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Standing Committee on Environment and Energy
House of Representatives
Parliament of Australia

16 September 2019

RE: Inquiry into the Prerequisites for Nuclear Energy in Australia

To Whom It May Concern,

I note that recently the Committee was recently asked to:

“... specifically inquire into and report on the circumstances and prerequisites necessary for any future government’s consideration of nuclear energy generation including small modular reactor technologies in Australia ...”

Furthermore, specific aspects included **waste management, transport and storage** and **environmental impacts** and **any other relevant matter**.

The fuel source for nuclear power is uranium – of which Australia has an extensive amount – making uranium mining clearly a relevant matter with its own issues of waste management and environmental impacts. As a world-renowned scholar on the environmental and sustainability issues of mining (especially uranium), I hope that the inquiry properly addresses the issues of uranium mining as part of its work and reporting.

I would be most happy to be available for and present at a public hearing of the Committee’s work on this inquiry.

I am making this submission in my personal capacity as a citizen only.

If there are any questions, please do not hesitate to contact me.

Yours Sincerely,

[REDACTED]

Gavin Mudd

Uranium Mining and Resources in Australia: Brief Review of Key Issues

1. World Uranium Resources

Since the 1970s Australia has demonstrated a significant quantity of uranium resources, with key deposits being Olympic Dam, Ranger and Jabiluka (amongst others). Depending on whether you examine the OECD Nuclear Energy Agency's reporting of world uranium resources, Geoscience Australia's assessment of Australian uranium resources or my published research on global uranium resources and mining (see Mudd, 2014), Australia clearly has substantial resources of uranium. This is illustrated in the relationship between the average ore grade of a given resource (or deposit) and its contained uranium, shown for global uranium resources based on 2011 data in Figure 1.

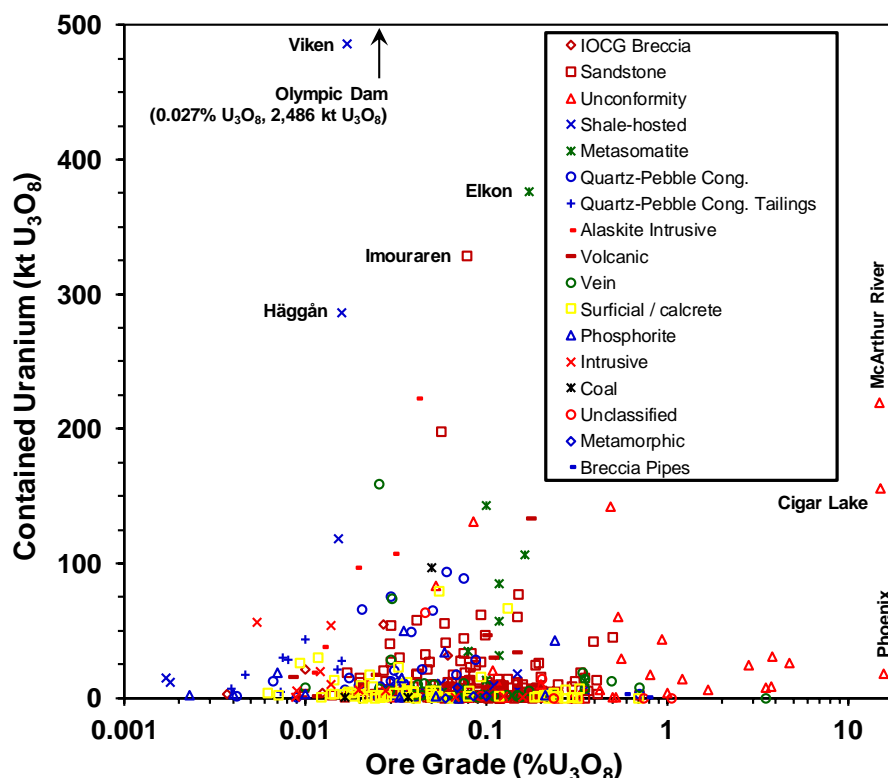


Figure 1: World uranium resources for ~2011 (Mudd, 2014)

According to the OECD's Nuclear Energy Agency, in its 2018 global review of uranium resources, mining and demand, Australia was stated as having about 2,422,610 tonnes of uranium oxide (as U₃O₈), just over double either Canada or Kazakhstan, and about a quarter of the world total of 9,418,560 t U₃O₈. For Australia, this fails to take into account the giant size of the full Olympic Dam mineral resource – which was reported to contain (in 2011) about 2,486,000 t U₃O₈ (Mudd, 2014).

However, just because Australia has extensive uranium resources does not mean we 'must' mine it. Australia has millions upon millions of tonnes of asbestos but we choose not to mine it due to the severe health risks associated with its mining and use – indeed Australia is still dealing with asbestos victims and cleaning up sites riddled with the material, demonstrating the lasting impacts from such reckless and wilful ignorance.

For Olympic Dam, owner BHP Billiton Ltd (now BHP Group Ltd) formally reported total mineral resources of 10,727 million tonnes (Mt) in 2018 (BHPB, 2018), containing copper (Cu), uranium (as U_3O_8), gold (Au) and silver (Ag), with the grades and contained quantities shown in Table 1. This demonstrates that Olympic Dam is, without doubt the largest uranium resource currently known in the world (compare to the deposits in Figure 1). However, there are also rare earths contained in the Olympic Dam deposit, which are not presently reported but the average grade is considered to be about 0.55% rare earth oxides (%REO) (value from Weng et al, 2015) – rare earths are crucial for renewable energy, military technology, electric vehicles, consumer electronics, catalysts, specialty alloys and many other uses. Olympic Dam is also known to contain important concentrations of tellurium (e.g. see Table 21.3, Schlesinger et al, 2011) – crucial for cadmium-tellurium based solar photovoltaic panels, as well as cobalt (see Mudd et al, 2013) – critical for energy storage batteries. Based on 2018 global market prices for Cu, U_3O_8 , Au and Ag from OCE (2019) and rare earth sales prices from Lynas (2019), the relative values of the various metals are also shown in Table 1. Rare earths clearly remain more valuable than the copper-uranium-gold-silver combined, with rare earths alone being over eight times the value of uranium alone – and rare earths certainly have a greater expected growth into the future as the world continues to expand renewable energy, consumer electronics and everything else which uses rare earths. Although BHP could extract greater value from Olympic Dam from rare earths, they continue to exclude this option from the processing of ore at Olympic Dam. As nuclear power declines, this could force Olympic Dam to stop uranium production and shift to rare earths instead – although this does not avoid radiation issues for workers and the environment and still leaves the tailings as radioactive as the original ore.

Table 1: Olympic Dam mineral resources and contained value

Resource	Copper	Uranium	Gold	Silver	Rare Earths
Mt	%Cu	%U ₃ O ₈	g/t Au	g/t Ag	%REO
10,727	0.72	0.023	0.30	1.3	0.55 ^A
Metal	Mt Cu	Mt U ₃ O ₈	t Au	t Ag	Mt REO
calculated	77.1	2.43	3,265	13,718	53.6
Price Reference	US\$/t Cu	US\$/t U3O8	US\$/t Au	US\$/t Ag	US\$/t REO
OCE (2019)	6,492	53,630	40,968,714	505,190	20,062 ^B
US\$millions	US\$500,362	US\$130,565	US\$133,770	US\$6,930	\$1,183,653
	Total: US\$771,627				

Note: ^AValue from Weng et al (2015). ^BAverage sales price for Lynas Corp rare earth sales (data from quarterly ASX reports).

2. World Uranium Mining

Australia has long been a modest supplier of uranium to the world, see Figure 1, based on a few key mines such as Ranger, Olympic Dam, Beverley-Four Mile and Nabarlek. Whilst major countries such as Canada have always maintained a strong global position, Kazakhstan has risen to dominance in the past decade, mainly through the use of in-situ leach (ISL) mining of uranium.

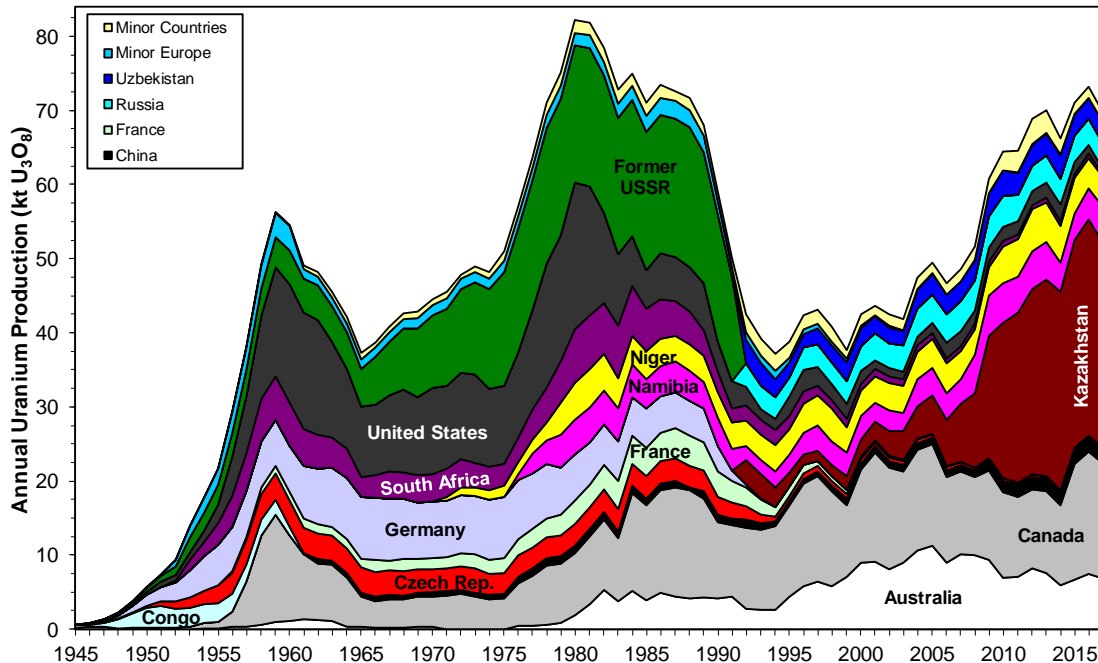


Figure 2: World uranium mine production (1945 to 2017) (data updated from Mudd, 2014)

For comparison, the global size and value of numerous metals and minerals including uranium is shown in Table 2 for the year 2018. This shows that the uranium sector remains modest (i.e. ~US\$3.4 billion) in comparison to major metals or minerals such as iron ore (the largest), copper, gold, phosphate ore and others (e.g. bauxite at ~US\$9.3 billion, copper at ~US\$138.9 billion or iron ore at ~US\$205 billion).

More specifically for Australia, our mineral exports are compiled and shown in Table 3, based on data from Australia's Office of the Chief Economist (OCE, 2019) (and limited data from other sources). This collectively also shows that uranium is a minor sector of Australia's mining industry – **and that lithium has now accelerated past uranium in value (i.e. \$1,602 vs \$624 million, respectively)**. This is a clear sign of the changing focus in energy technologies in the twenty-first century – that is, a greater focus on renewable energy systems and increasingly batteries for energy storage at home and grid / utility scales.

Finally, with respect to selected agricultural commodity exports, there are numerous commodities which significantly exceeded uranium in value in the financial year 2017/18, namely: wine (\$2,831 million), cheese (\$947 million), dairy products (\$3,422 million), barley (\$2,303 million), canola (\$1,532 million), chickpeas (\$1,047 million) or sugar (\$1,536 million), fruit (\$1,241 million) and nuts (\$803 million) (all data from ABARES, 2019).

Table 2: World metal and mineral production by price, volume and market size

Rank	Metal / Mineral	Units	Price (US\$/t)	Production	Value (US\$millions)
1	Iron Ore	Mt ore	82	2,500	205,000
2	Copper	Mt Cu	6,614	21	138,890
3	Gold	t Au	40,836,013	3,260	133,125
4	Alumina	Mt alumina	560	130	72,800
5	Zinc	Mt Zn	3,020	13	39,264
6	Nickel	Mt Ni	14,000	2.3	32,200
7	Potash	Mt potash	740	42	31,080
8	Phosphate Rock	Mt ore	68	270	18,360
9	Manganese Ore	Mt ore	960	18	17,280
10	Silver	kt Ag	491,961	27	13,283
11	Lead	Mt Pb	2,293	4.4	10,088
12	Chromite Ore	Mt Ore	280	36	10,080
13	Bauxite	Mt bauxite	31	300	9,300
14	Molybdenum	kt Mo	27,000	300	8,100
15	Palladium	t Pd	31,832,797	210	6,685
16	Tin	kt Sn	20,062	310	6,219
17	Platinum	t Pt	28,938,907	160	4,630
18	Tungsten	kt W	41,614	82	3,412
19	Rare Earths	kt REO	20,062	170	3,411
20	Uranium	t U3O8	53,630	63,087	3,383
21	Zircon Conc.	Mt conc.	1,500	1.5	2,250
22	Barite	Mt conc.	180	9.5	1,710
23	Fluorspar	Mt	270	5.8	1,566
24	Lithium	kt Li	17,000	85	1,445
25	Niobium	kt FeNb	21,000	68	1,428
26	Antimony	kt Sb	8,598	140	1,204
27	Ilmenite Conc.	Mt conc.	220	5.4	1,188
28	Boron	Mt B2O3	390	3.035	1,184
29	Rutile Conc.	Mt conc.	990	0.75	743
30	Tantalum	t Ta	218,000	1,800	392
31	Rhenium	t Re	1,500,000	49	74

Notes: All data from the USGS Mineral Commodity Summaries 2019 (USGS, 2019), except where noted.

Table 3: Australian metal and mineral exports by price, volume and market size

Rank	Metal / Mineral	Units	Price (US\$/t)	Production	Value (US\$millions)
1	Iron Ore	Mt ore	82.4	835.5	68,825.2
2	Gold (domestic)	t Au	54,528,907	341.0	18,595.1
3	Alumina	Mt alumina	579.7	17.87	10,357.3
4	Copper Conc.	kt cont'd Cu	6,359	943.0	5,997.2
5	Zinc	Mt Zn	3,233	1.29	4,155.0
6	Manganese Ore ^A	Mt ore	1,296 ^A	3.1 ^A	4,016.2 ^A
7	Nickel	kt Ni	21,029	165.6	3,482.7
8	Copper Refined	kt Cu	8,989	356.5	3,204.8
9	Lithium	kt spodumene	758.3	2,113	1,602.0
10	Lead (total)	kt Pb	2,977	528.2	1,572.6
11	Bauxite	Mt bauxite	41.5	31.51	1,308.4
12	Zircon Conc. ^B	kt zircon	1,316 ^B	516.0 ^B	679.0 ^B
13	Uranium	t U3O8	84,730	7,369	624.4
14	Diamonds	Mcarats	37.5	14.90	558.2
15	Rutile Conc. ^B	kt conc	679.9 ^B	734.2 ^B	499.2 ^B
16	Rare Earths	kt REO	20,062	18.1	363.0
17	Ilmenite Conc. ^B	kt conc	233.5 ^B	1,157 ^B	270.1 ^B
18	Silver	t Ag	678,900	286.4	194.4
19	Tin	t Sn	24,168	6,911	167.0
20	Antimony	t Sb	11,603 ^C	3,397 ^D	39.4
21	Platinum+Palladium	t Pt+Pd	44,058,848	0.54	23.8
22	Phosphate Rock	Mt ore	127.0	0	0

Notes: All data from the Office of the Chief Economist (OCE, 2019), except where noted. ^AData from USGS (2019). ^B2017 OCE export data, price data from USGS (2019). ^CPrice data from USGS (2019). ^DExport data assumes 100% of production from Costerfield Au-Sb mine operated by Mandalay Resources Ltd.

3. World Nuclear Power

There are various sources for statistics and information on world energy production and consumption, such as the International Energy Agency's (IEA) *World Energy Outlook*, petroleum company BP's *Statistical Review of World Energy* or the U.S. Energy Information Administration and their extensive publications and data. All of these groups, however, have vested interests in continuing to promote fossil fuels or nuclear power – especially at the expense of renewable energy technologies (although the IEA should be given credit for finally supporting renewables more positively in recent years). For nuclear power, the best and most clinical source of data and information is the *World Nuclear Industry Status Report* published about every two years by lead authors Mycle Schneider and Antony Froggatt (e.g. Schneider et al, 2018). Here we present the basic scenarios for future energy based on the most recent IEA World Energy Outlook 2018 (IEA, 2018) and the Schneider-Froggatt-led review of nuclear power.

The IEA publishes their *World Energy Outlook* ('WEO') annually, providing a global picture of key energy production and consumption trends, policy developments and challenges as well as presenting various scenarios for the future of energy – especially now in the context of climate change and the need to move away from fossil fuels (coal, oil, gas). The 2018 WEO (IEA, 2018) presented three scenarios – namely the Current Policies Scenario (CPS), New Policies Scenario (NPS) and the Sustainable Development Scenario (SDS). In brief, the CPS assumes nothing will change from current patterns with only existing policies continuing, the NPS assumes global policy action to address climate change risks in line with the Paris Agreement on Climate Change, while the SDS goes further to examine the policy and societal transformation required to achieve the United Nations Sustainable Development Goals (SDGs). Assuming the world aims for at least the NPS or perhaps even the SDS, the IEA's predictions of energy production by source under each scenario is shown in Figure 3 – highlighting the minor role of nuclear power globally in either scenario. **In the SDS, renewable energy clearly overtakes nuclear power by 2040.**

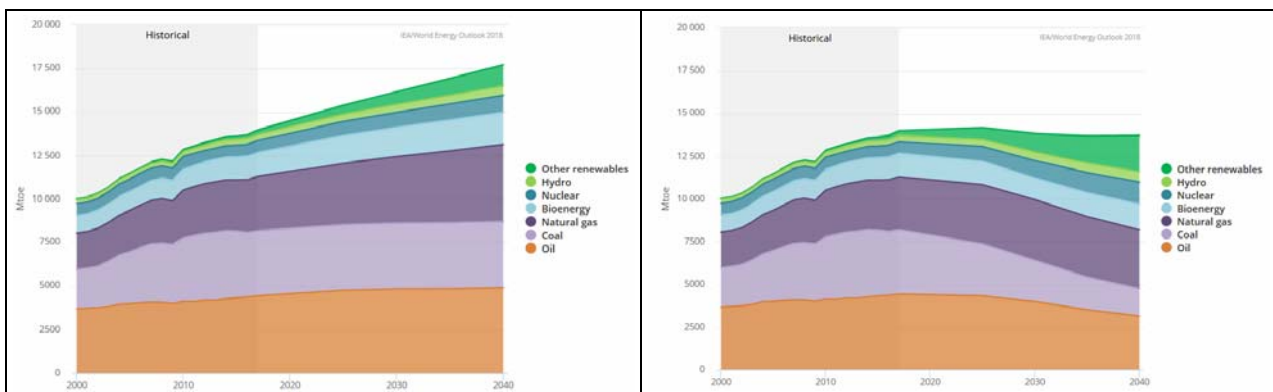


Figure 3: Principal IEA scenarios for the future of world energy supply based on the New Policy Scenario (NPS; left) and the Sustainable Development Scenario (SDS; right) (drawn from online data tool associated with IEA, 2018)

Furthermore, in WEO 2018, the IEA presented the respective greenhouse gas emissions for the three scenarios, shown in Figure 4. This shows that only the SDS is capable of reaching the climate change targets agreed globally in the Paris Agreement.

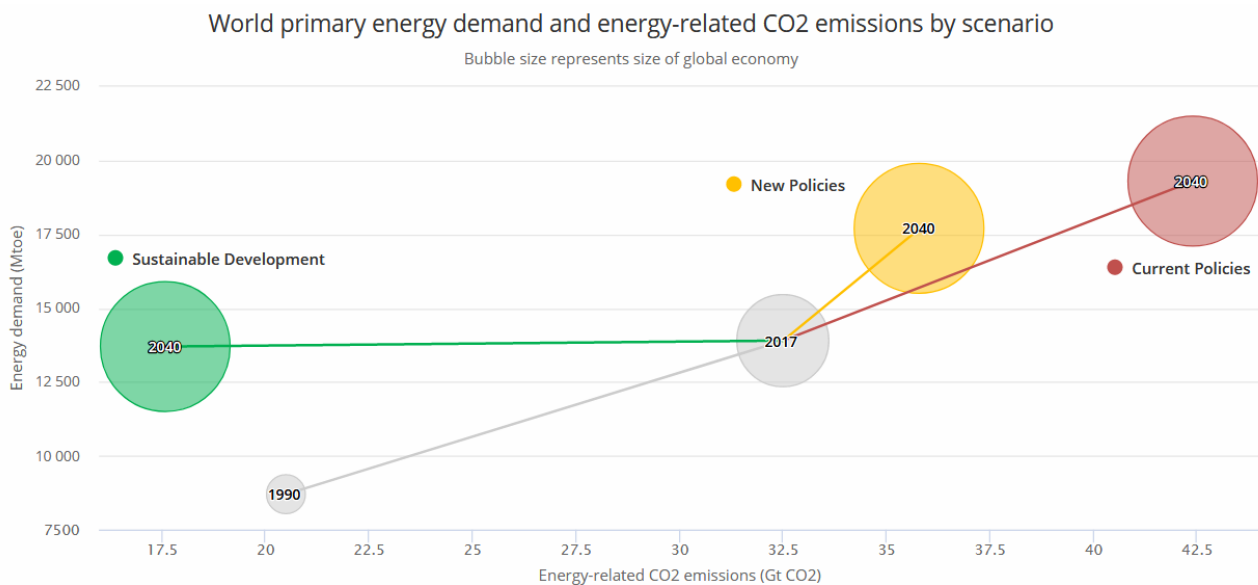


Figure 4: Principal IEA scenarios for the future of world energy demand and associated greenhouse gas emissions for the Current Policies (CPS), New Policies (NPS) and the Sustainable Development Scenarios (SDS) (IEA, 2018)

The *World Nuclear Industry Status Report 2018* (Schneider et al, 2018) presents extensive data and graphs on the historical and current global status of nuclear power – and is widely considered to be the most authoritative and truly independent analysis of nuclear power. The first critical graph is the net generation of electricity by nuclear power globally and its share of electricity overall, shown below in Figure 5. The second crucial graph is the opening and closure of nuclear power reactors over time, shown in Figure 6.

