and recovery.

Of Middle Cambrian age, these shales attain a thickness of up to 180 m and exhibit low grade metamorphism. The unit represents a long period of very slow marine deposition and reflects an interval of very great stability during the early Palaeozoic history of the Baltoscandian Platform (Fig. 64). The formation is dominated by black shale with an organic carbon content ranging up to 20%. Small lenses of organic matter termed ‘kolm’ occur locally and these are enriched in uranium (up to 0.50 % U) [132].

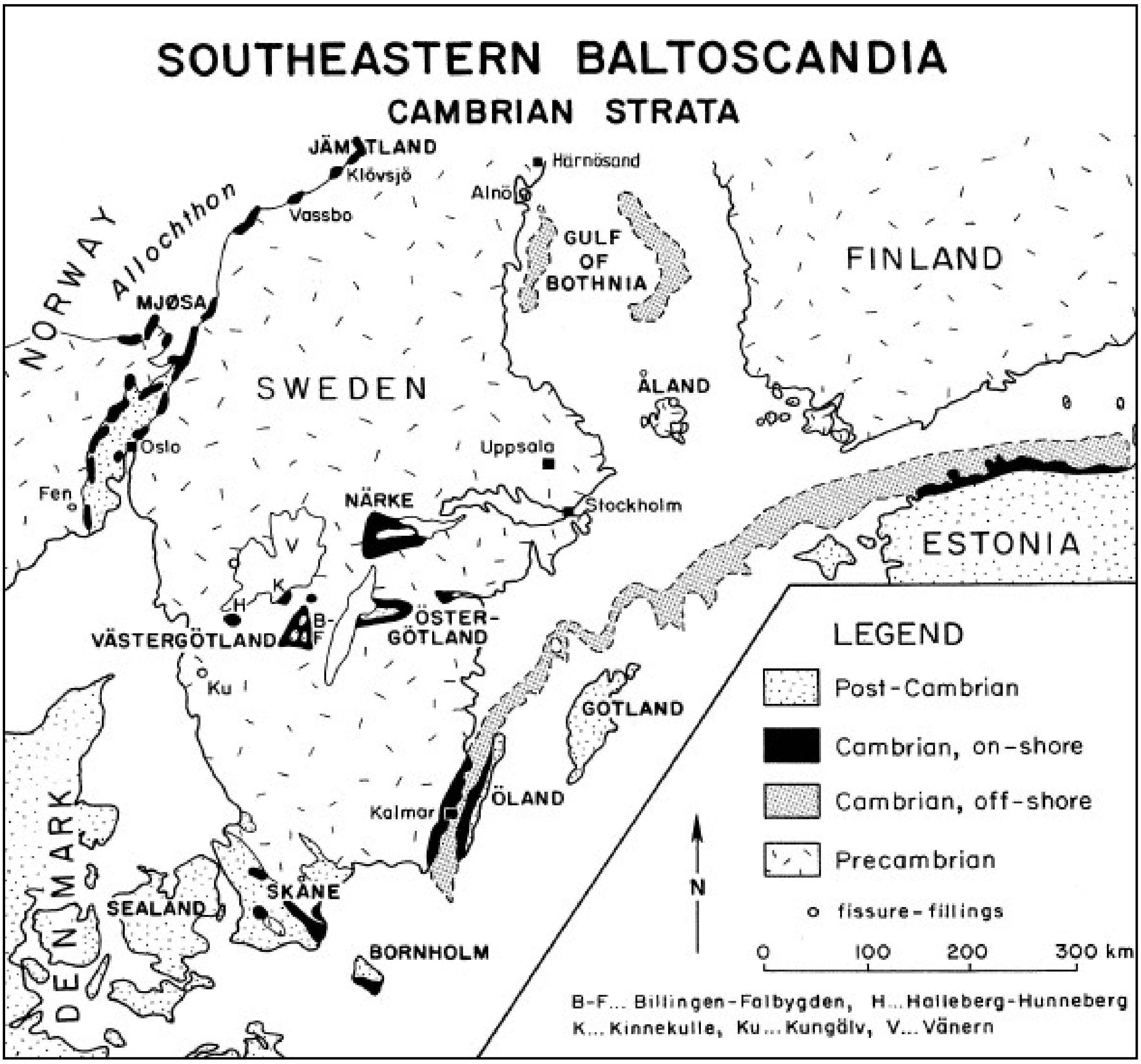

FIG. 64. Cambrian black shale deposits of southeastern Scandinavia [132].

4.10.2.1. Randstad historical uranium mining project (Sweden)

The Randstadt project is located within the Billingen-Falbygden outlier, Västergötland (Fig. 64). In 1905 a trial test to produce radium on an industrial scale from the alum shale was initiated by the AB Kolm company. The unprofitable project was abandoned in 1915 [133].

During the 1950s and 60s, Sweden had plans for the development of nuclear power implying the production of plutonium in heavy water reactors from natural uranium, but they were abandoned at the end of the 1960s. Then Sweden embarked on an ambitious project based on US light water reactors.

The Ranstad uranium mine began in 1959 and continued to 1983. Production started in 1965, but was never profitable. Mining continued up to 1969 on a reduced scale and a total of 182 tonnes of uranium was produced [133].

In 1973 uranium prices were increasing after the first oil crises and restart of uranium production in Ranstad was proposed to the Swedish Government in 1975 by the Swedish mining company LKAB. The proposal was withdrawn after massive protests of local organizations. A new proposal that included other minerals besides uranium was put together by LKAB in 1977, but was not accepted. That was the final stop to any plans for mining uranium from alum shale in Västergötland [133].

The concession for uranium mining in Ranstad lapsed in 1984 and it was the responsibility of the Swedish government for restoring the tailings deposits and the mining pits. The work took place from 1990 to 1993.

The total area of development of the full thickness of the Alum Shale Formation in the Billingen-Falbygden outlier is circa 490 km^2. The uranium resource in the uranium-rich unit alone is estimated at more than 1 million t at an average grade of 0.029% ppm and total resources of the outlier are estimated to be 1.7 million t at 0.021% U [133].

4.10.2.2. Viken MMS project (Sweden)

The proposed Viken MMS project in central Sweden sought to develop large scale metal production from these shales in the post-2000 era. In 2014, mineral resources for this project were estimated and these are shown in Table 43. Uranium resources stand at 445 755 tU at an average grade of 0.0147% U [134].

TABLE 43. VIKEN MMS ALUM SHALE PROJECT MINERAL RESOURCES

Classification	Ore (t)	U (%)	Ni (%)	Cu (%)	Zn (%)
Indicated	43 000 000	0.016	0.034	0.010	0.041
Inferred	3 019 000 000	0.014	0.034	0.012	0.042
Total/Average	3 062 000 000	0.014	0.034	0.012	0.042

The initial operating plan envisaged two adjacent open pit mines with depths of about 175 m and an overall waste/ore ratio of approximately 0.62:1.0.

Mineral extraction would be by means of bioheap leaching. Mineral recovery, as indicated by column leaching tests, could be expected to be of the order of 80% for uranium, 75% for nickel, 65% for copper and 80% for zinc.

This project, operated by Continental Precious Metals, is as yet undeveloped.

4.10.2.3. Häggån project (Sweden)

The nearby Häggån project is owned by Aura Energy Ltd and contains a resource of 309 155 tU at an average grade of 0.0133% with significant base metal by-products. A scoping study completed in May 2012 suggested that the project had excellent potential to become a major, low cost producer of uranium, with by-product nickel, zinc, molybdenum and vanadium [135].

Geology

The Häggån project is an alum shale deposit located in the Storsjön district of central Sweden. The wholly-owned Aura Energy project area covers 110 km^2. Aura Energy has completed a series of drilling programmes totalling 52 diamond drill holes during the period 2008–2010. This drilling has identified consistent mineralization at depths of between 20 m and 250 m across the mineralized area.

The Häggån project is hosted in Lower Palaeozoic shale, commonly referred to as Alum Shale, which extends throughout Scandinavia. The mineralized material is hosted in black shale which comprises up to 20% organic carbon.