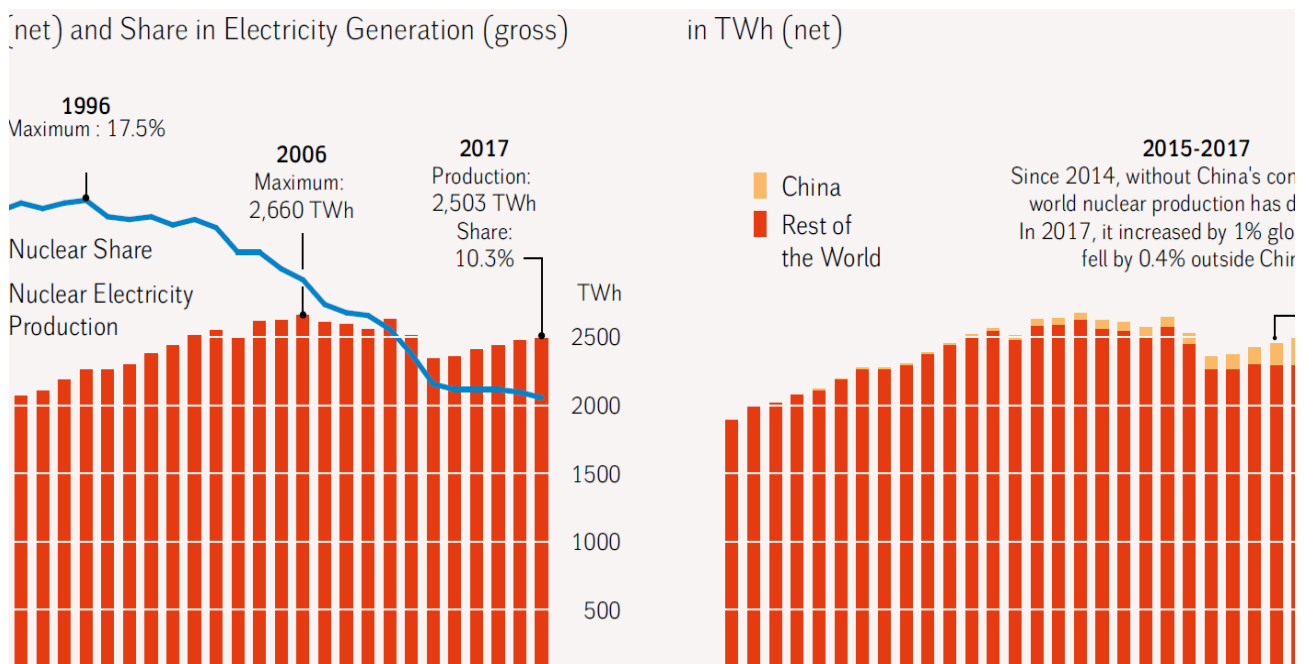


Figure 5: World net electricity generation and share from nuclear power (left), including the role of China's growing nuclear power sector in the past decade (right) (Schneider et al, 2018).

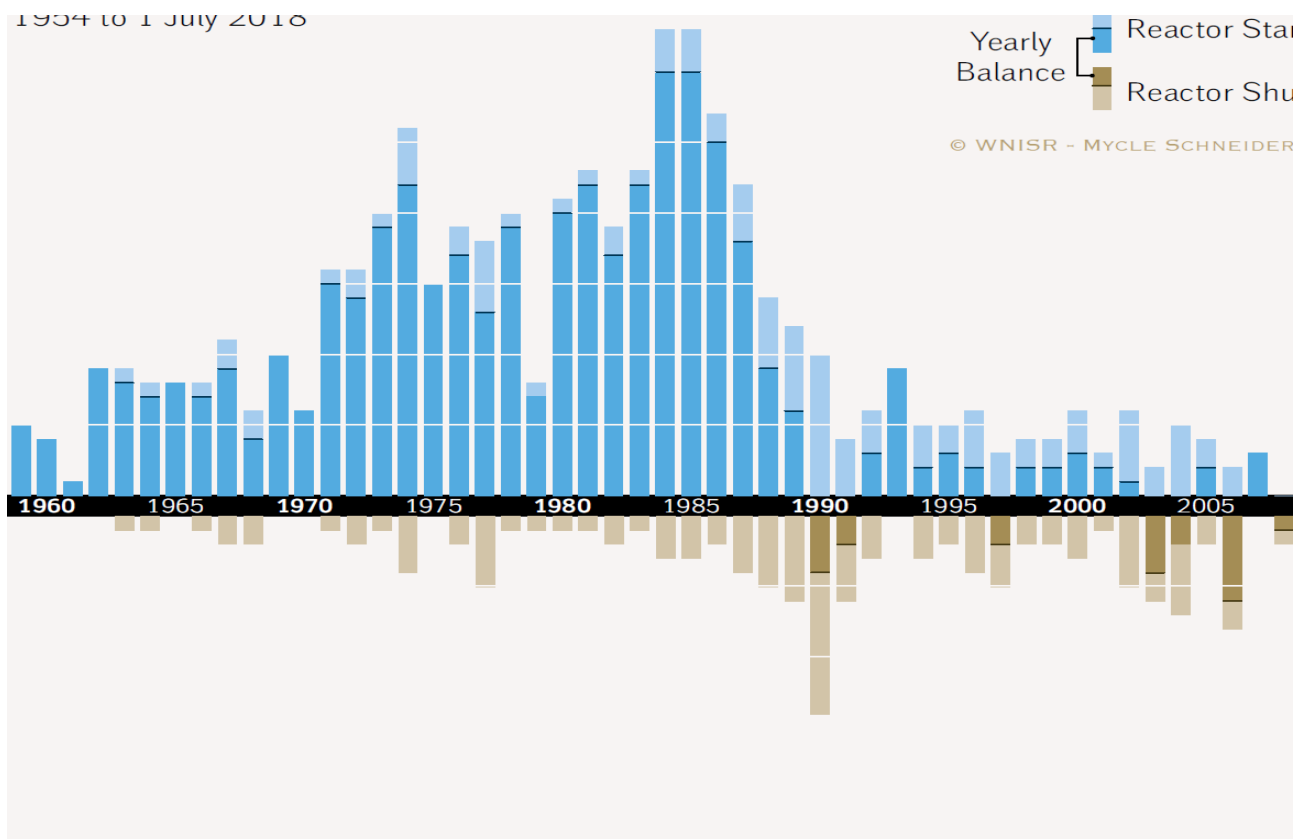


Figure 6: Opening and closure of nuclear reactors over time (Schneider et al, 2018).

Combined, Figures 5 and 6 clearly and absolutely show that there is no real growth in nuclear power occurring globally – despite new capacity being bought online in China. Furthermore, given that reactors rarely last much longer than 25-30 years, there are many reactor complexes built in the 1970s and 1980s which are due for closure very soon – meaning the trend of reactor closures will accelerate and negate any growth occurring in China. **I highly recommend that the Committee not only study the Schneider-Froggatt work but also that of Australian research Dr Jim Green, who has published and written widely on the current status of nuclear power.**

4. Uranium Mine Rehabilitation

A major area of concern with nuclear power is uranium mine rehabilitation. In the United States and Germany, billions of tax-payer dollars have been spent on the remediation and rehabilitation of old, mostly Cold War-era uranium mines which were in a derelict and dangerous state. In Germany, this was a 25 year program known as 'Wismut', remediating much (but not all) of the former Soviet legacy of uranium mining in the former East Germany states of Saxony and Thuringia. In the USA, given that much of the 1950s to 1970s era of uranium production was for military purposes, the federal government initiated the 'Uranium Mill Tailings Reclamation Act' (or UMTRA) program. Despite the often modest size of these mines in comparison to modern uranium mines, they cost billions to address.

Australia's track record on uranium mine rehabilitation is not as expensive, but it remains poor and widely misunderstood or, at times, even misrepresented. I provide the following summaries of each site based on my own research (including site visits to almost all of these sites). I would be happy to provide further material to the Committee if this helps its deliberations.

Ranger, Northern Territory:

- Producing 1981-present, scheduled to cease production by 8 January 2021.
- All rehabilitation works to be completed by 8 January 2026.
- Required to place all tailings in former mine pits and ensure no environmental impacts from tailings-derived solutes for 10,000 years.
- At some point in the future, the Ranger Project Area should be capable of being incorporated into the world heritage-listed Kakadu National Park.
- Despite extensive assurances, the costs of full site rehabilitation to exceptionally high standards – as promised by the company and government – are still constantly being revised upward and have now reached more than \$800 million. There is a real risk that the site cannot fully fund this from remaining expected production revenue.

Mary Kathleen, Queensland:

- Produced 1958-63, 1976-82.
- Rehabilitation works completed by 1985 and won national engineering excellence awards.
- The tailings dam was expected to show minimal seepage, no acid mine drainage, no uptake of heavy metals and radionuclides into overlying vegetation, by physically stable against erosion.
- Research in the late 1990s (e.g. Lottermoser et al, 2005) has shown that of the above assumptions, only erosion has proven correct with the site showing substantial acidic seepage occurring and there is uptake of heavy metals and radionuclides into overlying vegetation (proving a risk to cattle; see Lottermoser, 2011).
- Recent recognition by the Queensland Government that, despite promises, further rehabilitation works will be required.

Radium Hill, South Australia:

- Produced 1954-62 (as well as a very minor era of radium mining in the 1910s and 1920s, with radioactive waste from this period still on suburban blocks in central Sydney; see Mudd, 2005).
- Some site rehabilitation works in early 1980s which included dumping of some radioactive wastes from South Australian government research facilities (very secretive though, very little public documentation is available).
- By early 2000s erosion was notable in the covers over the tailings, facilitating dispersal into the immediate surrounds (see Lottermoser & Ashley, 2006).

Port Pirie, South Australia:

- Produced 1955-63.
- Tailings dam still allowing children to swim until about the mid-1970s when the site was fenced off – waters were still acidic.
- Rehabilitation works carried out in the early 1980s, simply by covering with slags from the adjacent lead smelter.

Nabarlek, Northern Territory:

- Produced 1981-88.
- Rehabilitated 1994-95, with minimal environmental monitoring.
- Detailed post-rehabilitation radiological assessment by Martin (2000) showed small increase in gamma radiation levels across the disturbed footprint – specifically about a 50% increase from pre-mine levels over an area 20 times the original deposit footprint.
- Major ongoing issues with revegetation and weeds, especially when Cyclone Monica caused widespread destruction of the plants and trees across the site.
- Concerns remain about groundwater contamination from the tailings in the former open cut.

Upper South Alligator Valley, Northern Territory:

- Produced 1956-64.
- Minimal rehabilitation works undertaken until the 2000s when major works were completed, including shifting radioactive wastes to an engineered containment facility.
- Management of the waste facility is showing physical stability but emerging issues from monitoring include potential seepage into the facility, raising concerns over seepage risks.

Rum Jungle, Northern Territory:

- Produced 1954-71.
- Extreme water pollution issues from acid mine drainage (AMD), leading to the effective death of the Finniss River for some 20 km.
- Major engineering works completed to reduce AMD problems in the early 1980s, principally by placing specifically designed soil covers over the waste rock dumps to ensure much lower rates of AMD generation and emplacing all tailings in the former open cuts; site widely acclaimed as successfully rehabilitated.
- By the mid-1990s, AMD was again a major problem – with investigations showing that not only were the soil covers built incorrectly in parts of the site but also that the design approach failed in any case. The failure at Rum Jungle is now widely viewed as a textbook case study in unsuccessful rehabilitation design – although some leeway can be given that this was amongst the first large scale mine rehabilitation projects designed to reduce AMD risks and that understanding was insufficient (i.e. we know more about the engineering these days).
- Since the mid-2000s, the Australian Government has spent >\$10 million in further rehabilitation studies to again try and reduce AMD pollution problems at the site – except this time the preferred approach is to completely backfill waste rock into the former open cuts to ensure that the AMD generation is completely shut off due to the waste being below the water table.

Olympic Dam, South Australia:

- Production 1988-present, with decades to centuries of potential future operations.
- Major standards for future rehabilitation remain undecided and left to the future.
- Issues with seepage from the tailings dams to groundwater.

Beverley & Four Mile, South Australia:

- Production 2001-present.
- No requirements for remediation of groundwater impacted by the acid in-situ leach mining process.

Honeymoon, South Australia:

- Production 2011-2013 (presently in care & maintenance and could re-open in the future).
- No requirements for remediation of groundwater impacted by the acid in-situ leach mining process.

A montage of photo's and images of many of the above uranium mine sites are shown below in Figure 7.

The simple message: Australia has not demonstrated successful long-term uranium mine rehabilitation at any site. In other words, all sites still exhibit various problems ranging from local impacts or risks to severe risks to adjacent streams and land use restrictions. Perhaps most alarmingly, there remains a complete lack of agreed standards as to define an acceptable standard of rehabilitation – such as gamma radiation, radon & progeny, water quality, ecosystem re-establishment, erosion – but most critically the time frame over which site monitoring and maintenance needs to occur.



Figure 7: Examples of the rehabilitation standards at selected former Australian uranium mines

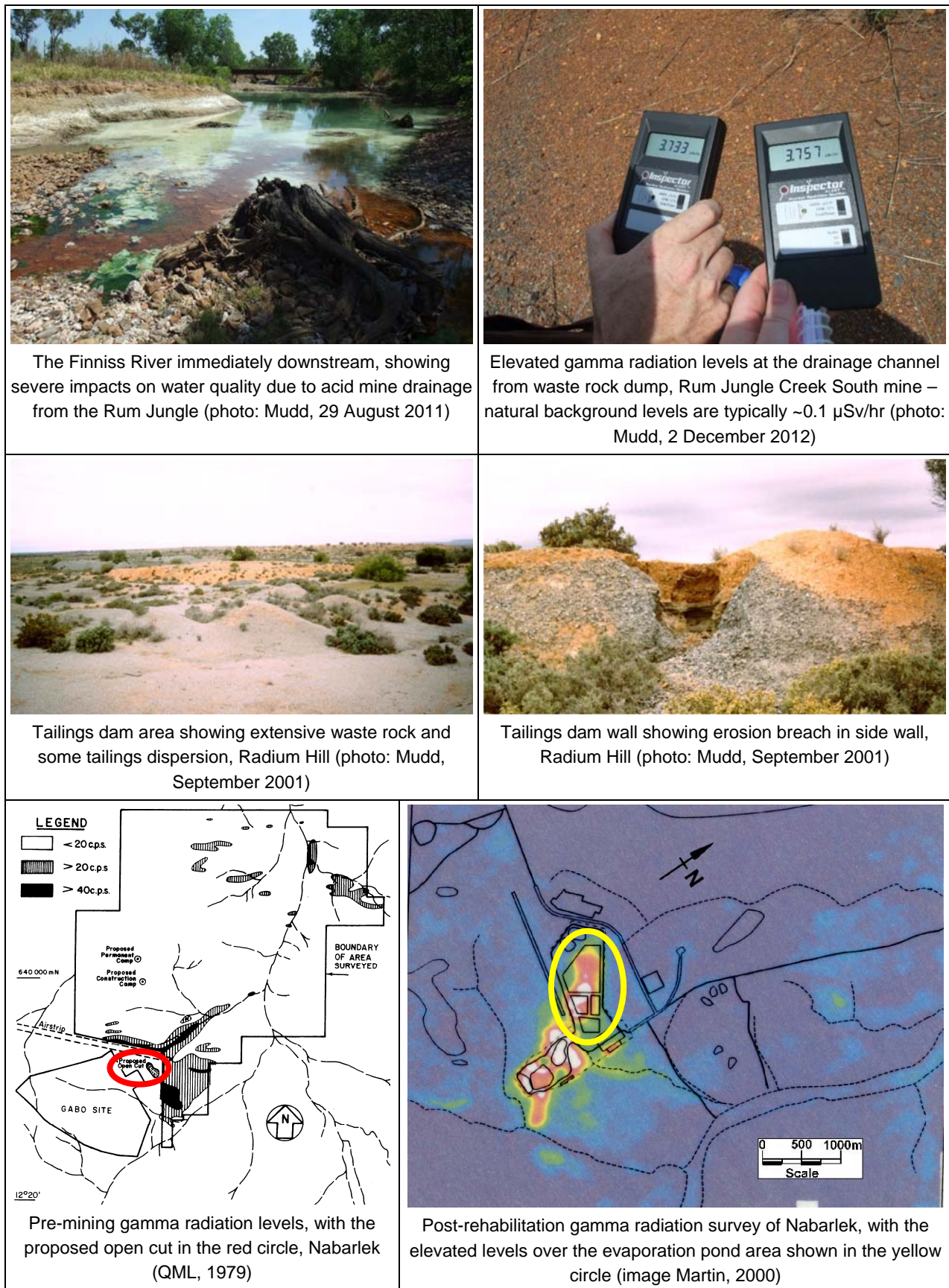


Figure 7: Examples of the rehabilitation standards at selected former Australian uranium mines (continued)

5. Nuclear Power Remains Inextricably Linked to Nuclear Weapons

I would like to make my final point that nuclear power continues to support the existence of and enhance the risk of the use of nuclear weapons. Australia should immediately join the Nuclear Weapons Ban Treaty, which was achieved through the action of an Australian-origin civic society group, the International Campaign to Ban Nuclear Weapons – or ICAN.

6. Academic Publications

Finally, as part of my submission, I attach the following detailed academic studies I have published over the years:

- Mudd, G M, 2001, *Critical Review of Acid In Situ Leach Uranium Mining: 1. USA and Australia. Environmental Geology*, December 2001, Volume 41, Issue 3-4, pages 390-403.
- Mudd, G M, 2001, *Critical Review of Acid In Situ Leach Uranium Mining: 2. Soviet Block and Asia. Environmental Geology*, December 2001, Volume 41, Issue 3-4, pages 404-416.
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- Mudd, G M, 2014, *The Future of Yellowcake: A Global Assessment of Uranium Resources and Mining. Science of the Total Environment*, Volume 472, pages 590-607.

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Critical review of acid in situ leach uranium mining: 1. USA and Australia

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Abstract The technique of in situ leach (ISL) uranium mining is well established in the USA, as well as being used extensively in Eastern Europe and the former Soviet Union. The method is being proposed and tested on uranium deposits in Australia, with sulfuric acid chemistry and no restoration of groundwater following mining. Test sites in the USA were required to restore groundwater to ascertain the extent of impacts and compare costs to alkaline ISL mines. The problems encountered include expensive and difficult restoration, gypsum precipitation, higher salinity and some heavy metals and radionuclides after restoration. One of the most critical issues is whether natural attenuation is capable of restoring groundwater quality and geochemical conditions in an acid leached aquifer zone. The history of acid ISL sites in the USA and Australia are presented in this study, with a particular focus on the demonstration of restoration of groundwater impacts.

Keywords Acid · In situ leach · Solution mining · Uranium plumes

Background

The unconventional mining technique of in situ leach (ISL) is now the primary producer of refined uranium in the United States, with a market share of around 95% in the mid 1990s (DoE 1999). The ISL technique, also known

as 'solution mining', appears set to assume a new part of Australia's uranium industry.

It is an historical curiosity as to where the conceptual processes for ISL were first conceived and applied. The Chinese were apparently the first to use solution mining to produce copper as early as 907 A.D., with references to the technology dating back to 177 B.C. (Morris 1984). During the 1890s, the Frasch process for mining elemental sulfur was invented, and ISL mining of gold was first suggested by the Russians (Morris 1984). The first trials of uranium ISL were developed in the USA and the Soviet Union in the early 1960s. It is uncertain who developed the first projects or if they originated separately (Mudd 1998).

By the mid-1970s, there were uranium ISL mines across the world as alternative, potentially low cost producers (Larson 1981). In the USA, ISL mines generally used alkaline chemistry with only a few sites trialling acid chemistry. In contrast, Soviet mines generally used acid with only a few sites using alkaline reagents. In Australia during the early 1980s, two ISL projects at Beverley and Honeymoon in South Australia proposed acid while a third at Manyingee in Western Australia trialled alkaline chemistry. The Honeymoon site undertook an acid ISL trial in late 1982.

The environmental regulation of mining generally requires the restoration of affected groundwater to be returned to its pre-mining quality or use category. In countries controlled by the Soviet block, the need for restoration of contaminated groundwater following mining was ignored during operation, and the problems and magnitude of groundwater contamination, which are now coming to light in the 1990s, can only be described as extreme (Mudd 1998, 2001).

Indeed, the use of alkaline chemistry in the USA has been related to the need to restore affected groundwater and that alkaline mine sites are considered to be easier to restore (Tweeton and Peterson 1981; Mudd 1998). In direct contrast, Australian ISL projects – historically and currently – proposed not to restore affected groundwater after acid ISL mining.

The resurfacing of the Australian acid ISL uranium mine proposals in 1996 (because of a change in federal government and uranium policy), the lack of acid ISL mines in the USA, the research coming to light through the International Atomic Energy Agency (IAEA) and others of the extent of impacts from acid ISL mines in the Soviet block, led to a detailed review of ISL uranium mining (Mudd

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1998). The history and experience of acid ISL in Australia and the United States is reviewed herein, with the sites of the Soviet Block reviewed in the companion paper (Mudd 2001).

Acid ISL in the USA

Brief history of ISL

The initial development of ISL mining in the USA occurred in Wyoming at the Shirley Basin uranium project from 1961–1963 by the Utah Construction and Mining Company (now Pathfinder Mines Corporation; Larson 1981). They experimented with five generations of well-field design and over 100 patterns using sulfuric acid leaching chemistry (Underhill 1992). The Shirley Basin ISL project operated on a small scale from 1963–1969 to produce 577 t U (Underhill 1992); however, it was closed and converted to an open pit operation from 1970 (Larson 1981).

The late 1960s to mid 1970s witnessed rapid development and promise in ISL mining, principally in Texas and Wyoming (Kasper and others 1979). By May 1980, a total of 18 commercial and 9 pilot-scale projects were either in operation or under active development (Larson 1981). Virtually all of these sites utilized alkaline reagents such as ammonia- or sodium-carbonate/bicarbonate. The difficulty of restoring ammonia-based sites saw a quick shift in emphasis to sodium bicarbonate- or carbon dioxide-based leaching chemistry by the early 1980s (Tweeton and Peterson 1981). Despite years of lower production in the late 1980s, ISL mines have gradually increased their share of uranium production in the USA from about 1.2% in 1975 (Underhill 1992) to greater than 90% during the mid-1990s (DoE 1999).

By 1991, a total of 62 ISL projects had been developed, although only 24 of these sites were commercialized (Underhill 1992), indicating more unsuccessful than successful projects (Rojas 1987). There has been no development of a commercial ISL mine using acid chemistry since the Shirley Basin experimental project (Mays 1984). Further detail on all ISL mines is given by Mudd (1998). There were some sites in New Mexico, Texas and

Wyoming, which underwent pilot-scale testing of acid ISL, compiled in Table 1, although most were poorly documented in the public literature (Mudd 1998).

The most documented acid ISL project is Nine Mile Lake near Casper, Wyoming. The project was developed by Rocky Mountain Energy Co. (RMEC) in association with research by the US Bureau of Mines. The landmark study was reported in detail by Nigbor and others (1981, 1982). RMEC's Reno Ranch acid ISL trial in Wyoming was reported by Staub and others (1986). Further acid ISL trial sites have not been reported widely in the literature. Although trialled at some sites, acid systems were generally considered unsuitable for Texan deposits because of higher carbonate (Hunkin 1977).

Nine Mile Lake, Wyoming

The geology and hydrogeology of the Nine Mile Lake (NML) site is given by Nigbor and others (1981, 1982) and Staub and others (1986). The following discussion is based on these references.

The NML uranium deposit is on the south-west flank of the Powder River Basin. The roll-front uranium mineralization occurs in the Teapot Sandstone within the Mesaverde Formation. The ore body extends over a strike length of 6,100 m in a north–north-west direction and ranges between 15 to 900 m in width, consisting of upper and lower zones. The site is at an elevation of 1,600 m. The uranium at NML was precipitated at the interface of oxidation–reduction boundaries in the Teapot sandstone because of the presence of carbonaceous material and pyrite. The principal uranium mineral was uraninite (UO_2), with minor quantities of coffinite (USiO_4); grades ranged from 0.04–0.12% U_3O_8 . Vanadium was associated with the mineralization (range 0.1–0.4% V_2O_5), and was proposed to be extracted from a commercial facility. The ore zones contained less than 0.1% carbonate, although total carbon content was higher at 0.2–2.0%. The major clay mineral present was kaolinite (2–5%), with minor montmorillonite, although this had a low cation exchange capacity at 5 meq/100 g.

Because of the low carbonate content of the ore body and the low cost of sulfuric acid, NML was considered an ideal test site for acid ISL mining. Extensive laboratory tests on

Table 1

Pilot scale ISL mines using acid leaching chemistry from Staub and others (1986), Underhill (1992) and Mudd (1998)

Project/site	Company	Time period
Nine Mile Lake, Wyoming	Rocky Mountain Energy Co.	Mining: Nov. 1976 to Nov. 1980 Restoration suspended: Feb. 1982
Reno Ranch, Wyoming ^a	Rocky Mountain Energy Co.	Mining: Feb. 1979 to Nov. 1979 Restoration suspended: March 1981
Irigary, Wyoming	Wyoming Minerals	Unclear – acid trial referred to by Kasper and others (1979)
Jackpile Paguete, New Mexico ^b	Anaconda	Early 1970 trial, two well-fields, with two in jection bores and 18 extraction bores, up graded to 29. Project discontinued
Dunderstadt, Texas	Cities Service	Trial operated between 1969–1971. No reports
Besar Creek, Texas	Rocky Mountain Energy Co.	Early 1970s?, details unclear (plant later used at Nine Mile Lake)

^aAlso called Reno Creek

^bPart of the North Windup Project

core samples suggested that savings in chemical costs would result from the use of acid. A total of four well-field patterns underwent testing and development at the NML project site. The chronology and detail for each pattern is summarised in Table 2.

Pattern 1, completed in the upper ore zone, experienced several problems that lead to poor operational performance. These included problems with the PVC well casing, cement baskets and pumps. A build-up of gypsum scale on the injection well screens, possibly related to the degradation of the casing cement by the acid, contributed to poor injectivity. Potential channelling and poor injectivity led to disappointing overall uranium recovery.

Pattern 2, completed in the lower ore zone, with a detailed assessment provided by Nigbor and others (1982), was considered a success. Injectivity was good, although plugging problems caused by 'fungus growth' and gypsum precipitation were encountered in April 1978. No evidence was provided to substantiate the 'fungus growth'.

The two injection bores of pattern 3 were completed in both the upper and lower ore zones to test the feasibility of simultaneously leaching both zones. The extraction bores were completed independently in each ore zone. The pattern experienced sporadic problems with well plugging, frozen lines and equipment failures, leading to poor operational performance. Further problems were encountered in controlling solution distribution to the two ore zones.

Pattern 4, using alkaline chemistry, was intended to give a comparison of alkaline and acid leaching on the same ore body. However, the results of this trial are not available although RMEC described it as "disappointing". No comparison can be made of the respective advantages and disadvantages of acid versus alkaline for this deposit.

There were five horizontal excursions detected at NML during testing, with three in pattern 3 and two in pattern 4. All excursions were brought under control by increasing the extraction rate. No monitoring of overlying and underlying bores was undertaken, and determination of any vertical excursions is not possible. This potential exists at every ISL mine because of casing failures and improperly sealed exploration bores (Marlowe 1984; Staub and others 1986). The risk increases with the total number of bores and age of a site (Marlowe 1984).

The restoration of each pattern was undertaken immediately after mining, followed by the regulatory period of monitoring to ensure the stabilization of groundwater quality. Post-restoration monitoring is critical in understanding the effectiveness of restoration efforts and long-term impacts on water quality at NML. The available baseline, leaching phase and restoration groundwater quality data for each pattern is compiled in Tables 3 and 4.

The restoration data is averaged from observation and extraction bores because of the tendency of injection bores to reflect the quality of injected solutions rather than groundwater after mixing in the aquifer. The high sulfate levels of the ore zone were thought to be related to influx from Nine Mile Lake itself, 1.6 km to the south, which is naturally high in sulfate.

The methods for restoring each pattern differed slightly. Pattern 1 was restored using a groundwater sweep, whereas pattern 3 involved a groundwater sweep combined with reverse osmosis treatment and mixed with 'clean' formation water before re-injection into the ore zone.

The post-restoration monitoring of pattern 1 from early 1978 to 1981 indicated substantial deterioration of water

Table 2

Research and development details for Nine Mile Lake. *Rest'n* Restoration

Pattern and type	Leaching solution chemistry	Period of testing ^a	PV ^b
1 Seven-spot, 15 m radius	4 g/l H ₂ SO ₄ (pH 1.7), 0.5 g/l H ₂ O ₂ , 0.15 g/l FeSO ₄ , flow ~2.5 l/s	Mining: Nov. 1976 to Aug. 1977 Rest'n: Sep. 1977 to Oct. 1978	7 12
2 Five-spot, 15 m radius	3–5 g/l H ₂ SO ₄ (pH 1.8), 1 g/l H ₂ O ₂ , flow ~2.6 l/s	Mining: Dec. 1977 to Sep. 1978 Rest'n: Sep. 1978 to Aug. 1979	13 12
3 Eight-spot ^c , 18 m radius	H ₂ SO ₄ , H ₂ SO ₅ or O ₂ , flow ~3.8 l/s	Mining: Sep. 1979 to April 1980 Rest'n: Aug. 1981 to Jan. 1982	5.6 6
4 Five-spot, 15 m radius	Na ₂ CO ₃ /NaHCO ₃ with CO ₂ (g) (pH~7.5), 0.5 g/l H ₂ O ₂ (later) O ₂	Mining: June 1980 to Nov. 1980 Rest'n: Nov. 1980 to Aug. 1981	? ?

^aRestoration refers to initial phase only

^bAquifer pore volumes reached during testing

^cIncluded two central injection and six extraction bores (effectively, one three-spot pattern for each ore zone aquifer)

Table 3

Baseline and restoration groundwater quality, patterns 1 and 3, Nine Mile Lake. All mg/l; except pH, EC in mS/m. From Staub and others (1986)

Pattern and phase	TDS	EC	pH	Cl	SO ₄	Ca	U ₃ O ₈	V
1 Baseline	2,483	316	6.9	3.3	1,240	87	0.384	0.1
1 Restoration	7,750	1,200	6.9	93	5,140	300	0.289	0.073
3 Baseline	2,034	238	6.9	35	1,244	74	0.060	0.18
3 Restoration	1,450	250	7.1	26	920	61	0.126	0.57

Table 4

Average baseline, leaching phase and restoration groundwater quality, with standard deviation, pattern 2, Nine Mile Lake (Nigbor and others 1981, 1982). Leaching values indicative only

	Units	Phase		
		Baseline	Leaching	Restoration ^a
TDS	mg/l	4,300±550		3,000 (2,390)
EC	mS/m	410±51	1,000–1,500	300 (308)
pH	–	6.7±0.3	1.5–2.0	6.1 (6.9)
Redox	mV	–120±200		–22 to 120
Diss. O ₂	mg/l	1		<0.1
Cl	mg/l	46±4.3		29 (37)
SO ₄	mg/l	2,510±244	Up to 8,000	1,585 (1,584)
HCO ₃	mg/l	290±30		
F	mg/l	0.77±0.25		0.6
Ca	mg/l	207±43	260	805 (102)
Mg	mg/l	92±31		42
Na	mg/l	830±145		485
K	mg/l	14±3.6		6.2
Al	mg/l	0.13±0.05		
Fe	mg/l	1.07±0.4	Up to 200	6.8
Mn	mg/l	0.31±0.18		0.24
As	µg/l	40		24
B	mg/l	0.67±0.40		
Cr	mg/l	0.01		
Cu	mg/l	0.01		
Hg	mg/l	0.01		
Mo	µg/l	8±18		
P	mg/l	0.2		<0.1
Se	µg/l	2		13
Si	mg/l	4.2±4.0		14.8
V	mg/l	0.5±0.2	Up to 800	11.1 (0.986)
Zn	mg/l	0.02±0.02		1.97
²²⁶ Ra	Bq/l	18.9±1.1	370±6.3	
²³⁰ Th	Bq/l	0.0031±0.00019	1,813±118	
U	mg/l	0.23±0.10	80–150	1.05 (0.132)

^aIncludes additional restoration work undertaken in 1981–1982 in parentheses (Staub and others 1986)

quality because of gypsum dissolution increasing salinity levels. Reverse osmosis treatment of approximately 2.5 pore volumes of recirculated groundwater was undertaken in 1981, although later monitoring again showed deterioration and stabilization at a high salinity level. The water quality, with salinity four times higher at 7,750 mg/l and SO₄ three times higher at 5,140 mg/l, is now unsuitable for stock purposes – the pre-mining use category.

The restoration of pattern 2, however, proved to be more recalcitrant. The first phase of restoration involved 4 months of a modified groundwater sweep with re-injection of process water and barren production fluid. Restoration using reverse osmosis treatment was then undertaken for a month. From May to mid-August 1979, a high pH, sodium hydroxide solution was injected to promote ion exchange and speed restoration. Clean water recycling with reverse osmosis continued for the next 3 weeks, by which stage nearly all major parameters were restored to pre-mining ranges, and active restoration ceased.

Post-restoration monitoring of pattern 2 during late 1979 and early 1980 detected scattered areas of contaminated groundwater around the pattern interior, migrating slowly down gradient. Pumping resumed in August 1980,

involving about 3.5 aquifer pore volumes, with the groundwater being treated with a lime/barium chloride precipitation process and re-injected into the well-field. Little improvement was apparent and, by May 1981, water quality was again deteriorating. As of June 1984, V, ²²⁶Ra and TDS remained above pre-mining levels.

The restoration of pattern 3 returned most parameters to baseline values or better but failed to restore U, V and ²²⁶Ra to pre-mining levels.

The Nine Mile Lake acid ISL trial demonstrated that acid was indeed an effective leaching alternative to the alkaline chemistry prevailing at the time, albeit non-selective. Other issues raised by the trial include the difficulty in scaling laboratory test results to the field. The column leaching tests performed on NML core samples suggested significantly lower reagent consumption than that required in the field. The restoration of the laboratory columns indicated that about 13 pore volumes would be required to restore water quality, whereas in the field it was closer to 20 and still experienced deterioration after treatment efforts. Nigbor and others (1982) concluded that because of greater reagent consumption and the difficulty and expense of restoration acid leaching was no more cost effective than alkaline leaching.

The expansion of the NML site to commercial scale by RMEC proposed to use a seven-spot production pattern with a radius of 21 m, 3–5 g/l H₂SO₄ and 1 g/l H₂O₂ with V recovery. The site is yet to be developed.

Reno Ranch, Wyoming

The Reno Ranch (Reno Creek) uranium deposit in Wyoming, although lesser known than the Nine Mile Lake site, underwent a trial of acid ISL at about the same time period. An alkaline six-spot trial was also developed. Unlike NML, however, the geology, hydrogeology and information on both the acid and alkaline ISL trials at Reno Ranch was published by Staub and others (1986).

Reno Ranch is on the eastern flank of the Powder River Basin. The roll-front uranium mineralization occurs in the Wasatch Formation, consisting of fluvial sandstones, siltstones, shales, claystones and coal seams. The ore zone contains high quantities of carbonate minerals, although quantitative data are unavailable. The site is at an elevation of 1,590 m.

Two well-field patterns were developed and tested, the first being a conventional five-spot pattern and the second being a six-spot pattern with two injection and four extraction bores. Pattern 1 was leached with acid whereas pattern 2 with alkaline reagents, details are in Table 5. Results for the two patterns are in Tables 6 and 7, respectively.

After mining was initiated in pattern 1, problems with gypsum precipitation and 'fungus growth' reduced the efficiency of well field circulation. No evidence was provided for the 'fungus growth'. The uranium recovery rates were low and the carbonate minerals in the host sandstone consumed high quantities of acid. Leaching was terminated prematurely and restoration began immediately, consisting of water treatment by ion exchange, groundwater sweeping and treatment with potassium carbonate to

Cases and solutions

Table 5

Research and development details for Reno Ranch. *Rest'n* Restoration

Pattern and type		Leaching solution chemistry	Period of testing	PV
1	Five-spot, 12 m radius	5 g/l H ₂ SO ₄ (pH 1.8), H ₂ O ₂ , flow ~2.5 l/s	Mining: Feb. 1979 to Nov. 1979 Rest'n: Nov. 1979 to Oct. 1981	?
2	Six-spot ^a , 15 m radius	Na ₂ CO ₃ /NaHCO ₃ , H ₂ O ₂ , flow ~1.6 l/s	Mining: Sep. 1980 to Dec. 1980 Rest'n: Dec. 1980 to April 1981	?
				6.5

^aIncluded two central injection and four extraction bores

Table 6

Range of baseline, leaching phase and restoration groundwater quality in the Reno Ranch ore zone for pattern 1 (acid). All mg/l; except pH; EC in mS/m. From Staub and others (1986). *Mon./Prod.* Monitoring/production bores; *ND* Not detectable

	Baseline		Leaching Range	Restoration Feb. 1981	Post-restoration ^a	
	Average	Range			Mon.	Prod.
TDS	1,176	283–1,597			1,267	2,551
EC	155	122–200				
pH	8.1	6.4–12.2	2.7–7.6	4.9	9.3	5.3
HCO ₃	91	ND–305				
CO ₃ ^b	50	ND–281				
Alk. ^c	89	ND–225				
SO ₄	588	11–1,006	981–3,928	1,482	764	1,551
Cl	24	6–66				
F	0.46	ND–0.57				
NH ₄	0.4	ND–16.8				
NO ₃	0.5	ND–7				
Na	228	87–322				
K	11.3	5–29				
Ca	104	72–203	156–666	217	102	263
Mg	21	1–51				
Fe	1.0	ND–3	<0.1–242	38.5		
Al	0.55	ND–1.45				
As	0.02	ND–0.03				
B	0.9	ND–2.6				
Mn	0.05	ND–0.22				
Se	0.02	ND–0.05				
SiO ₂	5.7	ND–8.7				
U ₃ O ₈	0.46	0.001–2.0	<0.1–22.0	1.4	0.059	0.64
V	<1.0	0–8	<0.1–9.0	0.4		

^aPost-restoration groundwater quality in March 1983

^bAs CO₃

^cAlkalinity as CaCO₃

Table 7

Range of baseline, leaching phase and restoration groundwater quality in the Reno Ranch ore zone for pattern 2 (alkaline). All mg/l, except pH; EC in mS/m; ²²⁶Ra in Bq/l. (Staub and others 1986)

	Baseline		Leaching Range	Restoration Post-leach	Post-restoration	
	Average	Range			Post-IX	Post sweep
TDS	1,438	1,384–1,506				
EC	199	189–223	350	340	200	199
pH	8.9	8.4–9.5	7.2	7.4	7.7	7.7
HCO ₃	84	11–178	1,800	1,670	160	125
CO ₃ ^a	9.9	2–32				
Alk. ^b	80	45–109				
SO ₄	918	890–975				
Cl	10.4	7.0–18.8	240	113	19	15
Na	296	273–360	900	770	305	322
K	13.8	7.4–44				
Ca	119	102–153	330	207	69	87
Mg	21	12–26				
Fe	–	0.03–0.61	8.0	0.6	0.16	0.39
²²⁶ Ra	–	3.9–28.4		11.5	8.8	8.2
U ₃ O ₈	0.049	0.007–0.287	65	16	1.64	1.37
V	0.07	0.04–0.34	6	3	1.05	0.45

^aAs CO₃

^bAlkalinity as CaCO₃

raise the pH and facilitate further removal of calcium, heavy metals and radionuclides. The restoration sequence, although aggressive compared to other efforts at ISL mine sites, encountered many difficulties.

The ongoing restoration efforts of pattern 1 failed to reduce free acidity, SO_4 and ^{226}Ra levels. RMEC proposed the use of a high salinity solution to displace the hydrogen ions from clay lattice structures, enabling these to be removed during the restoration process. The regulatory agencies refused this technique because of uncertainties and possible adverse effects on the aquifer and groundwater quality.

Very little post-restoration water quality improvement has occurred at pattern 1. During the first quarter of 1983, groundwater monitoring indicated that pH levels in the aquifers have not improved significantly, Ca and SO_4 concentrations have not changed significantly (270 and 1,500 mg/l, respectively), U levels have decreased marginally to less than 1.0 mg/l, and TDS (~2,650 mg/l) remains almost twice that before mining.

Pattern 2, leached with alkaline reagents, proved less problematic from an operational and restoration perspective; however, post-restoration monitoring indicated a significant increase in U concentration from 2.0 mg/l in April 1981 to 3.7 mg/l in February 1983. In comparison, the pre-mining U concentration was 0.23 mg/l. This pattern of increasing uranium concentrations after restoration has been noted at many restored ISL sites (Staub and others 1986). The V concentrations also remained significantly higher after mining and restoration efforts.

There were no reported excursions at the Reno Ranch site, although it was questionable whether the control limits were sensitive enough to detect such an event, especially for a vertical excursion. From a restoration perspective, the difference in post-restoration groundwater quality between acid and alkaline leaching appear minimal because both patterns experienced higher concentrations of some metals or radionuclides. The use of acid tends to exacerbate the concentrations of calcium and sulfate in groundwater, despite aggressive restoration efforts. As with Nine Mile Lake, the Reno Ranch site is yet to be commercialized, and new interest in the development of the deposit has been recently abandoned.

Acid ISL in the USA – summary

The experience with acid ISL uranium mining at Nine Mile Lake and Reno Ranch has shown that it can be an alternative, albeit non-selective, to the alkaline process. The choice, however, presents two major potential problems: (1) precipitation of gypsum on well screens and within the aquifer during mining, plugging wells and reducing the formation permeability (critical for economic operation); and (2) gradual dissolution of the precipitated gypsum following restoration, leading to increased salinity and sulfate levels in groundwater. The further effects of the release of heavy metals and radionuclides, especially ^{226}Ra , which were co-precipitated with the gypsum, have not been assessed or quantified. A critical issue is that acid leaching was not found to be more cost effective than alkaline, when taking restoration into consideration. No

commercial acid ISL uranium mine has yet been approved or developed in the USA and it remains unlikely in the near future.

Acid ISL in Australia

Brief history

The history of ISL uranium mining in Australia coincides with public concern regarding the environment, nuclear issues and indigenous land rights. Three sites have had pilot scale testing – two with acid, at Beverley (1998) and Honeymoon (1982 and 1998–2000) in South Australia, and one with alkaline chemistry at Manyingee (1985) in Western Australia. A location of Australian ISL deposits is shown in Fig. 1, with potential projects given in Table 8. There has been no commercial acid ISL copper mine, although several sites near Mt Isa, Queensland, have undergone trials in the late 1960s to mid 1970s (Bell 1984). More recently a variant of stope leaching was tested at the Gunpowder (Mammoth) copper mine (see Landmark 1992; Middlin and Meka 1993). A small experimental acid ISL copper project was trialled at the old Mutooroo mine, 100 km south of Honeymoon, during 1981–1982 (Bampton and others 1983). All projects proved difficult and sub-economic, and thus commercial acid ISL copper mining is yet to be realized in Australia.

A different site of note was the Western world's first proposed ISL gold mine at Eastville, central Victoria, in the early 1980s by mining company CRA Ltd (Bell 1984). The project planned to solution mine the deep alluvial gold leads by cyanide leaching chemistry (Hore-Lacy 1982). The community and regulators, however, expressed concerns about possible groundwater contamination arising from the use of cyanide in the rural farming area. After some initial hydrogeological pump and dye tracer tests (Hore-Lacy 1982), regulatory approvals were rejected (Bell 1984) and CRA later abandoned the project.