Scoping study

Project highlights are summarized below:

- Expected annual production of 3540 tU (7.8 million lbs) would place Häggån in the top five current and planned uranium producers based on outcome of a scoping study;
- With 308 000 tU of inferred mineral resource at an average grade of 133 ppm U, Häggån is the 3rd largest undeveloped uranium mineral resource;
- Mean NPV of US $1.85 billion implies a valuation of less than US $6.00 per share based on results of the scoping study;
- Uranium and by-product metals are recoverable under a large variety of leach conditions (acid, alkaline and bacterial);
- High capital costs of conventional leach options and the bacterial tank option render these possibilities uneconomic at current uranium prices;
- The most promising option for metal recovery from the Häggån black shale deposit is heap bioleaching:
  - Bioleaching offers the advantage of reduced acid consumption due to the bacteria producing acid while oxidizing the sulphide species in the heap;
  - Work to date has demonstrated that bioleaching is a possible option for good metal recovery, although considerable more testwork is required.
- The economics of this project will require that both uranium and by-product metals be recovered;
- From an overall point of view, the Häggån deposit has long term potential for being a viable uranium producer, depending on long term commodity price trends and technical development.

Leaching testwork overview

Conventional acid leach

Good metal recoveries were obtained under both low and high acid leach conditions:

- 1 g/L H_2SO_4: 57–73% U extraction, 8–48 h.
- 10 g/L H_2SO_4: 82–89% U extraction, 8–48 h.
- 20–25 g/L H_2SO_4: 88–91% U extraction, 8–48 h.

Conventional alkaline leach

- Alkaline leach tests recorded poorer extraction levels, but the leach time was only 24 or 36 h. Commercial operations (such as Beaverlodge in Canada) used 96 h leach campaigns;
- Recovery from alkaline tests amounted to 73–78% U extraction in 24 h with a leach solution containing 0.3M Na_2CO_3;
- It is recommended that a 96-h leach period be instigated to ensure that this option is not viable.

Tank bioleaching

- Use of -2mm ore;
- 4 bioleach tanks operated;
- Results were good, as shown in Table 44;
- These results indicate good bacterial leaching, but acid consumptions were higher due to the calcite content of the ore.

TABLE 44. TEST RESULTS FOR THE HÄGGÅN PROJECT

	Condition	Uranium extraction (%)	Acid consumption (kg H_2SO_4/t ore)
Inoculated tank leach	10% solids	79	41
	15% solids	81	49
	20% solids	83	19
	Iron hydrolysis	84	-10.7
Uninoculated tank leach	15% solids	62	82
	Iron hydrolysis	58	57.5

Heap bioleaching:

- Column testwork with 4 kg of ore per column at -25mm size;
- Agglomeration with acid and $CaCl_2$ ($CaCl_2$ not recommended);
- Eight columns completed with six columns inoculated with bacteria at 3 pH ranges (pH 1.2, 1.6 and 2.0) and at two temperatures (35°C and 50°C) with two columns operated under abiotic conditions;
- Initially, fungi were a problem until the bacteria were introduced;
- Bacteria were preconditioned and then columns were inoculated;
- Growth media were added to columns;
- Recovery results in 110 d were very good and the larger ore particle size reduced the acid demand due to calcite.

5. SUMMARY

Hosted within numerous deposits around the world, uranium is often associated with other metals, and thus the potential for mutually enhanced economics can be significant. This publication reviews historical, current and potential production situations related to these various deposit types.

Uranium is naturally occurring, although at low levels, in all areas, and high tonnage operations for other metals, such as copper, offer the potential for uranium by-product production. Uranium that was produced along with other metals corresponds to about 11% of historical world uranium production, which in 2016 amounted to 2 802 230 tU. Gold, copper and uranium/vanadium deposits have provided the bulk of this production.

Since the 1950s, the palaeo quartz-pebble conglomerates of the Witwatersrand Basin in South Africa have been the primary source of uranium as a by-product of gold, with a total uranium production of 160 500 tU. Acid leaching of ore and tailings to recover uranium prior to treatment for gold recovery has enhanced subsequent gold recovery.

Uranium and copper are associated in many deposits in Australia, South Africa, South America, Zambia, the Democratic Republic of Congo and the USA.

In South Africa, the Palabora copper (Au–Ag–Pt–Pd–Ni) deposit associated with a large intrusion of carbonatite, produced over 4000 tU by means of a gravity circuit with a recovered grade averaging 6 ppm U.

The Bingham Canyon and Anamax porphyry copper deposits (Cu–Au–Ag–Mo) in the USA produced about 400 tU each through dump leaching and direct leaching. Bingham Canyon dump leach solutions averaged about 4 ppm U and contained small quantities of recoverable scandium.

Olympic Dam, an IOCG–U deposit in Australia, is today a major producer of uranium (3000–4000 tU annually). Uranium was first produced as a by-product, accounting for about 10% of revenue, but more recently, uranium, along with important quantities of precious metals, assumed an economic relationship more similar to co-products. Very large resources will ensure production for decades (Olympic Dam is the world’s largest resource of uranium with more than 2 million t at an average grade of 210 ppm U).

Uranium–vanadium deposits in the northern Colorado Plateau (USA) account for a significant portion of historical US uranium production. Most of these deposits were small, comprising a few tonnes to several tens of tonnes of uranium, but hundreds of such deposits were present in the form of tabular bodies and roll-front mineralization. Small scale mechanized underground mining methods were used for the bulk of production and acid leach extraction was utilized for metal recovery.

By-products of uranium production also include molybdenum (Brazil, Russian Federation and Niger), gold (Canada), yttrium (Canada), scandium (Kazakhstan and USA) and rare earths (Kazakhstan), which are present in various deposit types. Recovery of these minor by-products of uranium production enhanced the economics of uranium recovery.

Black shale formations contain the largest geological uranium resource, estimated in UDEPO to be 21.5 million t and hosted in 29 deposits/resources. Considering their number globally, and their extent, they represent vast potential resources. Uranium can be extracted as a by-product at Talvivaara, Finland, or as a co-product associated with U–Mo–V–Ni–Zn–Cd–Pb in several advanced projects (Häggån and MMS Vicken in Sweden).

In conclusion, uranium resources as co- and by-products of polymetallic, base, rare earth and precious metal ore deposits are enormous, and in conjunction with the concept of ‘comprehensive extraction’, will probably play an important role in the future in global uranium production.

6. Appendix

IAEA CLASSIFICATION POLYMETALLIC DEPOSITS WITH URANIUM

Almost all types and subtypes of uranium deposits listed in UDEPO contain anomalous to economic metallic elements besides uranium and therefore can be termed polymetallic. Strictly monometallic uranium deposits are rare. For some types and subtypes, uranium is the main component and is extracted on its own merit, and any accompanying elements are generally not taken into account owing to their low grade, cost of extraction, or abundance on the world market. Uranium is a co- and by-product in several deposit types and subtypes associated with numerous elements such as REE–Cu–Au–Ag–Mo–Zn–Pb–Sn–V–Ni–Co–Ta–Nb–Zr–Y–W–Th–Ti–Li–P–F–PGE, which are the most common.

Uranium deposit types and subtypes are briefly described below along with their accompanying elements. Typical deposit examples are listed with their current status.

Type 1. Intrusive deposits

Deposits included in this type are hosted in intrusive rocks of many different petrological compositions. Two main subtypes are recognized: ***1.1. Intrusive anatectic*** deposits associated with partial melting processes and contained in granite–pegmatite (e.g., Rössing and Husab, Namibia, and historical deposits in the Bancroft area, Canada), and ***1.2. Intrusive plutonic*** deposits related to magmatic differentiation processes, and subdivided into 3 classes; ***Quartz monzonite***, ***Peralkaline complex*** and ***Carbonatite***.

1.1. ***Intrusive anatectic:*** Most intrusive anatectic deposits are generally monometallic, with uranium being the dominant element with few associated elements, such as Th–Mo–Ti. The largest operating mine is Rössing (Namibia), where about 110 000 tU has been produced since 1976.

- Deposits in pegmatites from the Bancroft district (Ontario, Canada): **U–(Th–Mo)**. Depleted (1955–1983).
- Charlebois Lake pegmatoids (Nunavut, Canada): **U–(Mo)**. Dormant.
- Crocker Well pegmatites (South Australia): **U–(Ti–Th)**. Dormant.
- Namibia alaskites (Rössing, Rössing South, Anomaly A): **U**. Two mines are operating, Rössing and Rössing South (Husab project).