The use of solution mining has also been applied to potash mining in Western Australia, as well as for potential use in several other locations around Australia (Hancock 1988). The Beverley and Honeymoon uranium deposits, in the Lake Frome Embayment in north-eastern South Australia, were discovered in the early 1970s at a time when the prospects for nuclear power and uranium mining seemed promising. The Beverley project was originally planned as an open pit operation but was shelved in 1974 as sub-economic. The Honeymoon deposit, however, was recognized to be uneconomic by conventional mining from the outset, and by the late 1970s, ISL was being investigated as a possible economic alternative.

Alkaline push-pull tests were first conducted at Honeymoon in 1977 using ammonia-bicarbonate solutions, and results were disappointing (Dobrowolski 1983). A second push-pull test using sulfuric acid was undertaken in 1979, giving positive results and the companies proceeded to towards commercial development (Dobrowolski 1983). The Draft and Supplementary Environmental Impact Statements (EIS) were prepared (MINAD 1980, 1981) with

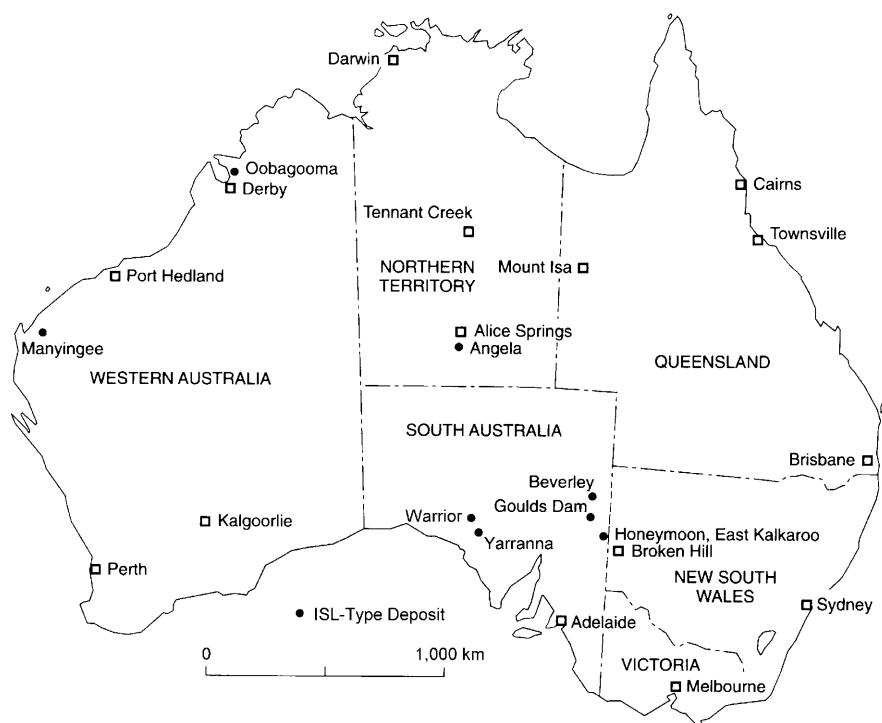


Fig. 1
Location of uranium deposits in Australia amenable to ISL

Table 8

ISL-amenable uranium deposits in Australia. References updated from recent company reports, Dunn and others (1990), Johns (1990), Mudd (1998)

Deposit	Discovered	t U ₃ O ₈	Grade	Company (and country)
Manyingee, WA	1974	12,078	0.08%	Paladin Resources Ltd (Australia)
Oobagooma, WA	1978	9,950	0.12%	Paladin Resources Ltd (Australia)
Bennett's Well, WA	1983	1,500	0.16%	Eagle Bay Resources NL (Australia)
Honeymoon, SA	1972	3,900	0.12%	Southern Cross Resources Inc. (Canada)
East Kalkaroo, SA ^a	1970	4,000	0.10%	Southern Cross Resources Inc. (Canada)
Gould's Dam, SA	1974	17,640	0.098%	Southern Cross Resources Inc. (Canada)
Beverley, SA	1969	21,000	0.18%	General Atomics (USA)
Paralana-Pepagoona, SA	1970	1,000	0.2%	General Atomics (USA)
Angela, NT	1974	12,000	0.1%	Black Range Minerals (Australia)

^aIncludes Yarramba deposit

federal government approval being obtained in late 1981 for pilot testing before commercial scale operations could proceed (Mudd 1998). A semi-commercial scale pilot plant, with a capacity of about 115 t U₃O₈/year, was built and operated briefly in 1982 but was plagued with operational problems because of jarosite precipitation and other issues (Mudd 1998).

The companies developing the Beverley deposit first investigated the use of ISL in about 1980, releasing their Draft EIS in 1982 (SAUC 1982). No Supplementary EIS was released and the environmental assessment process was not completed, however, and final approvals could not be given. Both projects proposed not to restore mining affected groundwater following the cessation of operations at each site, despite the acknowledged impacts on pollutant loads. In March 1983, the recently elected government of South Australia refused to issue mining leases for commercial operations at Beverley and Honeymoon, citing four reasons (Mudd 1998): (1) many of the economic, social,

biological, genetic, safety and environmental problems associated with the nuclear industry were unresolved; (2) endorsement of the Government's position by a wide range of community organizations; (3) commitment to the Roxby Downs (Olympic Dam) project; and (4) community unrest at the nature of the ISL process. The later introduction of the federal 'Three Mines Uranium Policy' in 1984 saw no further development until the election of a new federal government in 1996 and the immediate removal of this policy (Mudd 1998). The Honeymoon site was joined with all nearby deposits, including Gould's Dam 75 km north-west.

Further detail on the geology of ISL-type uranium deposits and their formation in the Lake Frome region is given by Callen (1975), Haynes (1975), Brunt (1978), Ellis (1980), Morris (1984), Johns (1990), Curtis and others (1990) and Hobday and Galloway (1999). The following discussion uses these references, except where noted.

Beverley ISL project

The Beverley deposit was discovered by the Oilmin-Transmin-Petromin (OTP) Group of companies in 1969, with drilling in 1970 confirming economic uranium grades (Mudd 1998). After nearly three decades and two unsuccessful attempts, the Beverley uranium deposit finally began development in 1996 towards a commercial operation through new owner Heathgate Resources Pty Ltd (HR), a wholly-owned Australian subsidiary of US-based General Atomics Corporation.

The geology and hydrogeology of Beverley is given in SAUC (1982) and HR (1998a). The 100 to 140 m of overburden consists of alluvial fans comprising lenses of gravels, sands, silts and clays. The uranium mineralization occurs within semi-isolated aquifer sands that resemble a concealed fluvial system or palaeochannel. The deposit contains three ore zones – Northern, Central and Southern, each with increasingly higher salinity, respectively. Beneath the ore zone aquifer is a thick mudstone sequence and the Cadna Owie sandstone of the Great Artesian Basin at 300 m depth. The hydrogeological relationships in the area are complicated by structural deformation and faulting, which may provide vertical interconnection between the deeper aquifers, while possibly truncating aquifers in the shallower sediments (Hancock 1986). A regional hydrogeological cross section is shown in Fig. 2. The potential for improperly sealed exploration boreholes (from the time when Beverley was planned as an open pit mine) to increase the risk of excursions has been recognized (Hancock 1986, 1988), although downplayed in more recent times.

A series of new groundwater bores were installed in 1996 and 1997, including two five-spot patterns, and hydrogeological pump testing was completed. HR applied for the operation of an acid ISL trial in late 1997 with no proposed groundwater restoration. The trial began on 2 January 1998 (before public release of a new EIS for the project) and was to leach the five-spot patterns in the Northern and Central ore zones, each for about 6 months.

After the release of the new EIS in July 1998, further studies were required by regulators to address outstanding uncertainties, such as the degree of isolation of the Beverley aquifer, long-term impacts on groundwater quality, and especially the potential to contaminate surrounding groundwater systems. HR still proposed not to restore affected groundwater following current and future mining operations. With completion of these extra studies, final government approvals were received in April 1999 with conditions on liquid waste disposal.

The significant outcomes of the approvals process for Beverley include (1) it is the Western world's first commercial scale acid ISL uranium mine (OECD, 2000), (2) it proposes to re-inject all liquid wastes back into the mined aquifer rather than deep re-injection (>1 km; as per some US sites) or evaporation (as per most US sites), (3) the extent of the palaeochannel system is underexplored beyond the surrounds of the three ore zones, and (4) it is the first mining project in modern Australian history not required to restore the majority of its environmental impacts

after cessation of operations (that is, groundwater contamination).

HR (1998a, 1998b) argued that following mining, the levels of radionuclides, heavy metals and pH will return to pre-mining conditions given several years; no mechanism was provided. This deserves critical assessment. The ore contains low sulfide (0.13%), organic carbon (0.05%), carbonate (0.06%), Fe, Mn and clay content (Hancock 1986, 1988; HR 1998a). Hancock (1986, 1988) argued that the exchangeable and soluble calcium and carbonate in the clays and sands surrounding the ore zones would be sufficient to neutralize the residual acid from migrating mining solutions and therefore precipitate gypsum. Because of the minimal degree of exploration data beyond the confines of the ore zones, however, this hypothesis remains untested and no data have been released to demonstrate this mechanism would perform satisfactorily at Beverley.

Buma (1979) argued that natural geochemical processes within aquifers can restore ISL-contaminated groundwater, thereby saving valuable chemical, energy and financial resources. The processes include precipitation of reduced compounds; scavenging of heavy metals by pyrite, organic matter, calcite and ferric oxyhydroxides; and adsorption by quartz, feldspars and clays. The key was for active reductants to be present. The conditions at Beverley, therefore, fail to provide any geochemical mechanism for natural restoration following acid ISL mining (because of the low abundance of active reductants). The 2-year-old trial site, if the data were to be published, might be able to shed important light on such behaviour.

Of further significance is that "reliance on this process (natural restoration) has never been tested" (Morris 1984). The time required and the rates at which natural geochemical processes could attenuate such levels of pollution are yet to be firmly established (Rojas 1987). The extreme levels of groundwater contamination wrought at acid ISL uranium mines across the Former Soviet Union suggests natural restoration appears to be spurious at worst, and ineffective at best (Mudd 2001).

The potential for excursions because of abandoned exploration bores still remains, as well as excursions caused by well casing failures (Marlowe 1984). Curiously, final approvals for Beverley included provisions that liquid waste re-injection only occur in the Northern zone – the zone of least exploration drilling and, importantly, the region of the best quality groundwater (TDS ~ 3 –6 g/l). This zone has similar water quality to pastoral use in the region (excluding radionuclides), although numerous mines in Western Australia often operate with much more saline groundwater (TDS up to 250 g/l).

The high Ca and SO_4 levels of the Beverley ore zones, especially the Central and Southern ore zones, create the potential for gypsum precipitation (see Tables 9, 10 and 11). This creates potential problems similar to Nine Mile Lake and Reno Ranch, both operationally and for post-mining geochemical conditions.

By August 1998, HR had apparently begun leaching of the Central trial pattern, although the full results from the

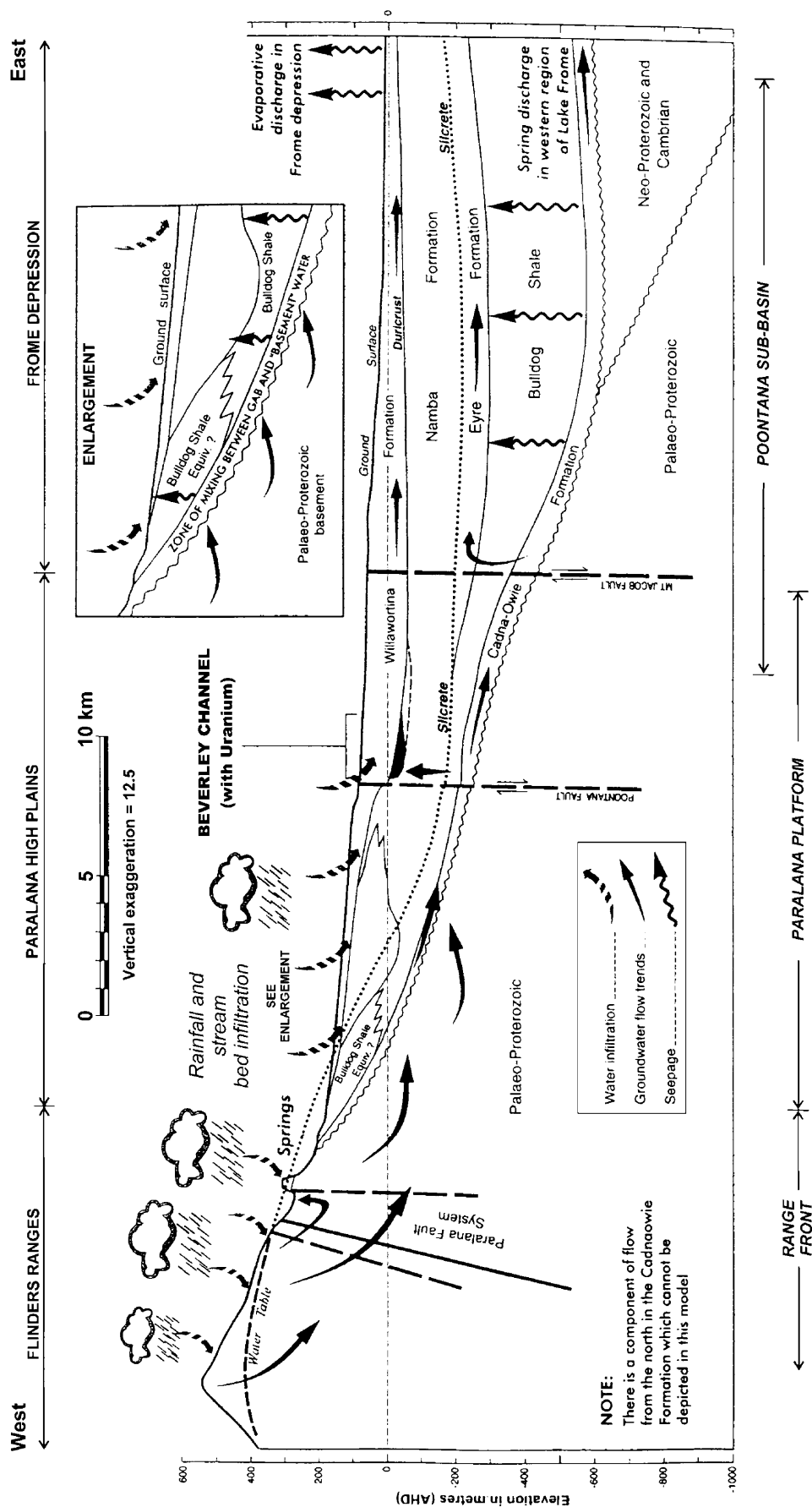


Fig. 2
Regional hydrogeological cross section of the Beverley deposit and region (HR 1998a)

Northern trial are yet to be published. This contrasts to the USA regulatory process where the results and restoration of a pilot scale facility form the permitting basis of a commercial mine (Kasper and others 1979). The construction of the commercial acid ISL mine at Beverley proceeded during 1999 and 2000, achieving operational status in late 2000.

Honeymoon ISL project

The Honeymoon deposit was Australia's first attempt at developing an ISL uranium mine. It is within the Yarramba palaeochannel, consisting of three distinct sands separated by thin, discontinuous clay layers. The Upper aquifer is occasionally used by pastoralists in the region while the Basal sand contains the uranium deposit. Traces of uranium mineralization exist in all three sands, with the Yarramba deposit to the north located in the Middle sand. The hydraulic head is identical for all three sands, suggesting a high degree of vertical interconnection. A hydrogeological cross section is shown in Fig. 3.

The deposit has several unique features related to the use of ISL, including pyrite content at 5–15%, compared with less than 2% in most USA deposits; higher salinity; low organic content (0.3%); and direct hydraulic connections between the three aquifers in the palaeochannel because of

gaps in the clay-confining layers. A compilation of groundwater and solution quality is given in Table 12. By mid-1982, the solvent extraction pilot plant and four five-spot patterns had been constructed (the fourth pattern intersected silt lenses with little or no uranium), and an ISL trial started using sulfuric acid and ferric sulfate as an oxidant. The trial encountered significant operational failure as a result of, principally, jarosite precipitation. The details have never been published, although it is known that preventing jarosite formation was difficult (Mudd 1998).

The new Canadian owners gained approval for a new trial at Honeymoon in March 1998, relying mainly on previous EIS approvals. The work began in April 1998 and continued through to August 2000. The new work is focused on trialling oxygen as the oxidizing agent, which should minimize the potential for jarosite formation, although other reagents, including hydrogen peroxide and ferric sulfate, are being tested. Detailed information from both the 1982 and new trial was not incorporated in the new EIS for the project, released in June 2000 (SCRA 2000a).

The approvals for Beverley set important precedents for ISL in Australia that have critical implications for the Honey-

Table 9

Groundwater and solution quality at Beverley: northern, central and southern ore zones, northern field leach trial data (injection and extraction averages March to July 1998) and Retention Pond (July 1998). Compiled from HR (1998a, 1998b), SAUC (1982). *n.a.* Not available

	pH	TDS (g/l)	S (g/l)	SO ₄ (g/l)	Cl (g/l)	F (mg/l)	Na (g/l)	K (mg/l)	Ca (mg/l)	Mg (mg/l)
Northern	7.3	3–6	n.a.	1.6	2	0.85	1.2	42	380	198
Central	7	6–10	n.a.	2.1	n.a.	n.a.	n.a.	n.a.	610	n.a.
Southern	6.8	11–13	n.a.	2.6	n.a.	n.a.	n.a.	n.a.	850	n.a.
Injection	1.93	11.5	1.6	4.79	2.0	7.67	1.43	59	610	337
Extraction	1.97	11.7	1.6	4.84	2.0	7.33	1.43	59	600	337
Ret. Pond	2.10	62.1	9.8	29.5	6.1	5.50	15.1	105	460	369

Table 10

Groundwater and solution quality at Beverley: northern, central and southern ore zones, northern field leach trial data (injection and extraction averages March to July 1998) and Retention Pond (July 1998). Compiled from HR (1998a, 1998b), SAUC (1982). *n.a.* Not available

	Al (mg/l)	Fe (mg/l)	Mn (mg/l)	Si (mg/l)	SiO ₂ (mg/l)	U (mg/l)	Ra ²²⁶ (Bq/l)	Rn ²²² (Bq/l)
Northern	0.2	0.7	0.2	48	n.a.	0.076	22–967	500–2,000
Central	n.a.	n.a.	n.a.	n.a.	n.a.	1.91	1.2–3,100	5–32,140
Southern	n.a.	n.a.	n.a.	n.a.	n.a.	0.70	13–111	20–585
Injection	91	109	0.7	138	294	2.9	8,414	n.a.
Extraction	91	105	0.8	133	283	162	9,881	n.a.
Ret. Pond	39	39	0.9	99	211	272	1,713	n.a.