1.2. ***Intrusive plutonic:*** Uranium is present as a co-product or minor element in many plutonic intrusions around the world. Most deposits/resources are unconventional type resources.

Quartz monzonite: Correspond to very large intrusions (porphyry copper) (Fig. 65) producing Cu, Mo, Au and Ag. Uranium was extracted in the past at Bingham Canyon and Twin Buttes (USA) (see Sections 4.1.1. and 4.1.2.). A project was initiated between CODELCO and AREVA to produce uranium at Chuquicamata (Chile) but was never finalized. In the intrusions the uranium content is always very low, in the order of 2–20 ppm U, but geological resources are large and estimated at 10 000–75 000 tU for each intrusion.

- Bingham Canyon (USA): **Cu–Au–Ag–Mo**. Operating (Cu–Au–Ag–Mo).
- Twin Buttes (USA): **Cu–Mo–(Ag–Pb–As–Zn–Sb)**. Closed (operated 1965–1995);.Chuquicamata and Rodomiro Tomic (Chile): **Cu–(Mo)**. Operating (Cu–Mo).

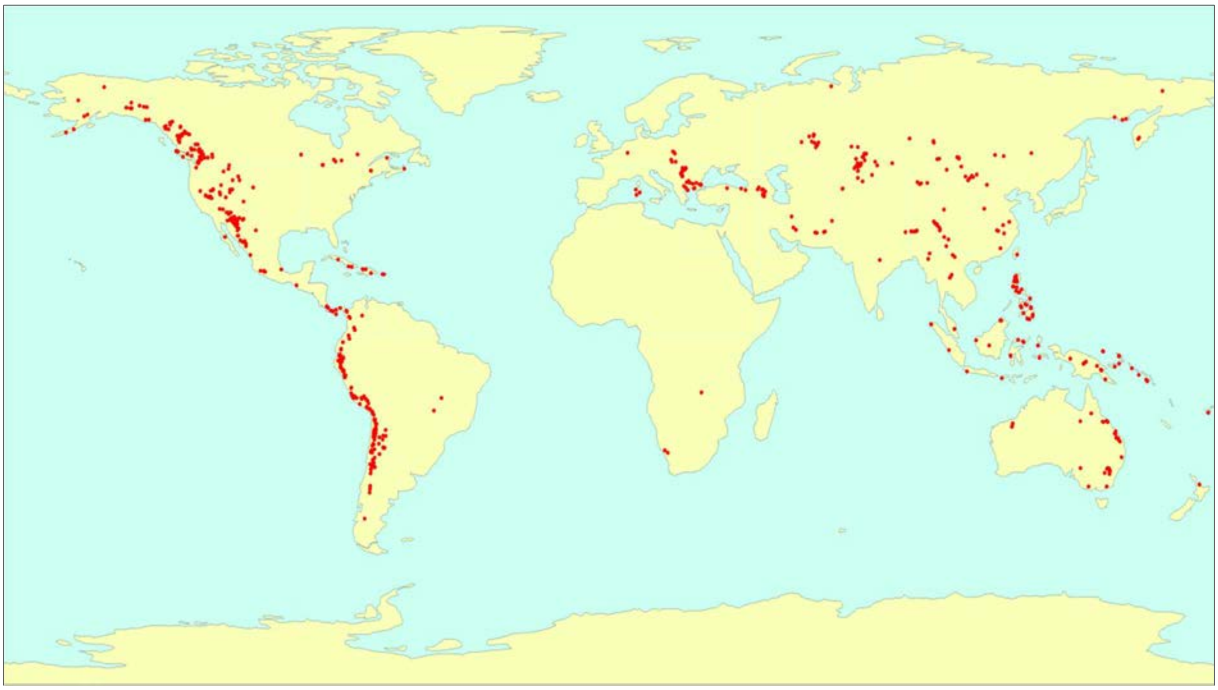

FIG. 65. World location of porphyry copper deposits, some of which are known to contain uranium, while others remain to be evaluated for uranium.

Peralkaline complex: Numerous peralkaline complexes are present around the world which contain anomalous to economic concentrations of Nb, Ta, REE, Zn, Zr, Y, Th and U (Fig. 66). The mineralogy is complex and most minerals are refractory in nature, thereby increasing extraction costs. Owing to the elevated grade of the ore, uranium was produced at Bokan Mountain (USA) and Poços de Caldas (Brazil) as the only element.

- Bokan Mountain (USA): **U–Th–(REE–Nb–Zr)**. Production of 720 tU (1957–1971) from the Ross Adams orebody. The nearby Dotson Zone of the complex is presently under evaluation for its REE–Y–Zr–Ta–Hf resources.
- Kvanefjeld (Greenland): **REE–Zn–U–(Y–F–Zr–Th)**. Under development to produce REE, U and Zn (see Section 4.1.4). One of the world's largest REE resource.
- Poços de Caldas (Brazil): **U–Mo–Zr–(REE–Th–Nb)**. Cumulative production of 1030 tU (1982–1995) (see Section 4.1.5). Closed.
- Ghurayyah (Saudi Arabia): **Ta–Nb–Zr–Y–(U–Th)**. Dormant.
- Twihinate (Morocco): **U–Th–REE–Nb–(Au–Mo–Zn–V)**. Dormant.

Carbonatite: As with peralkaline complexes, to which they are often associated, numerous carbonatite intrusions are present around the world (Fig. 67), which contain economic concentrations of Nb, P, REE, Ag, Zr, Y, Th, U and Ti. The mineralogy is also complex, with elements often trapped in refractory minerals. The Pilanesberg complex (South Africa), one of the largest of its type in the world, is thought to contain very large resources of Th and U of the order of several million tonnes of the both elements in addition to Zr–REE–Y–Nb–F–Zn.

- Palabora (South Africa): **Cu–Fe–P–(U–Au–Ag–Zr–PGE)**. The mine has produced some uranium as by-product (see Section 4.1.6). Operating for Cu–magnetite–vermiculite–(Zr–P–Co).
- Catalão, Araxa (Brazil): **Nb–P–(REE–U–Th–Ti)**. The Catalão I and II intrusions are mined for P and Nb, but U (average grade: 133 ppm U) is not extracted.
- Pilanesberg (South Africa): **Zr–REE–Y–Nb–Th–U–Sr–F–Zn**. Dormant.
- Glibat Lafhouda (Morocco): **Nb–Ta–REE–U–Th**. Dormant.

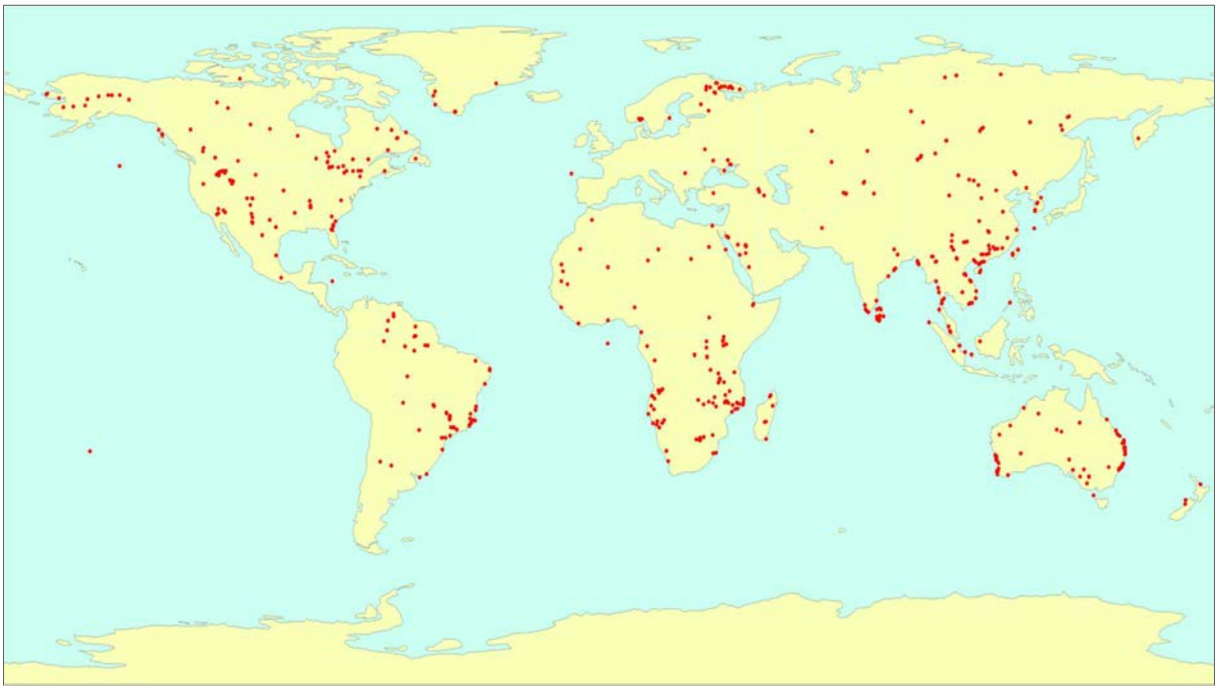

FIG. 66. World location of deposits containing REE (including peralkaline complexes and carbonatites), some of which are known to contain uranium, while others remain to be evaluated for uranium.