Table 11

Groundwater and solution quality at Beverley: northern, central and southern ore zones, northern field leach trial data (injection and extraction averages March to July 1998) and Retention Pond (July 1998). Compiled from HR (1998a, 1998b), SAUC (1982). *n.a.* Not available

	B (mg/l)	Ba (µg/l)	Cd (µg/l)	Co (mg/l)	Cr (µg/l)	Cu (µg/l)	Ni (mg/l)	Pb (µg/l)	Se (µg/l)	V (µg/l)
Northern	1.6	53	0.2	0.1	20	30	0.004	40	1	1
Injection	1.0	37	117	20	100	200	8.47	160	410	1,100
Extraction	1.1	39	116	20	580	200	8.33	790	410	1,130
Ret. Pond	3.4	76	49	6.6	260	180	2.48	70	310	780

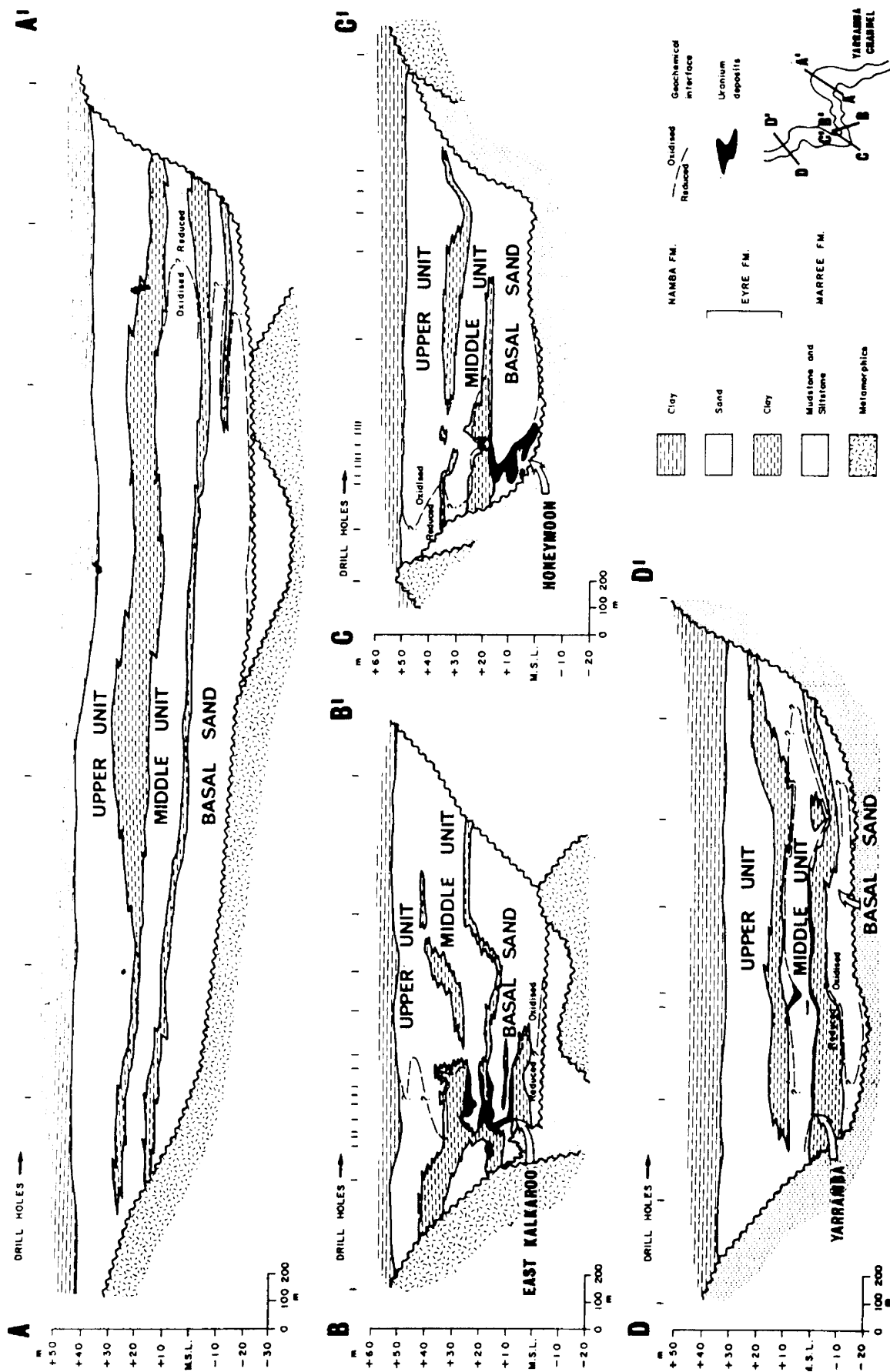


Fig. 3 Hydrogeological cross sections of the Yarramba Palaeochannel and the Honeymoon, East Kalkaroo and Yarramba uranium deposits (Brunt 1978)

Table 12

Groundwater and solution quality at Honeymoon (mg/l). Compiled from Bush (1999); SCRA (2000a, 2000b; Pirlo (2000). Units all mg/l, ²²⁶Ra, ²²²Rn and ²¹⁰Pb in Bq/l. *Org. C* Organic carbon; *Disposal* waste liquids for re-injection or surface evaporation; *RO brine* Rejected saline water from the water treatment (reverse osmosis) plant

	Basal sand aquifer		Middle sand aquifer		Upper sand aquifer		Acid leaching solutions		Disposal solutions	RO brine Average
	Average	Range	Average	Range	Average	Range	Average	Range		
Salinity	16,100	9,400–20,000	11,400	10,000–12,900	10,100	10,000–11,000	16,430	15,300–20,000	19,800	23,000
pH	6.9	6.5–9.1	7.0	6.7–8.9	7.5	7.0–8.0	2.2	1.8–2.6	1.8–2.8	6.8
Na	4,310	2,820–5,250	3,385	2,335–3,600	2,809	2,570–3,010	6,170	5,150–7,200	5,600	6,190
K	20.7	18.5–22.5	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	27.5	n.a.
Ca	906	100–1,480	560	480–710	478	440–545	940	810–1,050	1,000	1,040
Mg	390	200–560	270	85–390	260	245–297	210	90–460	430	600
Fe	1.0	<1.0–1.0	1.0	<1.0–1.0	1.0	<1.0–1.0	260	110–370	200	<1
Al	<1.0	<1.0	<1.0	<1.0	<1.0	<1.0	15	0.2–70	28.3	<1
SO ₄	1,754	1,080–2,670	1,540	1,130–1,860	1,445	1,360–1,611	5,300	3,580–6,800	6,110	3,410
Cl	7,850	4,020–9,740	5,370	4,710–6,260	4,800	4,610–5,220	8,470	7,650–9,760	8,020	10,500
HCO ₃	145	90–270	160	28–210	187	179–210	<5	<5	<5	270
Org. C	1.2	1–3	n.a.	n.a.	n.a.	n.a.	2	1–25	n.a.	n.a.
F	0.5	<0.5–1.1	0.5	<0.5–1.8	0.8	0.05–1.2	0.6	0.3–0.9	1.9	1.8
NO ₃	0.3	0.1–1.4	0.9	0.1–1.6	0.1	<0.1–0.1	<0.5	<0.5	<0.5	<0.5
SiO ₂	7.6	4.8–9.8	6.4	4.8–9.4	6.5	5.5–7.4	n.a.	n.a.	101	15
Co	0.060	0.045–0.80	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	2.2	n.a.
Cr	0.02	0.02–0.02	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	0.10	n.a.
Cu	0.02	0.01–0.12	0.03	0.01–0.1	0.008	0.005–0.010	7	1–20	1.8	0.01
Mo	0.013	0.001–0.040	0.010	0.007–0.020	0.009	0.007–0.011	0003	<0.0005–0.010	0.0074	0.022
Ni	0.078	0.060–0.105	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	3.53	n.a.
Se	0.030	0.013–0.075	0.019	0.010–0.030	0.014	0.011–0.022	0.055	0.040–0.070	0.079	0.036
V	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	4	2–8	1.1	<0.02
Zn	0.19	0.03–0.67	0.2	0.02–1.1	0.20	0.03–0.46	110	80–130	56.3	0.09
U ₃ O ₈	1.2	0.007–7.3	0.018	0.001–0.035	0.022	0.007–0.045	75	20–1,000	1.5–3.0	0.16
²²⁶ Ra	205	22–450	7	0.2–17	3.1	0.2–6.6	830	510–1,300	130–680	3.0
²²² Rn	5,000	3,100–6,800	7	6–10	23	15–30	12,700	n.a.	n.a.	n.a.
²¹⁰ Pb	0.8	0.02–3.1	6.6	1.9–17.0	3.6	0.3–6.8	n.a.	n.a.	560	3.8

moon project: (1) the project proposes to re-inject all liquid wastes into the palaeochannel – but still into the lower aquifer, which is known to be hydraulically connected to the Upper aquifer occasionally used by pastoralists; (2) the potential for ‘natural restoration’ is uncertain, although this depends on the reactivity of pyrite (or other reductants) remaining after mining; and (3) the Yarramba palaeochannel is the only groundwater resource in the region (the velocity is about 18 m/year; MINAD 1980).

Pirlo (2000) investigated the potential geochemical reactions in groundwater, leaching and wastewater solutions through modelling of water chemistry. This was based on samples of groundwater from the Honeymoon and East Kalkaroo ore zones plus a sample of wastewater from the Honeymoon trial mine. No field measurement of redox potential was undertaken (this was calculated by the geochemical model REACT using Zn²⁺/ZnS). Although mixing of the various solutions suggested that precipitation effects in the aquifer would be minimal and that heavy metals would not remain mobile after sufficient dilution, Pirlo (2000) acknowledged that kinetic effects are not incorporated in this mixing approach. No directly measured evidence from the trial or the analysis by Pirlo (2000), especially concerning the redox state in the aquifer, demonstrates that natural attenuation has or will work at Honeymoon, with no rates or timeframe established for such processes.

The potential for post-mining impacts on groundwater remain significant, especially if restoration is again not required by government regulators. Final regulatory approval for Honeymoon has been withheld until these and other issues have been adequately addressed by the current proponent.

Discussion and conclusions

The use of acid ISL in the USA was considered problematic and has never been approved or used on a commercial scale, despite the lengthy research at Nine Mile Lake and Reno Ranch, Wyoming. The problems included higher salinity and some radionuclides in post-restoration monitoring of groundwater compared with pre-mining conditions. In Australia, the results from acid ISL trials at Beverley and Honeymoon have never been sufficiently published and thus information is limited on their impacts. Critically, the issues of geochemical conditions in the groundwater following mining have not been satisfactorily addressed, with no clear evidence of natural attenuation at either site. The current configuration of the Beverley and Honeymoon projects – acid leaching with no restoration of polluted groundwater – is more akin to practices in Eastern Europe and the Former Soviet Union,

where the available evidence suggests that natural attenuation fails to reduce the impacts from such mines (Mudd 1998 and 2001). The standards applied at the Australian sites are not considered an acceptable approach for an arid region that is almost entirely dependent on groundwater.

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Critical review of acid in situ leach uranium mining: 2. Soviet Block and Asia

Gavin M. Mudd

Abstract The technique of in situ leach (ISL) uranium mining is well established in the USA, as well as being used extensively in Eastern Europe and the former Soviet Union. The method is being proposed and tested on uranium deposits in Australia, with sulphuric acid chemistry and no restoration of groundwater following mining. ISL mines in the former Soviet Union generally used acid reagents and were operated without due consideration given to environmental protection. At many former mine sites, the extent of groundwater contamination is significant because of high salinity, heavy metal and radionuclide concentrations compared with pre-mining and changes in the hydrogeological regime caused by mining. After the political collapse of the Soviet Union by the early 1990s, most uranium mines were shut down or ordered to be phased out by government policy. Programmes of restoration are now being undertaken but are proving technically difficult and hampered by a lack of adequate financial resources. The history and problems of acid ISL sites in countries of the former Soviet Union and Asia are presented in this study.

Keywords Acid · In situ leach · Solution mining · Uranium plumes

Background

The unconventional mining technique of in situ leach (ISL) is now the primary producer of refined uranium in

the United States, with a market share of around 95% in the mid-1990s (DoE 1999). ISL mines appear set to assume a greater role in Australia's uranium industry. The commercial ISL uranium mines in the USA use alkaline chemistry compared with the proposed projects in Australia, which are based on the use of acid (Mudd 1998). In contrast, the ISL uranium mines of the former Soviet Union and Eastern Europe have primarily used sulphuric acid, with apparently little consideration given to environmental concerns during operation.

The first trials of uranium ISL were both developed in the USA and the Soviet Union in the early 1960s. It is uncertain who developed the concepts or if they were developed separately (Mudd 1998). The use of ISL uranium mining continued to expand until the collapse in the late 1980s. It is worth noting that an unnamed Russian first suggested ISL mining of gold by as early as 1896 (Mineev and Shutov 1979; Morris 1984).

The majority of countries with uranium mining under the influence or control of the former Soviet Union have undertaken ISL projects, although different countries had contrasting success, from an operational perspective. Bulgaria, for example, experienced a major shift in uranium production from conventional to ISL mines, dramatically reducing the workforce and exacerbating already recalcitrant environmental problems.

Since the reunification of Germany and the collapse of the Soviet Union, the extent of the contamination of groundwater is beginning to come to light, mainly through co-operative programmes of the International Atomic Energy Agency (IAEA), although other agencies are becoming involved. There is also increasing interest in the use of ISL methods for the mining of low-grade uranium ores in other parts of Asia, most notably China and Pakistan.

The resurfacing of the Australian acid ISL uranium mine proposals in 1996, the lack of acid ISL mines in the USA, and the research coming to light through the IAEA concerning the extent of impacts from acid mines in the Soviet block, led to a broad-based review of ISL uranium mining by the author, completed in 1998 (Mudd 1998).

The history and experience of acid ISL in countries controlled by the former Soviet Union is reviewed herein, with sites in Australia and the United States reviewed in a companion paper (Mudd 2001). A compilation of leaching solution quality for most ISL mines reviewed in this paper is compiled in Table 1.

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Table 1

Leaching solution and groundwater quality at various Former Soviet Union ISL mines (mg/l, except redox in mV and ²²⁶Ra in Bq/l). *T* Total

	Bulgaria	Stráz Pod Ralskem, Czech Rep.	Königstein, eastern Germany		Former Soviet Union		Bukinaii, Uzbekistan	Yining, China
			Leach	GW	Acid	Alkaline		
TDS	15,000–20,000	50,000–100,000					33–45	
pH	1.4–2.0	0.5	1.9	6.7			1.0–2.0	1.26
Redox								+652
Acid ^a		15,000–38,000						
SO ₄	10–12	33,000–80,000	1,500	4.3	17,000–25,000	2,000–3,000	46,600	22,800
S ²⁻								2.72
CO ₂ ^b								0.99
HCO ₃			5	12		500–2,500		
NH ₄		1,000–2,000				400–600	533	
NO ₃		200–1,400	63	8.4			750	
Cl					400–600	500–1,200	300	762
F		100–300						5.2
P		50–150						
PO ₄							580	22
Ca	140–600	200–300	201	6.6	400–600	700–800	730	17
Mg	140–330	20–30	11.5	1.1	300–500	100–300	2,400	140
Na	30–900	10–15	49	3.9	50–150	200–750	650	172
K	30–200	40–70	3.1	1.5	25–100	25–400	570	28
Al	310–840	4,000–6,000			500–800		2,860	323
Fe ^(T)	700–2,200	500–1,500						
Fe ²⁺					400–1,000		3,300	377
Fe ³⁺					800–1,500		1,500	440
Mn	6–61						100	8.4
SiO ₂	210–350	100–200					560	
As								0.1
Ba							2.05	
Ce							8.35	
Cd								0.04
Co							5.6	
Cr ^(T)		5–15					4.05	0.74 ^c
Cu							1.07	0.5
La							3.15	
Li							6.2	
Mo					10–40			2.4
Nd							4.85	
Ni		20–30					12.3	1.3
Pb							1.13	0.67
Rb							8.0	
Re					0.2–0.5			3.1
Sb								0.4
Sc					0.15–0.6			1.86
Se					50–60			0.001
Sr							8.2	
Ti							3.44	<1
V	1–18	10–15			10–40		2.65	4.8
Y							5.67	
Zn	2.1–7.3						17.5	0.88
²²⁶ Ra	1–2	50–90			3.7			
U	5–30	20–500	46.9	0.009			86	75
Rare earths					10–40			

^aFree acid as H₂SO₄

^bFree CO₂

^cCr⁶⁺ 0.1

Bulgaria

This review is based on Tabakov (1992), Kuzmanov and others (1992), Vapirev and others (1993), Dimitrov and Vapirev (1994), Nedyalkov (1996) and IAEA (1999); more detail is given in Mudd (1998).

The ISL technique was first applied in 1967 to low-grade deposits (ranging from 0.006 to 0.03% U) at Orlov Dol and

Selishte (a former conventional mine). Given the success of these sites, a revolution was perceived whereby many previously uneconomic deposits were exploitable. By 1990, the share of uranium production from ISL had risen to 70%.

All uranium mining and milling in Bulgaria was closed down by government decree on 20 August 1992. Activity since has been aimed at cleaning up and rehabilitating the

numerous mine sites. The total contaminated area as a result of all uranium industrial activity is approximately 20 km², including 6 km² from ISL mining and 4 km² of contaminated forest.

There has been a total of 19 sites where ISL has been applied, and a further 11 sites where the ISL technique was applied within an underground mine. Most of these sites began operation in the late 1960s to early 1970s, although poor results from initial trials meant some sites were discontinued. The mine sites are all concentrated in the southern and western regions of Bulgaria, shown in Fig. 1. The deposits contain high amounts of organic matter, iron and sulphides.

The ISL mines had a dramatic impact on the workforce in conventional mines, falling from 5,000 workers between 1965 and 1970 to approximately 500 in 1988. The initial chemistry used was sulphuric acid, although this was later switched to sodium-carbonate and ammonium-carbonate leaching chemistry in deposits with a high carbonate content.

The environmental assessment, siting and operation of many uranium mining operations across Bulgaria were often neglected to enable fast tracking of projects and minimize the costs involved in establishing a project. Only one mine was closed because of contamination of drinking water. Almost no preventative or counter measures were implemented during the whole period of mining for the environmental protection of water, soil and air from mechanical, chemical and radioactive pollution. The secrecy of the uranium and nuclear industry was identified as a key reason behind this philosophy.

The leaching of uranium was generally progressed in three stages – first, acid was introduced at levels up to 10 g/l (lasting for a few months); second, the base period of acid leaching at 4–6 g/l (lasting for over 2 years); and third, the closeout period at 0.5–1 g/l. The overall time for a well field was between 3 to 5 years and the recovered uranium was about 60 to 80%. The Ra levels of leaching solutions was generally low. A compilation of typical leaching solution chemistry, summarized from 13 ISL mine sites, is shown in Table 1.

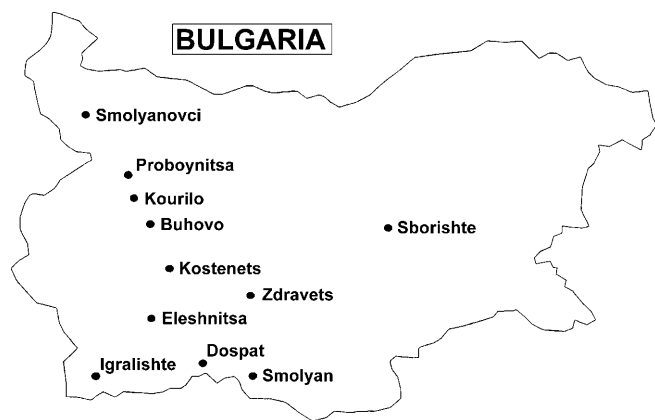


Fig. 1

Location of in situ leach (ISL) uranium mines in Bulgaria (Tabakov 1992)

The pre-mining quality of groundwater in the ore zone aquifers was typically 500–2,000 mg/l TDS, pH of 6.3–8.8, iron 0.1–648 mg/l, and sulphate 24–758 mg/l; which is, good quality groundwater.

There has been significant contamination of groundwater at most ISL sites, with major concerns arising from chemical, radiological and bacterial contamination. For the combined underground-ISL sites, Fe, Cu, Co, Ni, Mo, As and some rare earth elements are several times higher than allowable limits. In the Deveti septemvri ISL mine, Mo reaches 13.4 mg/l (regulatory limit 0.5 mg/l, 27 times higher), and Mn reaches values up to 13 mg/l in groundwater and 4.2 mg/l in the retention pond; quantities of B and Hg have also been detected.

At the Orlov Dol site, after 6 years of monitoring, the acidity of the ore zone aquifer declined from 1,300 to 10 mg/l, although the groundwater still contained elevated levels of uranium despite the associated small increase in pH.

The concentration of sulphate can be very high in surface waters and even in water supply wells of private owners as a result of accidental spilling of solutions at ISL sites. The average chemistry of contaminated groundwaters ranges in TDS from 15–20 g/l, SO₄ is from 10–12 g/l, U is between 5 and 20 mg/l, other salts and the presence of heavy metals and rare earths.

For the Cheshmata (Haskovo) site, in the valley downstream, SO₄ concentration is 1,400 mg/l (limit 300 mg/l), free sulphuric acid 392 mg/l and the pH is 2.2 (over 1,000 times more acidic than the surrounding aquifer). The private wells of residents of the area have also been affected with significantly high concentrations of SO₄ being noted, demonstrating that the leaching solutions have migrated into drinking water supplies.

A similar case has been recorded in Navusen where, in a valley, the SO₄ concentration is 13,362 mg/l and almost 5 g/l of free sulphuric acid, indicating the water is actually a leaching solution. The groundwater quality of such sites has a TDS (salinity) level of greater than 20 g/l, of which SO₄ is 12–15 g/l. Heavy and rare earth elements were detected in some cases, such as V, W, Mo and La, as a result of recycling of the solution.

There were also noted problems because of bacterial contamination, although their exact effects were not able to be predicted and were not studied. It was thought that they were beneficial in the leaching process.

There remains concern that solutions at the various ISL sites could contaminate the deeper groundwater systems, as well as the shallow systems. For the deep systems, which contain the ore being mined, the U content can reach 20–30 mg/l and Ra 1–2 Bq/l, with U content in shallow aquifers of around 3–4 mg/l and Ra 0.5 Bq/l, despite dilution effects during migration through the aquifer.

At some sites, where there were surface spills because of failure of distribution pipes, the U and Ra content of soils is 10 and 2–3 times the background level, respectively.

At most of the ISL sites undergoing restoration, the solutions were continually recycled through the mined aquifer without adding acid, and this led to deposition of

salts within the pipes. These salts contained increased and significant levels of radioactivity. This process has been now been stopped.

A principal problem of the restoration work currently in progress is that the environmental requirements are quite strict, making uranium production unprofitable. It was argued that achieving an acceptable level of environmental protection required that preventative measures, planning and funds are set aside during the early stages of a project, and during the operational phase of a particular project. As this was not done, the necessary funds are not available and restoration work is thus significantly impaired. Only one-third of the land used by ISL operations has been remediated, and as the land will be returned to the original owners for agricultural purposes, there are real concerns for public health and environmental safety. Some of the ISL sites (such as Bolyarovo, Tenevo, Okop and Gorna Trakiiska nizina) are close to areas where potable quality groundwater is extracted by local communities or the groundwater is considered to be an important future water resource.

Czech Republic

This review is based on Benes (1992), Fiedler and Slezák (1992), Khün (1992), Anděl and Pribán (1993, 1994), Tomas (1996) and IAEA (1999); more detail is given in Mudd (1998).

The Bohemian Massif mining district in the Czech Republic, shown in Fig. 2, has been an important source of uranium for the Russian military and nuclear power programmes, with uranium ore from the Jáchymov mine being used to manufacture the first Soviet atomic bomb. A total of more than 100 uranium mines were developed, including shallow investigation mines.

The Stráz Pod Ralskem district consists of sandstone-type uranium deposits, and acid ISL has been applied as the mining method since 1968 after successful trials during 1967. The associated mineralization is also unusual, with zircon, titanium and phosphorous present.

The Stráz region is characterized by complex hydrogeological and biological conditions that are unfavourable to

the application of ISL, primarily related to the slow dissolution rates of uranium. This causes two principal problems – firstly, large doses of chemicals are required (sulphuric acid at about 5% with nitric acid and nitrate as the main oxidants); and secondly the leaching periods are very long, ranging from 15 to 25 years. The total uranium production by ISL was 16,470 t U₃O₈.

The Hamr deposit was developed at the Hamr and Luzice mines with both underground and combined ISL techniques. The Hamr mine is only 5 km from the Stráz mine, exacerbating the technical problems at both sites, which led to higher production costs and greater environmental impacts. A principal problem for many of the sites is the density of population across the Czech Republic, with 40,000 people living near the Stráz mine, for example. After detailed evaluation of the negative impact of uranium mining and milling, a progressive programme of declining production from uranium mining has been adopted and an extensive remediation programme implemented by the Czech Government.