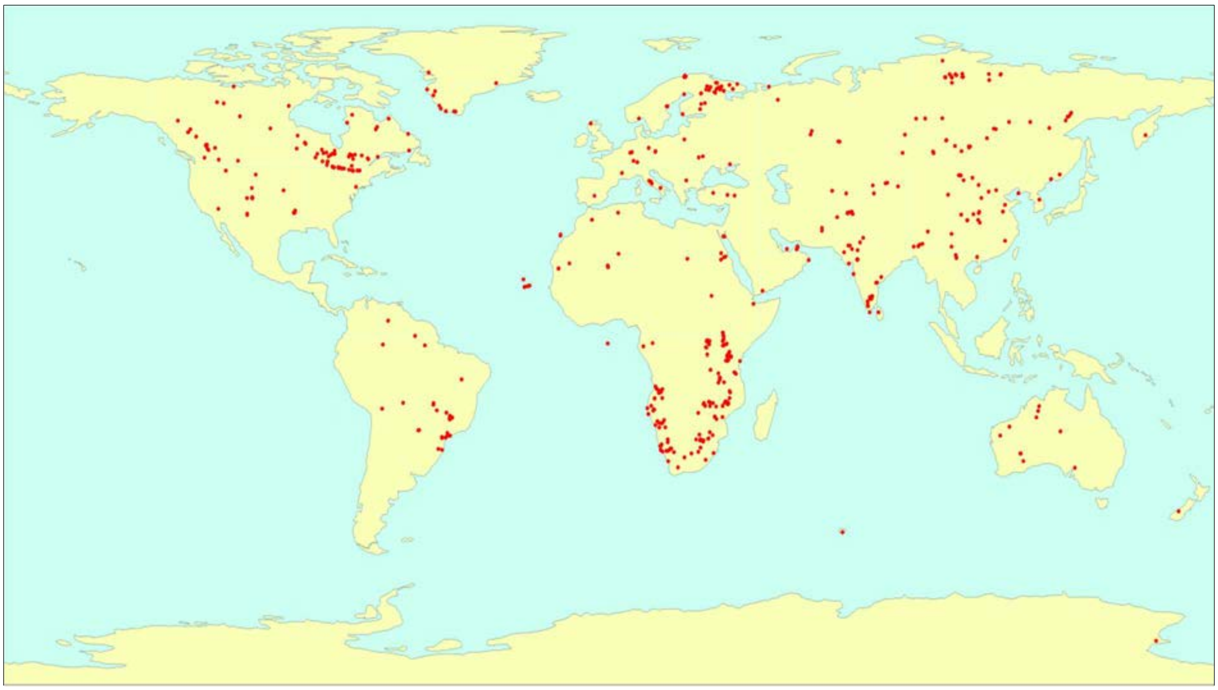

FIG. 67. World location of mineralized carbonatites, some of which are known to contain uranium, while others remain to be evaluated for uranium.

Type 2. Granite-related deposits

Deposits related to granite include veins composed of ore and gangue minerals in granite or adjacent rocks and disseminated mineralization occurring in granite as episyenite bodies. Two subtypes are distinguished based on their spatial setting with respect to the granitic pluton and country rocks; ***2.1. Endogranitic*** deposits and ***2.2. Perigranitic*** deposits.

2.1. Endogranitic: These deposits are considered monometallic even if they possess some anomalous concentrations of various elements which were never recovered during historical operation in such districts as:

- La Crouzille district deposits (France): **U**. Closed (1947–2001).
- Xiazhuang district deposits (China): **U-(F)**. Operating (U).
- Gornoye (Russian Federation): **U**. Development.

2.2. Perigranitic: Historically, deposits of the two large polymetallic districts (Pribram and Niederschlema–Alberoda) initially produced Ag–Co–Ni–Bi–Pb–Zn and subsequently, after World War II, uranium. The Salamanca Project deposits (Spain) currently being developed by Berkeley Energy are monometallic.

- Pribram Region, Jachymov district deposits (Czech Republic): **U–Ag–Co–Ni–Bi–Zn**. Depleted (1950–1990).
- Niederschlema–Alberoda district (Germany): **U–Ag–Co–Ni–Bi–As–Zn–Cu**. Depleted (1949–1991).
- Alameda, Gambeta and Retortillo, Salamanca district (Spain): **U**. Development.

Type 3. Polymetallic iron oxide breccia complex deposits

This type of deposit corresponds to a broad category of iron oxide–copper–gold deposits (Fig. 68). Olympic Dam and Carrapateena (Australia) are the two representatives of this type as both possess significant by-product uranium resources. Olympic Dam contains more than 2 million t of uranium at low grade (230 ppm U). Deposits of this group occur in haematite-rich granitic breccias (Olympic Dam) or in metasedimentary–metavolcanic breccias (Salobo, Brazil) and contain low grade disseminated uranium in association with copper, gold, silver and rare earth elements. They are also termed IOCG–U (iron oxide, copper, gold and uranium) deposits.

At Olympic Dam, uranium is produced as a by-product along with Cu–Au–Ag (see Section 4.2). IOCG–U deposits from Australia all contain anomalous uranium levels (50–250 ppm U). In Brazil, the Fe–Cu deposits located within metamorphic formations of the Carajas district contain anomalous uranium in the range 30–150 ppm U, which is not recovered.

- Olympic Dam (Australia): **Cu–Au–Ag–U–(REE–Y–Mo–Zn–Co–Pb–W–Bi)**. Operating (Cu–Au–Ag–U).
- Carrapateena (Australia): **Cu–Au–Ag–(U)**. Development (Cu–Au–Ag).
- Igarape Bahia, Sossego and Salobo, Carajas district (Brazil): **Cu-Au**. Operating (Cu–Au).

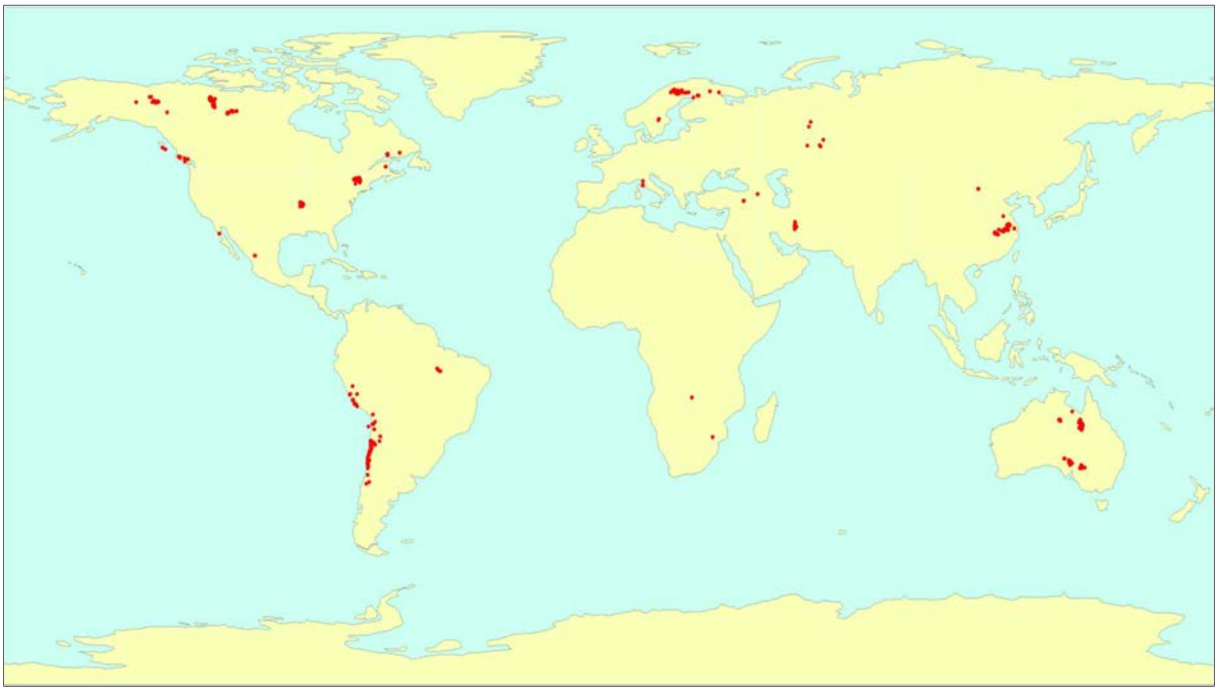

FIG. 68. World location of iron oxide breccia complexes (or Iron oxide copper gold - IOCG), some of which are known to contain uranium while others remain to be evaluated for uranium.

Type 4. Volcanic-related deposits

Volcanic-related uranium deposits are located within and near calderas filled with volcanic lavas or, more commonly, pyroclastic rocks and intercalated sediments. Uranium mineralization is present in volcaniclastic sediments (***Stratabound*** deposits, subtype 4.1) as disseminations and impregnations in permeable flows or preferentially controlled by structures veins and stockworks (***Structurally-controlled*** deposits, subtype 4.2). Uranium minerals are associated with molybdenum-bearing sulphides and pyrite. Associated gangue minerals consist of fluorite, carbonates, baryte and quartz. Uncommon ***Volcano-sedimentary*** deposits (subtype 4.3) consist of low grade carbonaceous sediments with an important tuffaceous component (Anderson mine (USA), El Boleo deposit (Mexico)).

The most common association is U–Mo–F, although some deposits also contain Zn and Li as recoverable elements. Molybdenum is recovered in deposits of the Streltsovska caldera (Russian Federation) and recovery of Li is planned in the deposits of the Macusani district (Peru).

4.1. Stratabound:

- Dornod No. 7, Mongolia: **U–(Mo)**. Dormant.
- Maureen (Australia): **U–(Mo–F)**. Depleted.
- Novazza (Italia): **U–(Zn)**. Dormant.
- Latium district: **U**. Dormant.

4.2. Structurally-controlled:

- Antei, Argunskoye and Stretlovskoye, Streltsovska Caldera (Russian Federation): **U–Mo–F**. Operating (U–Mo).
- Kuriskova and Novoveska Huta (Slovakia): **U–Mo**. Dormant.
- Xiangshan district (China): **U–(Th–Mo–P–REE–Pb–Zn)**. Operating (U).
- Macusani district (Peru): **U–Li**. Exploration.

4.3. Volcano-sedimentary:

- Anderson mine (USA): **U–V–(B–Cu–F–Li–Mo–Ni)**. Closed.
- Sierra Pintada district (Argentina): monometallic **U**. Closed.
- El Boleo project (Mexico): **Cu–Co–Zn–Mn–(U)**. Development.

Type 5. Metasomatite deposits

Metasomatite uranium deposits include those related to ***Na-metasomatism*** (subtype 5.1), which are the most common and are mined currently in Ukraine and Brazil, and those related to ***K-metasomatism*** (subtype 5.2), which are only known in the Elkon district (Russian Federation) and possess large resources that are currently being developed. Deposits associated with ***Skarns*** (subtype 5.3) are rare and small, but may contain REE and Th resources. Most Na-metasomatite deposits are monometallic. K-metasomatite deposits of the Elkon district (Russian Federation) are polymetallic; both Au and Ag will be extracted along with U as the main element. At Mary Kathleen (Australia) only U was produced.

5.1. Na-metasomatite:

- Kirovograd district (Ukraine): **U**. Operating (U).
- Lagoa Real (Brazil): **U**. Operating (U).
- Coles Hill (USA): **U**. Dormant.
- Valhalla (Australia): **U–(V–Cu)**. Dormant.

5.2. K-metasomatite:

- Elkon district (Russian Federation): **U–Au–Ag–(Mo–Cu)**. Development.

5.3. Skarn:

- Mary Kathleen (Australia): **U–REE**. Closed.
- Tranomaro (Madagascar): **U–Th**. Closed.

Type 6. Metamorphite deposits

Metamorphite deposits consist of disseminations, impregnations, veins and shear zones within metamorphic rocks of various ages. Three subtypes are recognized; ***Stratabound*** deposits (subtype 6.1), which are uncommon, ***Structurally-controlled*** deposits (subtype 6.2), which are well represented, and ***Marble-hosted phosphate*** deposits (subtype 6.3).

6.1. Stratabound: Stratabound deposits are generally monometallic. However, in the Chimiwungo and Malundwe copper mines (Zambia), these deposits are polymetallic and contain elevated uranium (0.05–0.08% U) which could be extracted.

- Forstau (Austria): **U**. Dormant.
- Nuottijarvi (Finland): **U-P**. Dormant.
- Chimiwungo and Malundwe (see Section 4.3.2), Lumwana project (Zambia): **Cu–(Co–Ni–V–U–Au)**. Operating (Cu).

6.2. Structurally-controlled: Most structurally-controlled metamorphite deposits are polymetallic, with U being the main component and often the only one extracted. In the Singhbhum Belt (India), U is recovered as a co-product of Cu within several mines.

- Schwartzwalder (USA): **U–(Mo–Ag–Cu–Pb)**. Closed (1953–2000).
- Ace–Fay–Verna, Beaverlodge district (Canada): **U–(Cu–Pb–Ni–Co–As–Au–Ag–Pt)**. Depleted (1953–1983).
- Rozna (Czech Republic): **U–(Ba–Zn–Pb)**. Operating (U).
- Shinkolobwe (Democratic Republic of Congo): **U–(Co–Ni–Mo–Se)**. Depleted (1921–1960).
- Port Radium (Canada): **U–Ag–(Cu–Bi–Ni–Co–Pb–Zn)**. Depleted (1938–1960).
- Lac Cinquante (Canada): **U–(Mo–Ag–Cu)**. Exploration.
- Kokshetau district (Kazakhstan): **U–(Mo–Sc–Y–REE–P)**. Closed (11 deposits mined, 1957–1998).
- Jaduguda, Bhatin and Turamdih, Shinghbhum Belt (India): **Cu–(U–Ni–Mo)**. Operating (Cu–U–Ni–Mo) (see Section 4.3.1).

6.3. Marble-hosted phosphate: such uncommon deposits are hydrothermal metasomatic uraniferous collophane deposits hosted by Precambrian metamorphic rocks.

- Itataia–Santa Quiteria district (Brazil): **P–U–(Tb)**. Operating (P).
- Zaozernoye (Kazhakstan): **U–P–(REE–Sc)**. Closed.

Type 7. Proterozoic unconformity deposits

Unconformity-related deposits are associated with an unconformable contact that separates Archaean–Palaeoproterozoic crystalline basement from overlying clastic sediments of Proterozoic age. Deposits consist of pods, veins and semi-massive replacements containing mainly pitchblende.

They are preferentially located in two major districts: the Athabasca Basin (Canada) and the Pine Creek Orogen (Australia). The Proterozoic unconformity deposits include three subtypes of variable importance; ***Unconformity-contact*** deposits (subtype 7.1), which all occur in the Athabasca Basin (Canada), ***Basement-hosted*** deposits (subtype 7.2), which occur in Australia and Canada, and ***Stratiform structurally-controlled*** deposits (subtype 7.3), which have only been found in India.

7.1. Unconformity-contact: Unconformity-contact deposits all occur in the Athabasca Basin (Canada). Deposits are situated at the base of the sedimentary cover, directly above the unconformity. They are characterized by their high to very high grades (1–20% U). They can be monometallic (McArthur River) or, for most, polymetallic (Cigar Lake, Key Lake, Dawn Lake, Shea Creek). Some Ni was produced at Key Lake. McArthur River and Cigar Lake, the richest deposits in the world, do not produce any by-products.