Stráz Pod Ralskem

The hydrogeology of the Stráz region is complex, but can be thought of as two distinct aquifers – the Cenomanian and the Turonian. The Cenomanian is a deep, confined and artesian aquifer, and the Turonian lies above this, separated by up to 100 m of thick low permeability clays and siltstones. A hydrogeological cross section is shown in Fig. 3. The Turonian is designated as an important high quality drinking water reserve with a calcium bicarbonate (CaHCO₃) type of water quality, and is known to discharge to the Ploucnice River at about 40 l/s. The Cenomanian aquifer was known to contain elevated levels of radium. For the Stráz deposit, every tonne of uranium (t U) produced:

- 274 t of sulphuric acid injected;
- 7.9 t of ammonia injected;
- 19.3 t of nitric acid injected;
- 1.79 t of hydrofluoric acid injected;
- 0.95 t of sulphuric acid released to the air;
- 1.18 t of nitrous oxides released to the air;
- 53 GBq of radium released to the air;

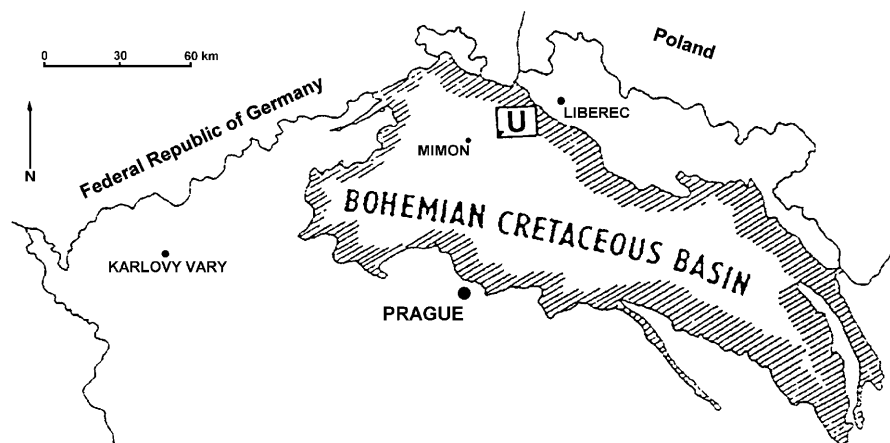


Fig. 2
Location of the Bohemian mining district, Czech Republic (Fiedler and Slezák 1992)

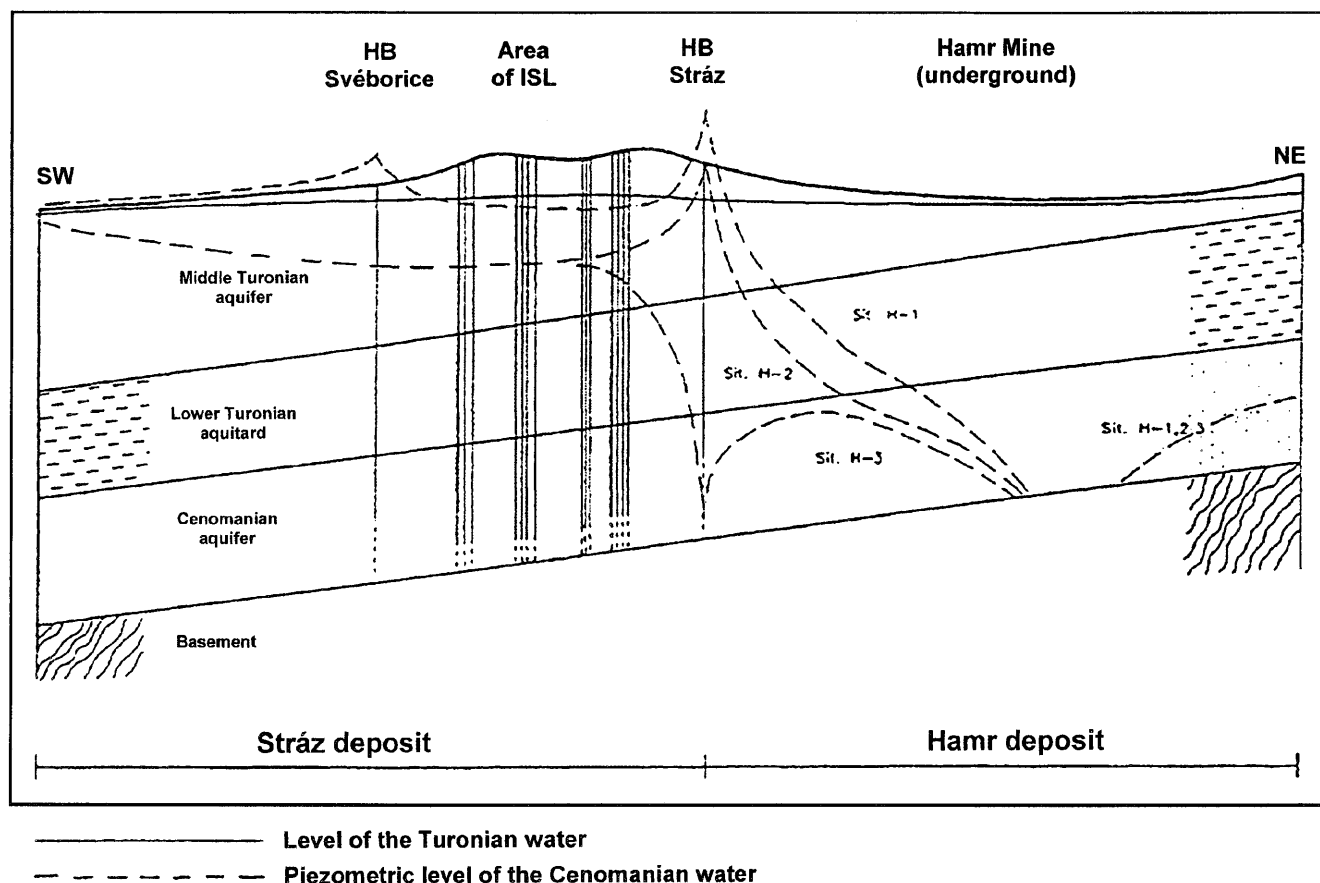


Fig. 3
Hydrogeological cross section of the Stráz Pod Ralskem and Hamr uranium mines, Czech Republic (Fiedler and Slezák 1992)

- 7,260,000 l of contaminated groundwater in the Cenomanian aquifer;
- 1,500,000 l of contaminated groundwater in the Turonian aquifer.

By contrast to experience in the USA, the Stráz ore deposit required 50–70 g/l of sulphuric acid and a leaching period of 15–20 years to reach a yield of 60–80% of the uranium because of the lower permeability of the aquifer materials. By 1994, a total of 32 ISL sites had been commissioned covering a total of 6 km² and consisting of 7,000 wells. The Stráz mining district has seen approximately 3,800,000 t of sulphuric acid, 270,000 t of nitric acid, 103,000 t of ammonia and 25,000 t of hydrofluoric acid injected into the well fields. Typical leaching solution chemistry is shown in Table 1. The interactions between the leaching solutions and aquifer sediments are not well defined, and the speciation of many heavy metals and radionuclides remains unstudied. The Stráz site ceased producing uranium on 1 April 1996.

The leaching solutions from the Stráz well fields were not operated with a bleed system to maintain a cone of depression around active well fields and this led to solutions being dispersed widely through the Cenomanian aquifer in the area, as well as vertically into the Turonian aquifer. The

excursions mainly occurred through production bores, but significant excursions also occurred at liquid waste disposal bores. A location plan of contaminated groundwater at the Stráz and Hamr mines is shown in Fig. 4.

The contaminated water in the Turonian aquifer alone is spread over 245 ha (43% of the area of the well fields). A total of 200 billion l of groundwater has been affected, covering a total area of 6 km² and the volume of aquifer material affected is thought to be 720 billion l. Approximately 50% of the contaminated water is thought to be residual leaching solutions, with sulphate higher than 20 g/l and salinities between 35–70 g/l. The remaining 50% is thought to be dispersed solutions, formed by migrating leaching solutions mixing with native groundwater, with a salinity level of 4.5 g/l. A comparison of leaching solutions, contaminated and pre-mining groundwater quality is given in Table 2. The urgent need for restoration is governed by the extremely high concentrations of radionuclides and heavy metals in the various solutions and the large volumes of contaminated water involved. The most critical factor is that the Cenomanian aquifer is artesian, and the pressure difference between the Turonian and Cenomanian aquifers will always ensure groundwater flow is vertically upwards, as was the case before ISL mining began.

The presence of known excursions through boreholes highlights the above problem, and if the bores are not effectively sealed during restoration, there will remain a pressure gradient for new excursions of contaminated groundwater from the Cenomanian into the Turonian aquifer.

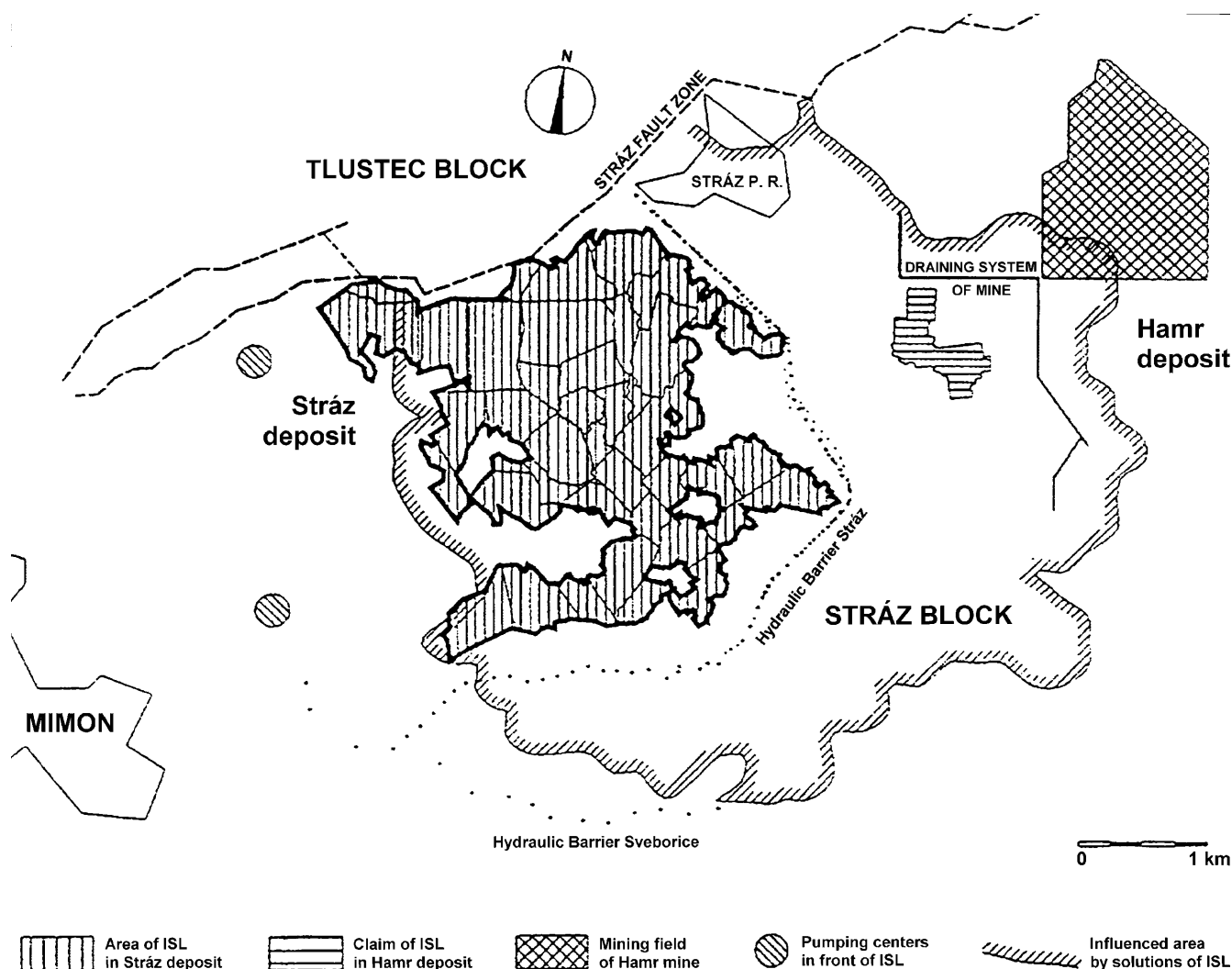


Fig. 4

Location of contaminated groundwater at the Stráž Pod Ralskem and Hamr uranium mines, Czech Republic (Fiedler and Slezák 1992)

The contaminated groundwater in the Cenomanian aquifer is approaching the sanitary protection zone of the Mimon water supply (70 l/s or 6.05 Ml/day). The contaminated groundwater in the Turonian is within 1.2–1.5 km of the sanitary protection zone of the Dolánky water supply

(200 l/s or 17.28 Ml/day). The region presently utilizes 1,500 l/s (129.6 Ml/day) of groundwater for drinking supplies.

The restoration of the groundwater is proving a difficult task, with 1 pore volume of groundwater only removing about 70% of the contaminated groundwater, and 5 pore volumes required for 90% removal. This equates to about 940 billion l of water. While regulators in the USA require a pilot-scale test to demonstrate effective groundwater

Table 2

Groundwater quality before and after ISL mining at Stráž Pod Ralskem

	pH (Units)	TDS (g/l)	SO ₄ (g/l)	NO ₃ (mg/l)	F (mg/l)	U (mg/l)	Ra (Bq/l)	H ₂ SO ₄ (g/l)
Leaching solution	0.5	50–100	33–80	600–1,400	150–250	1–30	50–90	15–20
Cenomanian before	6.7	0.14	0.033	<1	<1	0.02	8.74	n.a.
Cenomanian affected	1.8–2.8	5–20	3.3–13	5–100	5–50	0–15	30–70	0.5–5
Turonian before	6.7	0.1	0.035	5.2	<1	0.01	0.07	n.a.
Turonian affected	2.5–7.0	0.5–5.5	0.05–3.3	5–1,000	0.5–25	<1	0.1–1.0	<0.5

quality restoration prior to issuing commercial permits and operation, the Stráz mine only received approvals for liquid waste disposal and restoration requirements in the mid-1990s after three decades of operation.

Currently, restoration programmes are aimed at determining the optimal strategy for long-term remediation of groundwater quality. The technology being used for restoration involves pre-treatment, reverse osmosis, volume reduction by evaporation, crystallization and processing of the concentrated saline solutions or brines. Some components are re-utilized, such as sulphuric acid (H_2SO_4), aluminium oxide (Al_2O_3), ammonia (NH_3) and gypsum (CaSO_4). The presence and removal of Ra and other metals of concern is a significant barrier to these programmes.

Further options for groundwater quality restoration are being investigated with a view to a compromise between environmental demands and economic feasibility. It is intended to return the Turonian aquifer to its original groundwater quality (as much as possible). Recent modelling studies indicate that restoring the Cenomanian groundwater to a salinity of 3 g/l can cause undesirable impacts on the Turonian aquifer mainly because of structural and tectonic conditions and the instability of the groundwater regime after returning to natural flows.

For the Cenomanian aquifer, though, it appears impossible to achieve restoration to its original good quality. The philosophy being adopted is to ensure that any escape of Cenomanian groundwater will dilute and disperse to an appropriate quality and not impact on potable or surface waters.

The region, once covered by pine forests, underwent deforestation for mining purposes. This was undertaken hastily with many trunks left in the ground, dead and rotting. The surface soils, devoid of tree cover, were therefore exposed to accelerated rates of weathering, sheet erosion and wash-down of the poorly cohesive sandstones and deep furrows. This was exacerbated by the movement of heavy machinery across the site. In the low-lying areas near the Ploucnice River, the alteration of surface drainage patterns, together with the removal of vegetation, led to a gradual rise in the water table and the formation of lagoons and wetlands.

In the areas where pine trees had been left, to try and preserve some of the remnant forest, it was found that the forest was weakened and unsustainable since it was no longer continuous. This led to increased exposures of the wells and piping systems and high incidences of dead, falling trees. In the hill areas, wells and piping systems were often built partly on benches and partly on platforms. Together with the spills of solutions from pipes and surface runoff, the siting of these parallel to slopes led to significant rates of erosion and the prevention of further vegetation growth because of the lack of suitable soil. Attempts were made from the mid-1980s to address these problems, such as hydromulching, different seed species and other techniques, but they were of varying short duration and thus limited success.

Because of the intransigence of the chemical and physical changes caused by ISL mining at the Stráz site, the

restoration efforts are anticipated to last several decades, or even centuries.

Germany

This review is based on Hähne and Altmann (1992), Diehl (1995), Ettenhuber (1996), Nitzsche and Merkel (1999) and Biehler and Falck (1999).

The pitchblende uranium deposits of former East Germany were one of the former Soviet Union's first targets for supplying uranium during the late 1940s and 1950s for nuclear weapons and power programmes. Simultaneous work was carried out on all types of uranium deposits in the area. Most uranium mines and mills in East Germany were underground, although Königstein and to a lesser extent Ronneburg, also had underground ISL applied for the extraction of uranium. Underground uranium leaching began in 1968 to take advantage of low-grade ores and increasing conventional costs because in situ leaching costs were only 60–70% of conventional methods. The locations of Ronneburg and Königstein are shown in Fig. 5.

Ronneburg

Although a less prolific producer of uranium by ISL, repeated attempts were made to increase uranium recovery using ISL from 1970. The leaching solutions used included sulphuric acid or alkaline reagents. Between 40 and 70% of the uranium reserves were recovered at concentrations of sulphuric acid from 3–10 g/l, pH of approximately 1.5–2.5 and uranium content between 20–100 mg/l. A total of 3,203 t of U_3O_8 was produced by heap and waste pile leaching, with 106 t of U_3O_8 produced by underground ISL techniques.

Königstein

From 1971, underground and ISL mining (in stopes) were being used until ISL took over in 1984. The extraction of uranium with ISL operated until 1990, with the total uranium production from the life of the mine being 22,711 t of U_3O_8 , 6,517 t by sulphuric acid ISL.

The aquifers at Königstein generally contain water quality with a salinity less than 200 mg/l, near neutral pH and low heavy metal concentrations. The uranium mineralization at Königstein is found within the fourth aquifer of a regional groundwater system, shown in Fig. 6. The clay layer separating the third and fourth aquifers was intersected by the underground mine workings. The third aquifer is used by residents of the region for their water supply, as well discharging into the Elbe River 600 m east of the mine site. The dewatering of the fourth aquifer for the mine led to a decrease in water level of the third aquifer.

One of the most difficult problems associated with remediation of the contaminated groundwater is that, at the time of closure, a new underground block had just been prepared for leaching. The prevailing unsaturated condi-

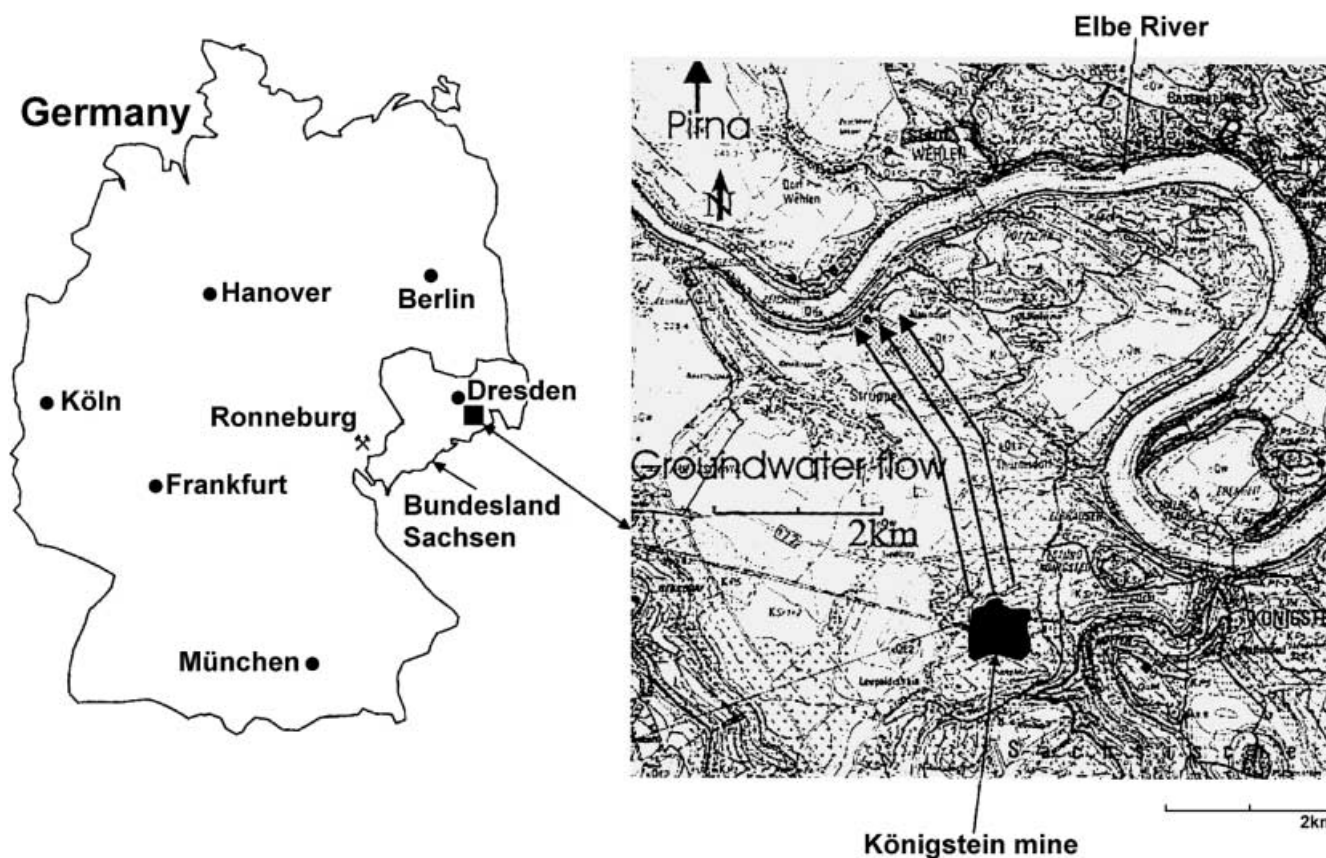


Fig. 5
Location of the Ronneburg and Königstein uranium mines, eastern Germany (Nitzsche and Merkel 1999)

tions allowed the pyrite to oxidize, generating significant quantities of sulphuric acid, which further mobilized heavy metals, uranium and radionuclides and added to the contaminant load to remediate.