- Cigar Lake (Canada): **U–(Ni–Pb–Co–Cu–Mo–As–Au)**. Operating (U).
- Key Lake (Canada): **U–(Ni–Pb–Co–Au)**. Depleted.
- McArthur River (Canada): **U–(Ni–Pb–Co–Au)**. Operating (U).
- Midwest (Canada): **U–(Ni–Co–As)**. Dormant.
- Shea Creek deposits (Canada): U–(Ni–Co–Au). Exploration.

7.2. Basement-hosted: Basement-hosted deposits are stratabound and structure bound in metamorphic rocks below the unconformity. Disseminated and vein uraninite/pitchblende occupies fractures and occurs within the breccia matrix. Canadian deposits are generally monometallic whereas Australian deposits also contain Au and PGE. However, some gold was recovered from the Cluff Lake deposits (Canada) (see Section 4.5.1) and episodic Au-rich intersections are recorded in the Shea Creek deposits.

- Jabiluka (Australia): **U–(Au–Pt)**. Dormant.
- Ranger (Australia): **U–(Au–Pt)**. Operating (U).
- Coronation Hill (Australia): **U–Au–Pt–Pd**. Dormant.
- Eagle Point (Canada): **U**. Care and maintenance.
- Arrow (Canada): **U**. Exploration.
- Cluff Lake deposits (Canada): **U–(Au)**. Depleted.
- Andrew Lake, Thelon Basin (Canada): **U**. Dormant.

7.3. Stratiform structurally-controlled: Low grade stratabound deposits in India are monometallic.

- Lambapur, Chitrial (India): **U**. Dormant.

Type 8. Collapse breccia pipe deposits

Deposits in this group occur as cylindrical, vertical breccia pipes, and have only been recorded on the Colorado Plateau (USA). Pitchblende is associated with numerous sulphide and oxide minerals containing Cu, Fe, V, Zn, Pb, Ag, Mo, Ni, Co, As and Se. Resources are small–medium (300–2500 tU) with relatively high grades of around 0.20–0.80% U.

The Orphan Lode (1956–1969) was initially a Cu–Ag mine before any U was produced. The Canyon deposit is being developed to produce both U and Cu.

- Orphan Lode, Colorado Plateau (USA): **U–Cu–Ag–(V–Zn–Pb–Ni–Co–Mo–Se)**. Depleted (see Section 3.5.1).
- Canyon, Colorado Plateau (USA): **U–Cu–Ag–(V–Zn–Pb–Ni–Co–Mo–Se)**. Development (see Section 3.5.2).

Type 9. Sandstone deposits

Sandstone hosted uranium deposits occur in medium- to coarse-grained sandstones deposited in continental fluvial or marginal marine sedimentary environments. Volcanic ash may represent a major uranium source within the sandstone in some regions (Arlit district, Niger; Wyoming Province and Texas coastal plain, USA). Uranium is precipitated by reduction processes caused by the presence of a variety of extrinsic and intrinsic reducing agents.

Sandstone hosted uranium deposits can be divided into five main subtypes, with frequent transitional types between them.

9.1. ***Basal channel:*** Deposits consist of wide channels filled with permeable alluvial–fluvial sediments. The uranium is predominantly associated with detrital plant debris. Most deposits are monometallic.
- Dalmatovskoye (Russian Federation): **U**. Operating (U).
- Khiagda (Russian Federation): **U**. Operating (U).
- Beverley (Australia): **U**. Care and maintenance.
- Four Mile West (Australia): **U**. Operating (U).

9.2. ***Tabular:*** Tabular deposits consist of uranium matrix impregnations that form irregularly shaped lenticular masses within reduced sediments. The mineralized zones are parallel to the depositional trend. Deposits can be both monometallic and polymetallic; Mo is recovered in Niger, both U and V were co-products in the Salt Wash district, and Cu and Ag will be recovered along with U from the Proterozoic Falea deposit (Mali).
- Hamr (Czech Republic): **U–(Zr)**. Depleted (1972–1993).
- Salt Wash Uranium district deposits, Colorado Plateau (USA): **U–V–(Cu)**. Closed (1910–1988) (see Section 4.6.1).
- Arlit district deposits (Niger): **U–(Mo)**. Operating (U–Mo) (see Section 4.6.3).
- Imouraren (Niger): **U**. Dormant.
- Falea (Mali): **U–Ag–(Cu)**. Dormant.
- Kayelekera (Malawi): **U**. Care and maintenance.
- Ambrosia Lake district deposits (USA): **U–(V)**. Closed (1957–1990).

9.3. ***Roll-front:*** In roll-front deposits, the mineralized zones are convex in shape and oriented down the hydrological gradient. The mineralized zones are elongated and sinuous along strike and perpendicular to the direction of deposition and groundwater flow. Uranium is recovered through in-situ recovery techniques.

In the USA, deposits are generally monometallic. In Kazakhstan, both Re and Se can be concentrated to recoverable grades. In Uzbekistan, Se–Mo–Re–Sc were locally recovered as by-products while V–Y–REE are considered potential by-products at some deposits.
- Crow Butte and Smith Ranch, Wyoming Province (USA): **U**. Operating (U).
- South Texas deposits (USA): **U**. Nearly all closed (1960–2007). Alta Mesa and La Palangana operating.
- Budenovskoye, Tortkuduk, Moynkum, Inkai and Mynkuduk, Chu-Sarysu Basin (Kazakhstan): **U–(Re–Se)**. Operating (U).
- Uchkuduk, Zafarabad and Nurabad, Kyzylkum Region (Uzbekistan): **U–(Se–Sc–V–Mo–Re–La)**. Operating (U).

9.4. ***Tectonic-lithologic:*** Tectonic-lithologic deposits are discordant to strata. They occur in permeable fault zones and adjacent sandstone beds in reduced environments created by hydrocarbons and/or detrital organic matter. Uranium is precipitated in fracture or fault zones related to tectonic extension. They all are monometallic.
- Deposits of the Lodève Basin (France): **U**. Depleted (1975–1997).
- Deposits of the Franceville Basin (Gabon): **U**. Depleted (1961–1999).

9.5. ***Mafic dykes/sills in Proterozoic sandstone:*** Mineralization is associated with mafic dykes and sills that are concordant with or which cross-cut Proterozoic sandstone formations. Deposits may be subvertically oriented along the dyke's margins (Matoush, Otish Basin, Canada), or hosted within the dykes, or stratabound within the sandstones along lithological contacts with mafic sills (Westmoreland district, Australia). These deposits are characterized by the presence of Au.
- Deposits of the Westmoreland district (Australia): **U–(Au)**. Dormant.
- Matoush (Canada): **U–(Au)**. Dormant.

Type 10. Palaeo quartz-pebble conglomerate deposits

Detrital uranium oxide ores are found in quartz-pebble conglomerates deposited as basal units (Elliot Lake district, Canada) or intraformational conglomerates (Witwatersrand Basin, South Africa) in fluviolacustrine braided stream or lacustrine systems older than 2400–2300 Ma. The conglomerate matrix is pyritic and gold and other accessory and oxide and sulphide detrital minerals are often present in minor amounts. In the Witwatersrand Basin, uranium is mined as a by-product of gold (***Au-dominant*** deposits, subtype 10.1). Deposits in the Blind River/Elliot Lake area (***U-dominant*** deposits, subtype 10.2) also contain economic concentrations of REE. Minor amounts of Th and Y were produced at the Denison mine.

10.1. U-dominant:
- Deposits of the Elliot Lake district (Canada): **U–(REE–Th–Y)**. Depleted.
- Denison mine (Canada): **U–(REE–Th–Y)**. Dormant (1957–1992) (see Section 4.7.2).
- Echo Ridge mine (Canada): **U–(REE–Th–Y)**. Dormant. Project to recover REE with U.

10.2. Au-dominant:
- Vaal River and Ezulwini–Cooke, Witwatersrand Basin (South Africa): **Au–(U–REE–Th)**. Operating Au–(±U) (see Section 4.7.1).