The average concentration of leaching solutions was 2–3 g/l sulphuric acid, pH 1.5–1.8, salinity (TDS) 10–14 g/l and uranium 10–150 mg/l. A compilation of groundwater and leaching solution quality is given in Table 1. The uranium recovery was generally about 65–75% within 3 years. A total of 100,000 t of sulphuric acid was injected into the mine. The leaching process has chemically affected a volume of more than 55 billion l of rock and aquifer, while approximately 1.8 billion l contains 1.2–1.7 g/l sulphuric acid and more than 30 mg/l uranium remains circulating or trapped in the pore space of the rocks. A further 850×10^6 l are circulating between the leaching zone and the recovery plant. Expressed as multiples of applicable German drinking water standards, the trapped solutions are 400 times higher in Cd, 280 times higher in As, 130 times higher in Ni and 83 times higher in U.

The principal concerns for restoration of the site are centred around the flooding of the underground mine workings that will occur after the mine is closed down. There is potential for contamination of surrounding groundwater and surface water streams with U, Ra, SO_4 , Fe and heavy metals. Although small-scale flooding trials are currently being conducted, restoration is still not complete

and the mine still represents a threat to the surrounding aquifer, an important potable groundwater resource for the region.

Former Soviet Union

A general overview of the use of ISL in the former Soviet Union is presented. A more detailed review of each new republic (the 'stans'), however, will be treated separately. This review is based on Skorovarov and others (1987) and Skorovarov and Fazlullin (1992).

There has been active development of ISL-type mines across the former Soviet block countries such as the Ukraine, Uzbekistan, Kazakhstan and Russia since the early 1960s, beginning with tests at the Deviadovskoye site in the Ukraine. After the success of initial tests, ISL technology was rapidly adopted throughout the various territories. The ISL technique was typically applied to low-grade deposits between 0.03–0.05% U. Sulphuric acid was the preferred leaching agent, although alkaline carbonate-bicarbonate agents were used at some sites, depending mainly on the carbonate content of the ore. The well field patterns used were quite variable, including 10×10 m, 10×20 m, 25×50 m and up to 10×100 m spacings. The concentration of sulphuric acid ranged from 2–5 g/l, with the stronger the acidity the greater the recovery of uranium and the shorter the period of leaching. The typical composition of leaching solutions is given in Table 1. The average acid consumption per 1 kg of

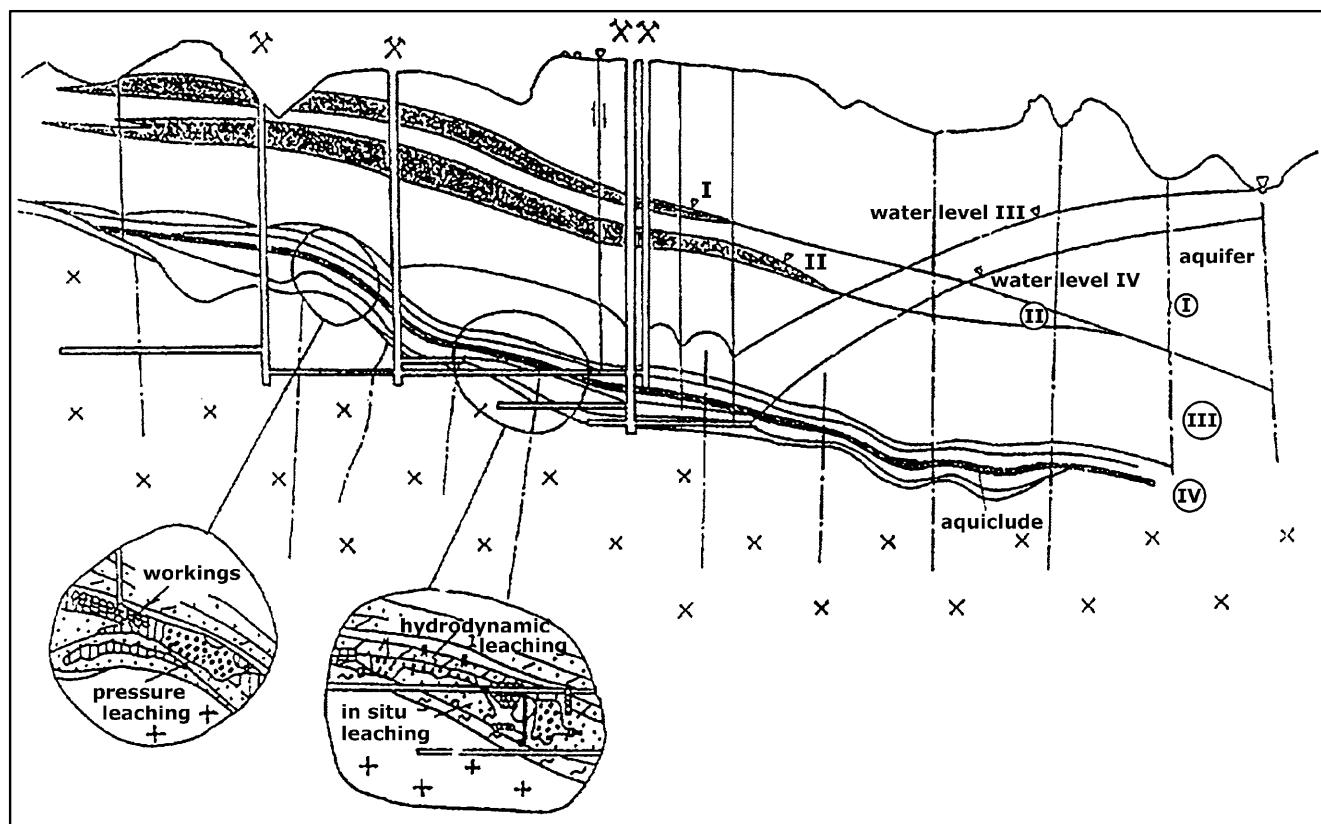


Fig. 6

Hydrogeological cross section of the Königstein uranium mine, eastern Germany (Hähne and Altmann 1992)

uranium recovered as an end product varied widely from 18–150 kg. The recovery rate of uranium was generally between 70 and 90%. No oxidant was needed to ensure dissolution of the uranium. Associated metals were also thought to be extractable, such as V, Re, Se, Mo, Sc, Yt and rare earths.

Some of the main problems of using sulphuric acid was the necessity to use acid-resistant materials and equipment, deterioration of the ore zone permeability because of chemical and gaseous plugging, and the very high salinity levels during mining (ranging from 15–25 g/l).

For ores with a carbonate content higher than 1.5–2.5%, alkaline solutions were used consisting of ammonium bicarbonate with concentrations generally varying from 500 to 5,000 mg/l. Sodium bicarbonate was also applied at some sites with oxygen as an oxidant. The recovery of uranium was generally between 50 and 60%. The use of alkaline agents also tended to show much smaller increases of salinity during mining. The main recognized problems of alkaline ISL were the high degree of solutions escaping outside the mining zone (often caused by gaseous oxygen plugs forming), compulsory pre-treatment to soften the water and restoration difficulties following completion of ISL.

Numerous techniques were being trialled to restore the quality of the groundwater, including lime pulp treatment, hyperfiltration and electrosorption. The success of these

technologies on restoring contaminated groundwater is not known. The production costs of ISL were 40–45% of conventional costs, with significantly lower energy and capital costs and reagent consumption.

Kazakhstan

This review is based on Yazikov (1993), Akin (1995), Catchpole (1997), Carroll (1997) and OECD (2000). The uranium resources of Kazakhstan are considerable and second only to Australia, of which a large proportion is amenable to ISL extraction (OECD 2000). The location of conventional and ISL uranium mines is shown in Fig. 7. As with many former Soviet-controlled states, the use of ISL in Kazakhstan began in 1970 (after successful testing in the Ukraine) and continued to increase in importance, centred around the large amenable deposits in southern Kazakhstan. The large-scale ISL mines began in 1978. All ISL mines utilize sulphuric-acid-leaching chemistry. By 1990, ISL technology had displaced conventional mines as the predominant method of uranium production. The large ISL-amenable resources are seen as the future of the Kazakhstan uranium industry. There are several operating ISL projects, although an accurate assessment of current and prospective projects is not an easy task. Some operating sites include Stepnoye, Centralia and Chiili. Further sites being assessed and/or developed are the Inkai, Mynkuduk and Moynkum sites. It is known that following acid ISL mining, the aquifer is left to return to its pre-mining state, although timeframes

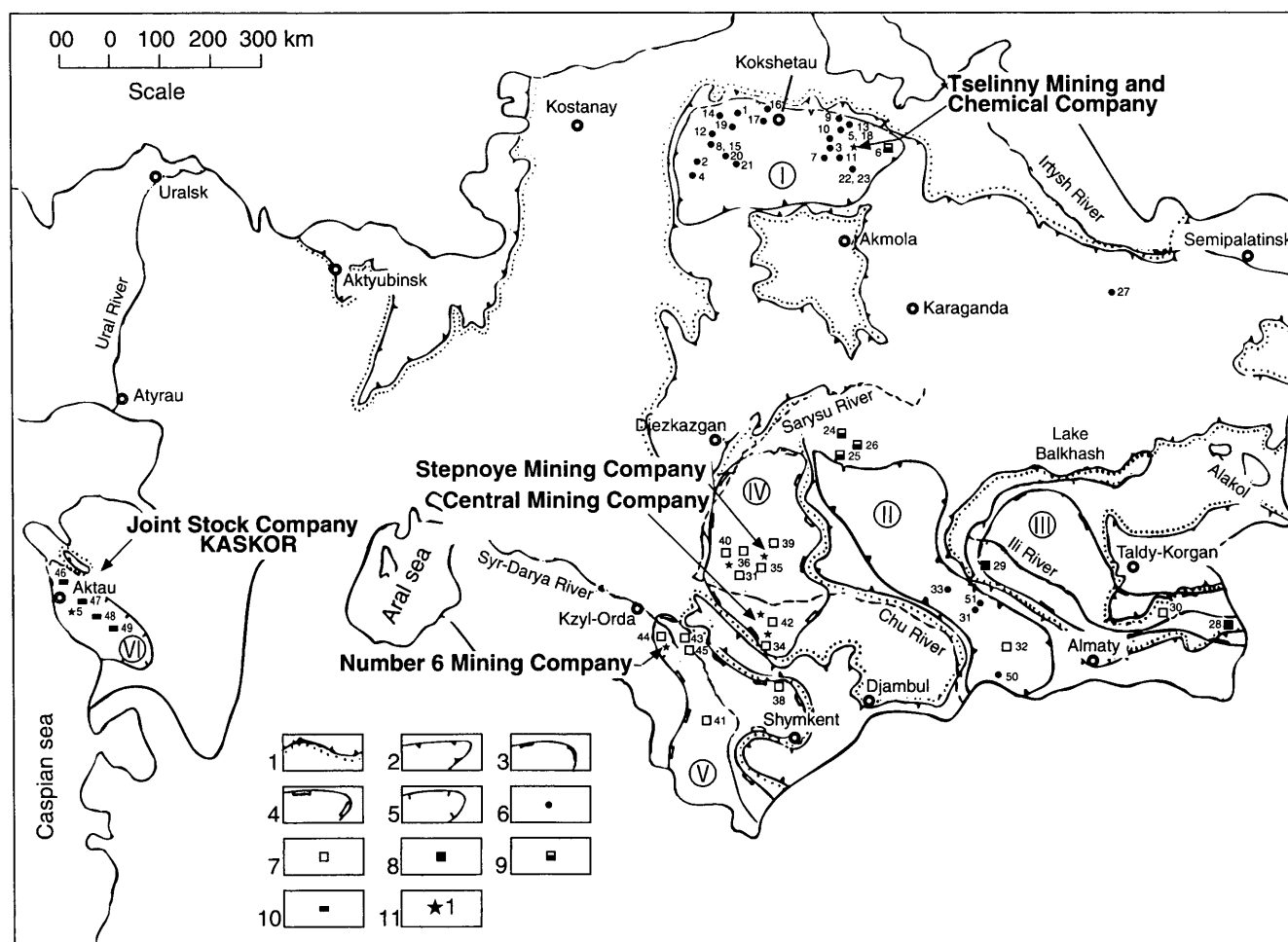


Fig. 7
Location of uranium mines and ISL projects in Kazakhstan (OECD 2000)

are not yet established for this process nor its rate of success. To avoid possible negative impacts from residual mining solutions, an exclusion zone of 150 by 15 km has been declared from which the extraction of drinking water is prohibited. Detailed assessment of the environmental impacts and operational issues are yet to be published, although given the history of the nuclear industry in Kazakhstan, it is likely to be similar to other parts of the former Soviet Union.

Ukraine

This review is based on Molchanov and others (1995), Rudy (1996) and Chernov (1998). ISL uranium mining was carried out on the Deviadovskoye, Bratskoye and Safonovka deposits.

The Deviadovskoye site was the first ISL mine in the Soviet Union and operated from 1966 to 1983, using sulphuric and nitric acids. The surface area of the mine is 0.12 km²,

with a ore body of about 2.18 km² and an area for underground storage of 1.2 km². As a result of ISL mining, groundwater was contaminated to a depth of 80 m. The residual solutions are distributed at a distance of 1.7 km along the flow path and for 0.35 km against the upstream hydraulic gradient. The nearest settlement down gradient is 4 km. The volume of residual solutions after the ISL mining of uranium in the Buchak aquifer is 7.09 billion l. The volume of retention pond water is 1 billion l with contaminated silt in pond-collectors of about 40×10⁶ l. Leakage from pipelines has contaminated surface soils, totalling about 50×10⁶ l.

The Bratskoye ISL mine site operated from 1971–1984, using sulphuric and nitric acids. The orebody area is 0.96 km². At the end of mining, the leaching solutions within the orebody were simply abandoned. The 5.2 billion l of contaminated groundwater is distributed 3 km down-gradient and 1.2 km up-gradient, to a depth of 50 m. The Safonovka ISL site was mined from 1982–1993. The surface area of the mine site covered 0.05 km², although no further information is available on the extent of groundwater impacts and future management and remedial programmes.

The severe lack of financial resources has led to the freezing of restoration activities in 1996.

Uzbekistan

This review is based on Venatovskij (1992) and Solodov (1998).

The state of Uzbekistan has numerous uranium deposits that host operating or potential ISL mines, concentrated within the large central Kizilkum province. They are generally around 300 m in depth and contain uranium ore in several distinct layers. The ore grades vary from 0.03–0.70% U. Many contain low carbonate content less than 2.5%, although some deposits are rich in carbonaceous matter (higher than 5%). Various leaching agents are used, with sulphuric acid being the preferred acid. A large ISL mine operated at Bukinaii in south-eastern Uzbekistan for about 20 years. The host sands for the roll-front uranium mineralization contain calcite up to 2% and pyrite at 0.2–1.8%. The redox potential in oxidized zones is typically about +150 mV, in the ore zone it varies from –150 to +100 mV and can be as low as –330 mV in more highly reduced sands. The leach solution quality is given in Table 1 and compares with the pre-mining salinity (TDS) of 1,700 mg/l and near-neutral pH of 6.3–6.7.

Six years after ISL mining had been completed on the Bukinaii deposit, a series of field and laboratory investigations were undertaken to assess the migration of the high salinity brine (leaching solutions) abandoned after mining. Field and laboratory testing showed that pH and redox barriers develop as the brine migrates, similar to the natural geological processes that lead to the formation of roll-front mineralization, based on the reactivity of calcite in the aquifer sediments, redox reactions and helped by bacterial activity. Precipitation tended to be in localized zones of thin, fine-grained sands, with a permeability decrease of about one order of magnitude. Medium- and coarse-grained sands only showed limited permeability reductions as a result of precipitation reactions. It was estimated that, assuming an ideal brine lens of 400 m width, it would take about 80 years to achieve complete attenuation and the resulting area would be about half the width of the initial lens (~200 m). No field or laboratory data are presented to demonstrate that this attenuation process returns the levels of all heavy metals and radionuclides to the low levels prior to mining. Importantly, research was still inconclusive on the possibility of redissolution of secondary phases and the remobilization of various heavy metals and radionuclides.

China

This review is based on Jian and Ning (1992), Xu and others (1998) and (OECD 2000).

Approximately 61% of Chinese uranium is contained within deposits smaller than 3,000 t U_3O_8 , mostly below a grade of 0.2% U. The ISL technique, for underground mines and the more traditional solution mines, has been viewed as the preferential method for economically extracting uranium since the early 1980s with trials on all

types of mineralisation being conducted. Two large ISL projects are currently being actively developed or operated at Tengchong and Yining.

The Tengchong uranium deposit is hosted in sandstone, with gangue minerals including pyrite and carbonaceous matter. A trial of sulphuric acid ISL lasting 42 days was undertaken on a pattern of 31 wells, with the uranium content reaching a maximum of 150 mg/l and an effective yield of 62%. The deposit is being expanded to a commercial facility.

ISL mining at Yining, also known as deposit no. 512, began in 1994 using sulphuric acid and hydrogen peroxide as an oxidant. The deposit is hosted in sandstone being up to 20 m thick and is 0.011–0.17% U in grade. Sulphuric acid levels were initially injected at 2% (20 g/l) and gradually increased to 8% (80 g/l) with hydrogen peroxide concentrations of up to 0.55 g/l. The acid was later reduced to 4–6 g/l. The ISL trial ran for 92 days, with the injection of 9.8 million l of leaching solution, 41.59 t of sulphuric acid, 2.11 t of hydrogen peroxide, and the extraction of 11.7 million l of solutions (18.7% bleed rate) with uranium averaging 300 mg/l. A compilation of leaching solution quality is given in Table 1. No data appear to be available on restoration or if any such activities were undertaken.

Discussion and conclusions

The experience of acid ISL uranium mining in areas controlled by the former Soviet Union provides a stark contrast to experiences in America and Australia. In most applications of the technique, there have been extreme occurrences of groundwater contamination. At some sites, this contamination has migrated considerable distances to impact on potable drinking water supplies. For other sites, the potential for contamination to reach an undesirable receptor remains significant. The problems at these sites were severely exacerbated by the prevailing paradigm of uranium production without regard for environmental damage. Apart from Asian nations, such as Kazakhstan, Uzbekistan and China, all countries are in the process of closing down and developing remedial action programmes. The restoration of groundwater is proving difficult, both technically but also because of a lack of financial resources within these countries.

Morris (1984) noted that reliance on natural attenuation processes has never been tested for restoration of ISL. The former ISL sites across Eastern Europe and the former Soviet Union allow some insight into the use of 'natural restoration' as a remedial technique.

Buma (1979) argued that natural geochemical processes within aquifers can restore contaminated groundwater from ISL mines, thereby saving valuable chemical, energy and financial resources. The processes include precipitation of reduced compounds; scavenging of heavy metals by pyrite, organic matter, ferric oxyhydroxides and calcite; and adsorption by quartz, feldspars and clays. The key was for active reductants to be present.

At many ISL mines outlined above, there was high organic, iron, carbonate or sulphide content, such as Bulgaria and the Czech Republic. The contamination at these sites, including the high concentrations of major ions, heavy metals and radionuclides, has not attenuated significantly over time, and instead migrates away from the mine sites, up to several kilometres in some instances. The geochemical mechanisms controlling this migration are unclear, although co-precipitation, which may give rise to higher solubilities for species such as $(\text{Ca,Ra})\text{SO}_4$, and the complete oxidation of reducing agents during ISL mining with no active agents remaining after mining or coating of possible reactive surfaces with precipitates during or after mining, are likely to be significant, key issues. The extent and ease of remobilization of many elements following migration from the mined aquifer is also poorly understood.

It would appear, therefore, that 'natural restoration' is not a desirable approach in the slightest, even given the complex hydrogeochemical conditions known to exist at most former (and current) ISL sites across Eastern Europe and the former Soviet Union. It would be a more conservative approach to require and provide funds for restoration prior to the commencement of mining, with detailed studies of aquifer geochemistry and physical behaviour to establish likely requirements and potential remedial strategies.

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The Legacy of Early Uranium Efforts in Australia, 1906–1945: From Radium Hill to the Atomic Bomb and Today

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The existence of uranium minerals has been documented in Australia since the late nineteenth century, and uranium-bearing ores were discovered near Olary ('Radium Hill') and in the Gammon Ranges (Mount Painter) in north-eastern South Australia early in the twentieth century. This occurred shortly after the discovery of radioactivity and the isolation of radium, and a mining rush for radium quickly began. At Radium Hill, ore was mined and concentrated on site before being transported to Woolwich in Sydney, where the radium and uranium were extracted and refined. At Mount Painter, the richness of the ore allowed direct export overseas. The fledgling Australian radium industry encountered many difficulties, with the scale of operations generally much smaller than at overseas counterparts. Remoteness, difficulties in treating the ore, lack of reliable water supplies and labour shortages all characterized the various attempts at exploitation over a period of about 25 years to the early 1930s. Hope in the potential of the industry, however, was eternal. When the British were working with the Americans during the Second World War to develop the atomic bomb, they secretly requested Australia to undertake urgent and extensive studies into the potential supply of uranium. This led to no exports but it did lay the groundwork for Australia's post-war uranium industry that has dominated the nation's nuclear diplomacy ever since. Some three decades later, the modest quantity of radioactive waste remaining at Woolwich was rediscovered, creating a difficult urban radioactive waste dilemma. The history of both the pre-war radium–uranium industry and Australia's involvement in the war-time exploration work is reviewed, as well as the radioactive waste problems resulting from these efforts, which, despite their relatively small scale, persist and present challenges in more modern times.