Type 11. Surficial deposits

Surficial uranium deposits are young (Tertiary–Recent), near-surface uranium concentrations in sediments and soils. The largest of the surficial uranium deposits are found in calcretes. These calcrete-hosted deposits mainly occur in valley-fill sediments along drainage channels (***Fluvial valley***, subtype 11.2) and in ***Lacustrine–playa*** (subtype 11.3) sediments in areas of deeply weathered, uranium-rich granites. Carnotite is the main uraniferous mineral and therefore V is often associated with U. Surficial uranium deposits also occur less commonly in ***Peat bogs*** (subtype 11.1). Otherwise, these deposits are monometallic.

Placer deposits (subtype 11.5) are accumulations of heavy minerals formed by gravity separation during sedimentary processes. Types of placer deposit include alluvium, eluvium, beach placers and palaeoplacers. Valuable mineral components are monazite, rutile, zircon, ilmenite, leucoxene, xenotime, magnetite, chromite, wolframite and cassiterite. The principal minerals containing thorium and uranium are zircon, monazite and xenotime.

11.1. Peat bog:
- Kamushanovskoye (Kyrgyzstan): **U**. Dormant.
- Flodelle Creek (USA): **U**. Dormant.

11.2. Fluvial valley:
- Yeelirrie (Australia): **U–(V)**. Dormant.
- Langer Heinrich (Namibia): **U–(V)**. Operating (U).
- Trekkopje (Namibia): **U–(V)**. Care and maintenance.
- Marenica (Namibia): **U–(V)**. Dormant.

11.3. Lacustrine–playa:
- Lake Maitland and Lake Way (Australia): **U–(V)**. Development.
- Manyoni district deposits (Tanzania): **U–(V)**. Dormant.
- Aquelt Assfaya (Mauritania): **U–(V)**. Exploration.

11.4. Pedogenic and fracture fill:
- Beslet (Bulgaria): **U**. Dormant.

11.5. Placer deposits:
- Bangka Belitung (Indonesia): **REE–Ti–Zr–Th–U**. Exploration;
- Tash Bulak (Kyrgyzstan): **U–Th–Zr–P**. Dormant.

Type 12. Lignite–coal deposits

Elevated uranium contents occur in lignite or coal mixed with mineral detritus (silt, clay). Pyrite content is generally high. Two subtypes are recognized: ***Stratiform*** deposits (subtype 12.1) and ***Structurally-controlled*** deposits (subtype 12.2).

Stratiform lignite–coal deposits cover very extensive areas and may, in addition to uranium, contain anomalous concentrations of various elements such as Mo, V, Cu, P, Zn, As, Se and Ag. The uranium content is generally low to very low (1–20 ppm U). Concentration of these elements is increased up to tenfold in power plant ash and extraction of uranium from some of the extensive tailings deposits was planned in China. In some cases, such as in the Nizhne Ilyskoye deposit (Kazakhstan), the average grade can reach 0.1% U together with 0.03% Mo, 30–60 ppm Ag and 2–5 ppm Re. At the Mulga Rock project (Australia), it is planned to extract Cu–Co–Ni–Zn along with U from four deposits.

Uranium concentration levels may attain several per cent when lignite–coal seams are fractured, as for example in the lignite–coal deposits at Freital in the Dohlen Basin (Germany).

12.1. Stratiform:

- Springbok Flats (South Africa): **U–(Mo)**. Dormant.
- Nizhne Ilyskoye and Koldzhat (Kazakhstan): **U–Mo–(Ag–Cd–Se–Re–Co–Ni–Cu–Pb–Sc–Ga–Ge–V–Y–Zr–REE)**. Dormant.
- Daladi (China): **U–(Se–Mo)**. Dormant.
- Mulga Rock project (Australia): **U–(Cu–Zn–Ni–Co–Sc)**. Development.

12.2. Structurally-controlled:

- Freital, Dohlen Basin (Germany): **U–(Pb–Zn–Ni–Cu–Mo–V–As)**. Depleted (1947–1989).
- Turakavak (Kyrgyzstan): **U**. Depleted (1955–1960).
- Cave Hills and Slim Buttes (USA): **U–(Mo–As)**. Closed.

Type 13. Carbonate deposits

Deposits are hosted in carbonate rocks (limestone and dolostone). Mineralization can be syngenetic and stratabound or, more commonly, structure related within karsts, fractures, faults and folds. Three types of carbonate-hosted uranium deposit are recognized.

13.1. Stratabound:

- Tumalappalle (India): **U**. Operating (U).

13.2. Structurally-controlled:

- Mailuu-Suu (Kyrgyzstan): **U–(minor V–Mo–Pb–Ni–Co)**. Closed.
- Todilto District (USA): **U–(V–F)**. Depleted (1950–1981).

13.3. Karst:

- Bentou-Sanbaqi (China): **U–(Ni–Mo–Pb–Zn)**.
- Tyuya-Muyun (Kyrgyzstan): **U–V–Cu–Ba**. Dormant. Historical production of U–V–Cu (1908–?).

Type 14. Phosphate deposits

Phosphate deposits are principally represented by marine phosphorite of continental shelf origin containing syn-sedimentary, stratiform and disseminated uranium in fine-grained apatite. Phosphorite deposits constitute very large uranium resources (millions of tonnes), but at low grade (0.005–0.015% U). Uranium can be recovered as a by-product of phosphate production.

Three subtypes of uranium-bearing phosphate deposit are identified: ***Organic phosphorite*** deposits (subtype 14.1), ***Minerochemical phosphorite*** deposits (subtype 14.2), and ***Continental phosphate*** deposits (subtype 14.3), which are only known in the Bakouma District (Central African Republic).

14.1. Organic phosphorite:

Uranium was recovered as a co-product with P–REE–Sc from the Melovoye deposit (see Section 4.9.1).

- Melovoye, Precaspian Region (Kazakhstan): **P–U–(REE–Sc)**. Partially mined (1959–1993) (P–U–REE–Sc).
- Ergeninsky Region (Russian Federation): **P–U–(REE–Sc)**. Dormant.

14.2. Minerochemical phosphorite:

- Phosphoria Formation (USA): **P–(U)**. Operating (P). Small U production (1980–1987).
- Land Pebble District, Florida (USA): **P–(U)**. Operating (P). Some production of uranium (1978–2000).
- Gantour District (Morocco): **P–(U)**. Operating (P).

14.3. Continental phosphate:

- Bakouma District (Central African Republic): **U–P**. Dormant.

Type 15. Black shale deposits

Black shale uranium mineralization includes marine, organic-rich shale and coal-rich pyritic shale, containing synsedimentary, disseminated uranium adsorbed onto organic material and clay minerals. Of all deposits, black shales contain the largest geological resources of uranium, estimated in 2017 at 22 million t of uranium (Table 2). Anomalous trace elements (U, Mo, Ni, V, As, Sb) contents are typical for the carbonaceous shales. Two deposit subtypes are recongnized: ***Stratiform*** (subtype 15.1) and ***Structurally-controlled*** (subtype 15.2).

15.1. Stratiform: Uranium mineralization consists of synsedimentary disseminated uranium adsorbed onto organic and clay particles in marine, organic-rich, pyritic shale with thin coalified, phosphatic and/or silty intercalations. Discrete primary uranium minerals are absent.

Two advanced projects, Vicken MMS and Häggån (Sweden), are planning the co-production of Mo–Ni–V–Zn–U (see Sections 4.9.2 and 4.9.3).

- MMS Vicken and Häggån (Sweden)**: U–Mo–V–Ni–Zn–(Cd–Pb)**. Exploration and development.
- Chattanooga Shale Formation (USA): **U**. Dormant.
- Buckton Zone (Canada): **Mo–Ni–U–V–Zn–Co–Cu–Li–REEs–Y–Th–Sc**. Exploration.

15.2. Structurally-controlled: Carbonaceous shale related stockwork uranium deposits consist of strata-controlled, structure-bound uranium concentrated in stockworks within or immediately adjacent to carbonaceous, pyritic black shale/pelite beds with high organic carbon (up to 9% C) and sulphide (up to 3.5% S). The Ronneburg Orefield produced a total of about 113 000 tU from several deposits. It was the largest uranium producing area in the former German Democratic Republic and one of the largest producing areas in the world.

- Ronneburg district (Germany): **U–(Zn–Cu–Co–Ni)**. Depleted;
- Dzhantuar (Uzbekistan): **U–V–P–Mo–Y–Au**. Exploration.

Table 45 presents the number of deposits listed in UDEPO corresponding to the types with unconventional resources compared with the global number of deposits of the same type. UDEPO unconventional geological uranium resources are of the order of 51 million t hosted in 280 deposits. Table 45 also indicates that global geological uranium resources are probably much higher, considering the number of potential deposits/resources associated with these geological formations and which are not listed in UDEPO. Most of them contain anomalous values of various elements which may become economic to extract in the future.

As an example, Figs 69 and 70 illustrate the comments above with respect to 1635 global phosphate deposits listed in the USGS database [136] (Fig. 68) in comparison to the 73 deposits/resources listed in UDEPO.

TABLE 45. COMPARISON OF NUMBER OF DEPOSITS IN UDEPO DEPOSIT TYPES VERSUS NUMBER OF DEPOSITS OF THE SAME TYPE KNOWN AROUND THE WORLD

Deposit type/subtype	Number of deposits/resources in UDEPO	UDEPO resources (tU)	Grade range (%U)	Number of global deposits/resources[a]
Intrusive plutonic	40	1 936 123	0.0010–0.10	1660
IOCG–U	21	2 562 526	0.0045–0.035	50
Lignite–coal	75	7 406 507	0.0010–0.10	1600
Phosphate	73	14 326 175	0.0030–0.030	1635
Black shale	75	21 748 873	0.0010–0.10	≈ 1000
Surficial (placer)	13	66 948		≈ 1000
Total	297	48 047 152		≈ 7000

[a] Estimated from various databases.

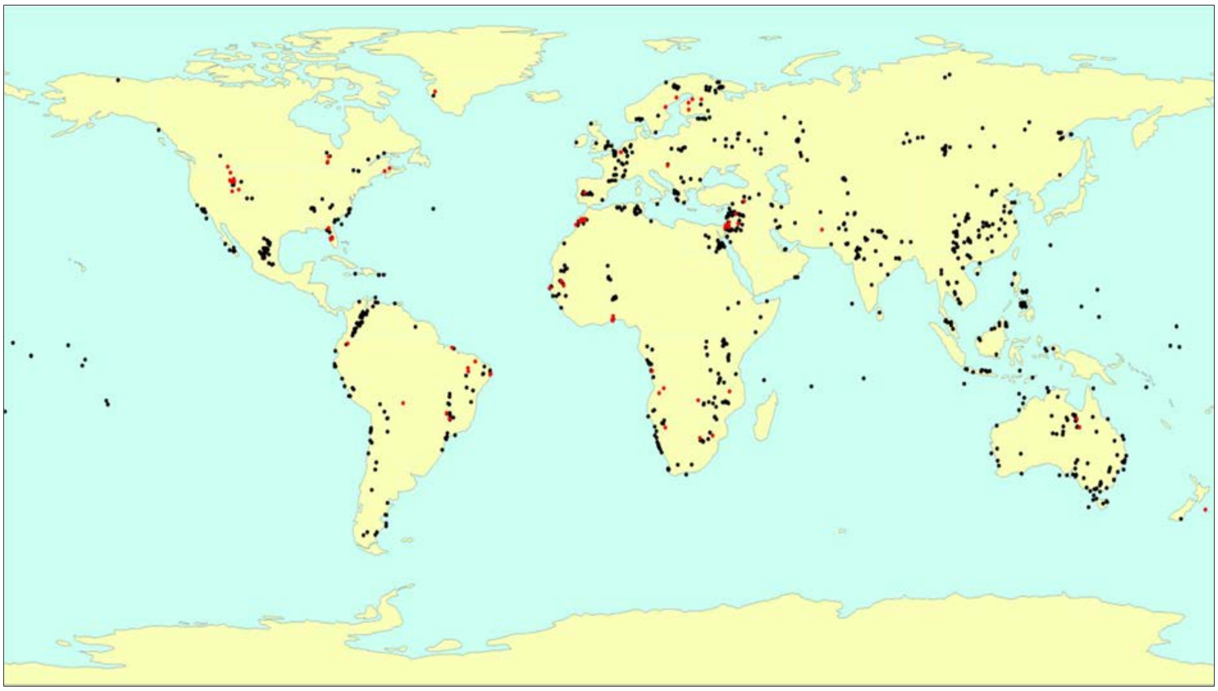

FIG. 68. Location of world phosphate deposits (black), some of which are known to contain uranium (red), while others remain to be evaluated for uranium.

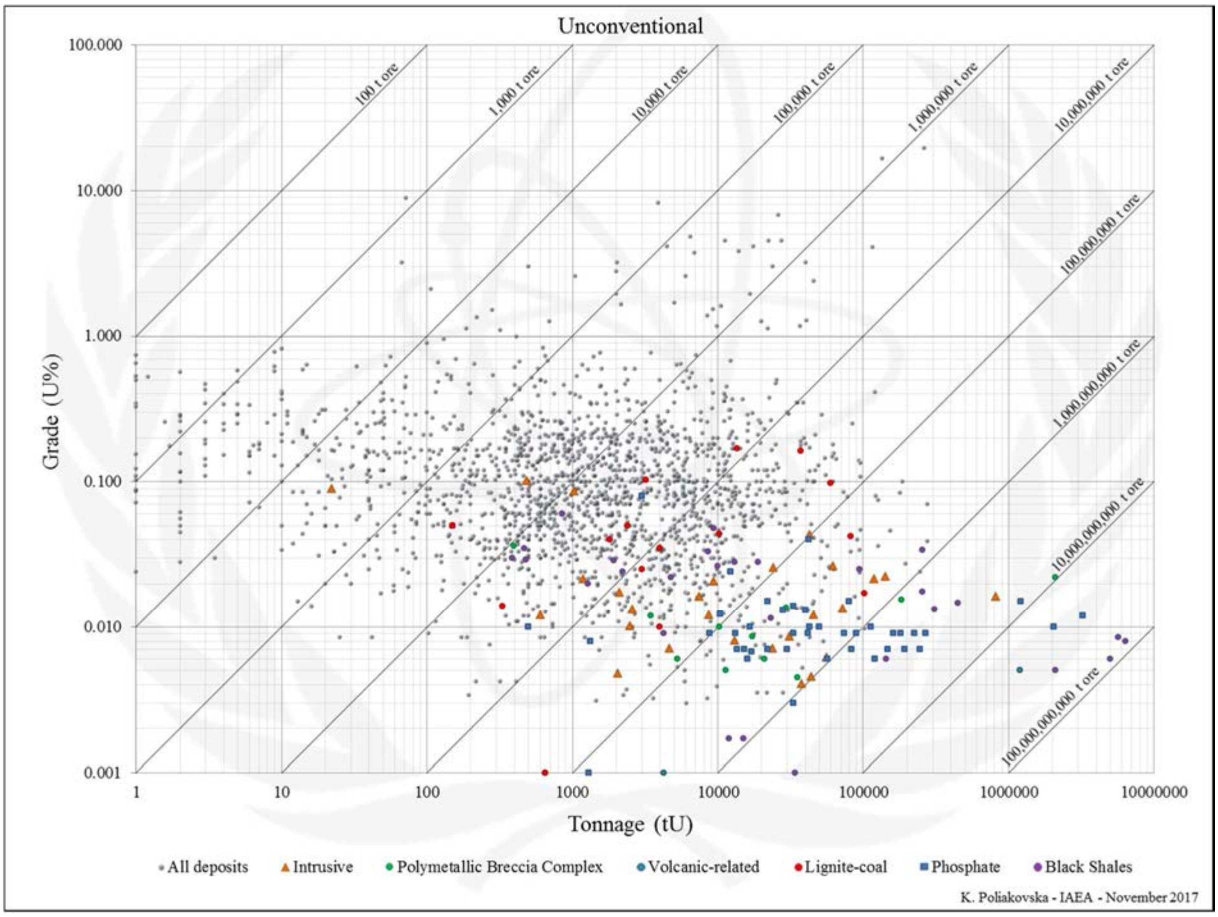

FIG. 69. Grade-tonnage relationship for unconventional resources. They are dominantly low grade, large tonnage deposits.

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