Introduction

The Discovery of Uranium, Radioactivity and Radium

The element uranium is the heaviest naturally occurring element. It was first identified by the German chemist Martin Heinrich Klaproth in 1789 in mineral samples from the Erzgebirge ('Ore Mountains') in eastern Germany and Joachimstal in Bohemia (today's Czech Republic).¹ Although it was mainly of scientific curiosity for many decades, uses found for uranium included as a colourful glaze in ceramics and glassware, as a potent poison, and in alleged medical treatments (e.g. 'uranium wine') for diabetes, stomach ulcers and consumption.² The mining of uranium ores began to attract a limited commercial interest.

In early 1896, the French physicist Antoine-Henri Becquerel discovered the phenomenon of radioactivity in uranium minerals. Soon afterwards, his Polish assistant Marya Skłodowska (soon to become famous as Marie Curie) and her French husband Pierre Curie between 1898 and 1902 isolated the main sources of the radioactivity as the new elements polonium and radium.³ The more intense radioactivity from radium was quickly seen as a potential aid in the treatment of cancer and interest multiplied in uranium ores as a source of radium.⁴ Throughout Europe and North America, researchers raced to find the nature of 'radioactivity' and in the process gained new insights into the structure of the atom.

The primary demands for the limited radium supply were for scientific research

and medical use, principally cancer treatment. Within a short time radium was attracting the staggering price of £5000 (US\$120,000) per gram.⁵ New deposits were soon discovered and techniques developed to extract uranium from older mines. The radium came mainly from the well-known Eastern European uranium deposits as well as increasingly from the USA, until the Belgian company Union Minière announced in 1922 its discovery of rich uranium ore at Katanga in the Belgian Congo, from which point the Belgians dominated the global radium market.⁶ By this stage radium could be obtained more easily but still at a cost of some £10,000 per gram. The Australian Government ordered 10 grams of radium from the Belgians for cancer treatment in September 1927, at a cost of £100,000.⁷ Rich uranium ores were discovered at Great Bear Lake in the Canadian Arctic in 1933, finally breaking the Belgian monopoly but again damaging dreams for an Australian radium industry.

Rapid progress in the scientific understanding of radioactivity and nuclear or ‘atomic’ physics continued in the 1930s. The year 1932 has been seen as an ‘annus mirabilis’ for nuclear physics — the neutron was discovered, atoms were ‘split’ using a new device called an accelerator, and the existence of deuterium was finally proved.⁸ In 1934 tritium was discovered⁹ and uranium was first made to undergo fission, though this was not recognized at the time.¹⁰ In 1938 and 1939 experiments using neutrons and uranium were undertaken in numerous laboratories. By the outbreak of the Second World War in September 1939, the concept of a fission chain reaction and the theoretical potential for an ‘atomic bomb’ were clearly pictured by leading nuclear physicists around the world, though significant uncertainty remained as to the extent of the further research that would be needed and the time it would take, the likelihood of eventual success, and even the

practicality of extracting energy from the nuclear fission process.¹¹

It is against this backdrop in medical demand, fundamental science, nuclear physics and the eventual emergence of a strategic imperative that Australia attempted to carve a niche.

The First Discoveries of Uranium in Australia

The Australian continent has long been recognized to be endowed with rich mineral deposits. The South Australian Government Geologist, Henry Y. L. Brown, stated in 1903 that prospectors should not waste their time searching for radium due to the extremely low concentrations in which it was found in uranium ores (‘one grain per ton’ or 0.065 parts per million).¹² Others, however, were more hopeful. With uncanny foresight, the following comments on the discovery of uranium in mines in Cornwall, in England, were printed in the *Mount Barker Courier* on 8 November 1889:

It has often occurred to me that scarcely sufficient attention is given to the collection of a variety of minerals in the colony. South Australia is so rich in all common minerals of commerce that anything not belonging to that category and not appearing to the uneducated eye to be of special value, is apt to be passed by as worthless. Why may there not be uranium in the colony?¹³

The first confirmation of uranium minerals in Australia is generally accepted as being at Carcoar, New South Wales, in 1894, published by George W. Card in 1896 (Fig. 1).¹⁴ There had been an unconfirmed report of uranium in a small chromium deposit in South Australia in 1890 by Captain Stevens of the Mount Rhine Silver Mining Company,¹⁵ and there was a further unconfirmed report of uranium minerals in the Flinders Ranges, South Australia, around Nickols Nob and Mount Ogilvie, in 1896.¹⁶ Even earlier, in the Northern Territory, the 1869 Darwin survey teams under South Australia’s

Surveyor General, George Woodroffe Goyder, noted an unidentified strange green mineral in association with malachite (a copper carbonate mineral) at Rum Jungle, about 65 km south of Darwin. In 1912, the Northern Territory's Government Geologist, Harald Jensen, reported the existence of uranium at Rum Jungle, but due to the lack of interest at the time did not continue the work to confirm the find.¹⁷ Curiously, when the Mary Kathleen uranium deposit was discovered in July 1954, it was claimed by one of its discoverers that the prospect may have been first noted in 1914 or 1915, during working of a shallow copper mine in the area.¹⁸

After the discovery of radioactivity and of radium, radioactivity in several Australian minerals was studied by Douglas Mawson and Thomas H. Laby at the University of Sydney.¹⁹ This work confirmed the presence of torbenite at Carcoar, euxonite at the Marble Bar tin fields and gadonilite at the Cooglegong River–Greenbushes tinfield in Western Australia, and radioactive monazite²⁰ in Western Australia's Pilbara region, at Tumberumba, Tooloon River, Broken Head/Richmond River, Torrington and Emmaville, New South Wales, and in Tasmania.

The pace of discovery was largely scientific until radium and uranium made



Figure 1. Locality map of Australian radium mining and milling.

public news in early May 1906. A prospector, Arthur John Smith, was working in the remote north-east of South Australia near Olary, close to the New South Wales border and Broken Hill. Smith had discovered what he hoped was tin ore in March 1906 and sent specimens to the South Australian School of Mines for analysis.²¹ The analyst, Walter S. Chapman, recognized a coating of carnotite and some gummite on a darker mineral that Mawson, now at the University of Adelaide, later identified as a new mineral he named davidite.²² Carnotite is a mixed oxide mineral of uranium and vanadium while gummite is a weathering product of pitchblende (uranium oxide), and both were fetching high prices on the world market at the time (for high ore grades).²³ Smith's discovery, made public on 3 May 1906,²⁴ was the first confirmed find in Australia of potentially economic uranium-bearing ore — and therefore of a potential source of the highly prized radium. Mawson's work helped cement his growing reputation as a pioneer in radioactive minerals.²⁵

The finding of uranium created immediate and intense scientific interest, with lively debates at the University of Adelaide, the South Australian Department of Mines, the Royal Society of South Australia and elsewhere in Australia. South Australia's Government Geologist, Henry Y. L. Brown, visited the Radium Hill site on 3 May 1906, while Henry Gilbert Stokes from the Queensland Museum visited the site and through the press on 9 May 1906 stated publicly his doubt that the uranium was present as carnotite and his opinion that the site was of 'no commercial value'.²⁶ Many hoped that a commercial industry could soon prosper, and bulk samples were sent to Marie Curie in Paris and the Imperial Institute in London, as well as a minor quantity to the USA for research and promotion.²⁷

The new uranium deposit was worked until 1908 by Smith, who had sunk an

exploration shaft some 21 m, shown in Plate 1. The site was popularly known as 'Smith's Carnotite Mine', with Smith even awarded a 'Diploma for Gold Medal' by the Franco-British Exhibition of 1908.²⁸ After Smith pegged the area, 'he came to Mawson with an offer: half a share in whatever might develop in return for Mawson's footing all expenses and attending to investigation and exploitation of the find. Mawson agreed, subsequently identifying and naming the primary mineral there as davidite but finding that as a source of radium it was at that time uneconomical. The lease expired.'²⁹ The mine became popularly known as 'Radium Hill'.

At the time of the Radium Hill discovery, the assistant chemist at the Moonta copper mines northwest of Adelaide, S. Radcliff,³⁰ announced that he too had identified uranium minerals, in the rich copper ores at Moonta, but had been awaiting further test results before going public. Radcliff initiated his search in June 1905 and had apparently confirmed radioactivity and uranium mineralization in some ore zones by March 1906, possibly as early as October 1905. The uraniferous samples were tested in March 1906 at the University of Adelaide and often contained around 5% U_3O_8 and up to 10% U_3O_8 . The mineralization was in small localized pockets, however, and mainly of mineralogical interest.³¹

A short time later Mawson was again to be associated with the discovery of a new uranium field in South Australia. The Gammon Ranges in the northern Flinders Ranges were roamed by a tenacious prospector by the name of William Bentley Greenwood ('Dolomite Bill') and his sons. In 1910 he sent some mineral samples, discovered by son Gordon ('Smiler'), to the South Australian Department of Mines for analysis where they remained unidentified (except for a trace of copper) and almost forgotten for several months.³² In anger Greenwood took his samples to Douglas

Mawson, who at first failed to identify the odd minerals in the rock samples but eventually discovered the bright green mica-like mineral to be torbenite. The Mount Painter uranium field was thus revealed.³³ Curiously, at the time of Radium Hill's original publicity, Greenwood claimed that earlier samples from the Mount Painter area that he had collected in 1898 and 1899 had been thrown out by the South Australian Department of Mines without being examined.³⁴ Mawson moved quickly and sent one of his most promising geology students, Arthur ('Archie') Broughton, into the field to investigate. Broughton, discovering grades of 12.5% to 41% U_3O_8 , quickly proclaimed that 'this will be the richest mine of its kind in the world'.³⁵

Australian Efforts in Global Context

The presence of uranium deposits had now been established in Australia beyond mere speculation and mineralogical curiosity. There were intermittent but determined efforts to mine the ores over three decades but numerous factors combined to make the mines falter, including transport problems, labour shortages, lack of potable water, ore treatment, financing and marketing challenges. Hope remained, however, with the South Australian and Commonwealth Governments regularly promoting the Radium Hill and Mount Painter prospects at international exhibitions and congresses. Following the breakthroughs in nuclear physics in the late 1930s, the potential strategic importance of Radium Hill and Mount Painter seemed to open new opportunities.

A history of radium–uranium mining at Radium Hill and Mount Painter is now presented, focusing on the challenges encountered and the moderate radioactive waste problem that still lingers from this work. This will be followed by an account of the minor but determined attempts to exploit these uranium deposits for the British — thus documenting Australia's

little-known but keen contribution in the development of the atomic bomb. This laid a foundation for the following decades of uranium mining and thus for a key plank in Australia's nuclear diplomacy in the post-war world.

Radium Hill

The promise of commercial radium–uranium mining at Radium Hill was quickly realised by many in the Adelaide scientific and mining community, highlighted by the wide interest displayed in 1906.

The initial assays through the South Australian Department of Mines returned results of 0.28% U_3O_8 , and importantly, the secondary or weathered nature of the carnotite was recognized as suggesting that a larger body of primary or unweathered ore lay at depth below the site.³⁶ Further analyses were subsequently performed in Adelaide by Mawson, and in London, as well as in Paris at Marie Curie's laboratory. All tests confirmed the low-grade and generally uneconomic nature of the ore, as well as the difficulty in treatment to extract the radium and uranium from the titanium-rich ore.

Despite the poor test results, there was a 'flurry of activity on the Stock Exchange'.³⁷ A few companies were floated and activities slowly began to get under way at Radium Hill, including the sinking of shafts for exploration and mining, ore treatment research and testing, and marketing of the refined radium (uranium was a 'co-incidental' by-product in this work). The major company active on site at this time was the Radium Hill Company, formed in June 1909 to take over Smith's work, with other smaller but unsuccessful companies also active for brief periods of time.³⁸

By September 1911 some 800 tonnes of ore were at the surface and 9000 tonnes were within sight of being mined, and the price for refined radium bromide had reached a staggering £13,000 per gram.³⁹

Around this time Smith unsuccessfully sought a government reward of £1000.⁴⁰

The milling and radium refining process was developed over twelve months through research on 44 tonnes of ore at the Bairnsdale School of Mines in Victoria, published by Radcliff in 1913.⁴¹ The process was claimed to be relatively simple and allowed processing of the ore to try to compete with overseas radium projects. The work on site at Radium Hill continued, extending shafts and mining ore for beneficiation at a small on-site magnetic mill to pre-concentrate the ore. This mill led to some 30% of the ore being concentrated for further processing.⁴² The concentrate was then transported to a newly constructed radium refinery at Woolwich in suburban Hunters Hill⁴³ in Sydney, which cost more than £15,000 to develop and had a capacity to process about 10 tonnes of concentrates per week.⁴⁴

The Woolwich radium refinery operated from June 1911 to June 1915 (see Plate 1) and processed some 500 tonnes of concentrates of about 1.6% ${}^3\text{O}_8$ from Radium Hill, to produce up to 1.8 grams of high purity-radium bromide and possibly up to 7 tonnes of U_3O_8 in a slurry of about 75% purity.⁴⁵ Mining and milling data for Radium Hill and Woolwich are set out in Table 1, based mostly on records published by the South Australian Department of Mines. The operating costs of the project in late 1913, including mining, concentrating, transport and metallurgical treatment, were estimated at £29 17s 11d per ton. The radium content of 2% U_3O_8 , assuming radioactive (secular) equilibrium, can be estimated as 5.44 mg, leading to an estimated radium value of £148 per ton of 2% U_3O_8 ore. (The New York market price paid for this ore by European buyers, however, was just £15 per ton, leading to eager demand.) It was the early recognition of the 'ratio of the cost of production to the intrinsic value of the ore' that led the Radium Hill Company to build its radium refinery at Sydney rather than Adelaide.⁴⁶

Some of the radium bromide produced at Woolwich was sold on the London market to pre-eminent nuclear physicist Ernest Rutherford, who used it to undertake scientific research on radioactivity. In his report on the purity of the radium bromide, he stated it to be 'free of meso-thorium and other radio-active substances' (that is, to be pure ${}^{226}\text{Ra}$).⁴⁷ The Mark Foy's building at the University of Sydney has tiles that include Radium Hill uranium in the glaze.⁴⁸

The outbreak of the First World War led to a downturn in demand for radium overseas, and with no viable market in Australia, operations ceased in 1915. An adjacent site at Woolwich processed and smelted tin ore and concentrates that contained uranium-bearing and thorium-bearing monazite minerals from 1895 to about 1966, when the land was made into a residential area. The tin ores, especially those derived from mines near Cairns in Queensland, contained elevated natural thorium (${}^{232}\text{Th}$). The radioactive content of tin ores from Ravenshoe and Emmaville in New South Wales was relatively low in comparison.⁴⁹

With the recovery in the radium price and market after the First World War, a fresh attempt was made at developing Radium Hill by the Radium and Rare Earths Treatment Company NL. In 1924, Mawson inspected the field and reported that some 1000 tonnes of concentrates were available, averaging 1% uranium oxide and 1% rare earths. The use of magnetic and gravity concentration was claimed to be successful.⁵⁰

A metallurgical mill was constructed in 1923 at Dry Creek, just north of Adelaide, to extract titanium oxide for pigment production, with radium, uranium, vanadium and scandium as valuable by-products.⁵¹ In 1924, despite marketing difficulties in Germany, a Belgian company, Société des Alliages Industriels, 'made an attractive offer' for a large quantity per annum, but the directors failed to proceed.⁵² Every endeavour was apparently

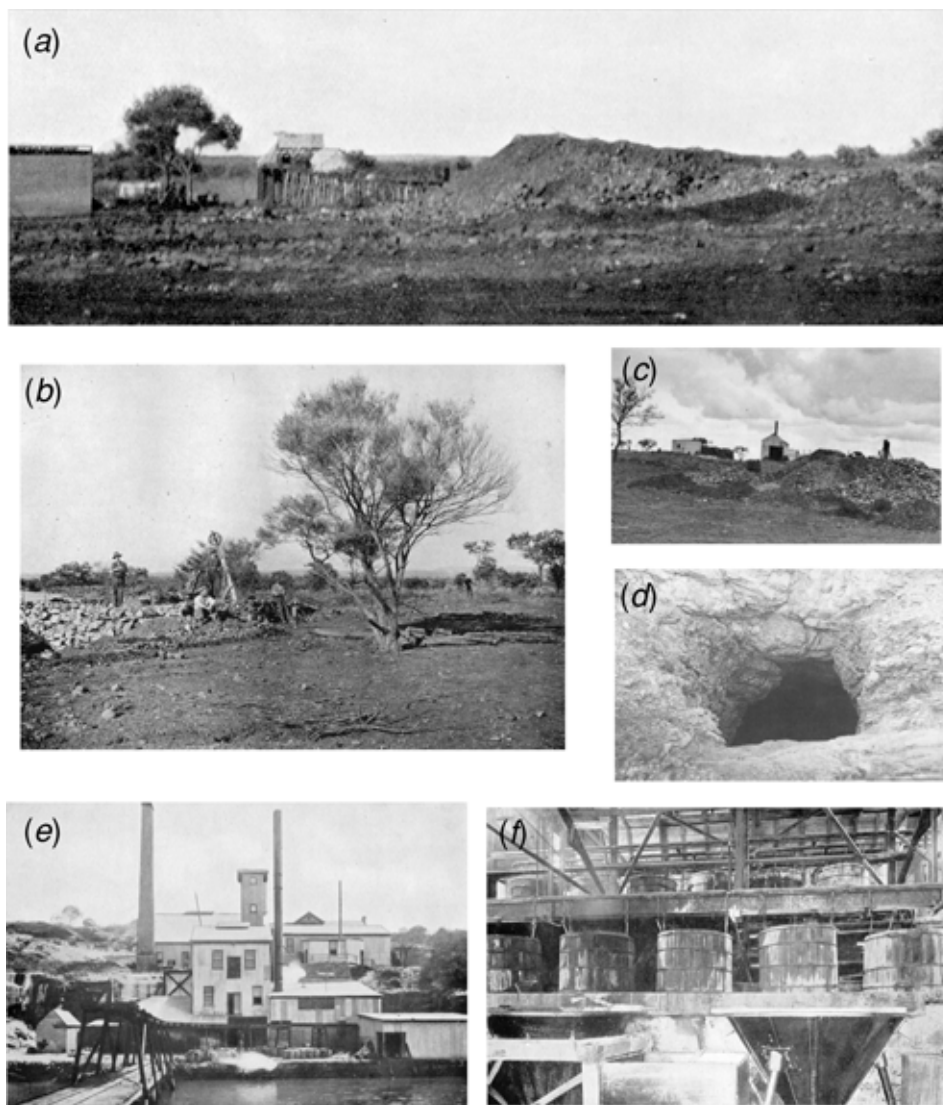


Plate 1. Radium Hill and Woolwich. (a) Radium Hill mine and magnetic mill, ~1912. SA Department of Mines, *Review of Mining Operations in South Australia*, No. 16 (June 1912), p. 9; (b) Radium Hill mine, ~1908. D. Mawson, 'The Nature and Occurrence of Uraniferous Mineral Deposits in South Australia', *Transactions of the Royal Society of South Australia*, 68 (1944), 334–357; (c) Radium Hill mine, January 1925. B. O'Neil, *In Search of Mineral Wealth: The South Australian Geological Survey and Department of Mines to 1944* (Adelaide, 1982); (d) Old stope, Radium Hill, 1944. R. C. Sprigg, *Geology is Fun* (Arkaroola, SA, 1989); (e) Woolwich radium refinery, Sydney, 1912. SA Department of Mines, *Review of Mining Operations in South Australia*, No. 17 (December 1912), p. 12. (f) Inside Woolwich, Sydney, 1912. SA Department of Mines, No. 17 (December 1912), p. 12. Photographs used with permission from Primary Industries & Resources South Australia (PIRSA), Arkaroola Pty Ltd and the Royal Society of South Australia (RSSA).

Table 1. South Australian radium mining and milling data

conc, concentrate; DC, Dry Creek radium refinery, Adelaide, SA; HH, Hunters Hill radium refinery, Woolwich, Sydney, NSW; RH/MP, Radium Hill/Mount Painter; RHN, Radium Hill North mine. All grades in %U₃O₈. References: S. B. Dickinson, *Report on Investigation of Uranium Deposits at Mt Painter, South Australia, June 1944 to September 1945* (Adelaide, 1945); B. O'Neil, *In Search of Mineral Wealth: The South Australian Geological Survey and Department of Mines to 1944* (Adelaide, 1982); SA Department of Mines, *A Review of Mining Operations in South Australia, 1906–1947*, and *Annual Report, 1906–1947*; 'Investigation: Mining, Radium', Series CP211/2/1, Control 32/1, NAA

Year	Radium Hill	Mount Painter	Value
1949		~0.45 t ore to USA	?
1934		18.0 mg Ra	£240
1932		72.0 mg Ra; 0.152 t 'NaUO ₃ ' ^{A,B}	£1050
1927 Dec. half year		45 mg Ra (£450); 0.187 t 'NaUO ₃ ' ^A (£118)	£1088
1927 Jun. half year		2.5 t ore conc; 52 mg Ra	
1926	No Ra	DC, 18.3 t (0.75%); DC, 3 t ore conc (2.6–3.8%); MP, 2.17 t ore conc (6.2%); 700 t ore at surface; no Ra	
1925	3 t ore conc; 7.01 mg Ra; 0.230 t 'NaUO ₃ ' ^C		£172.17
1918			£686
1915 Jun. half year	215 t ore milled, 41 t ore concentrate		
1914 Dec. half year	406 t ore milled, 41 t ore concentrate	6.1 t ore 'high' grade	£5215
1914 Jun. half year	132 t ore milled; >239 mg Ra	20.3 t @ 3.24%; 61 t @ ~1%; 3 t @ 0.8%; 0.8 t @ 5–20%. All to Europe	
1913 full year	167 t mined @ 1.4%	466 mg Ra	£3620
1913 Jun. half year		127 t ore to England @ ~2.6%	
1912 Dec. half year	RH mill @ 10 t/week HH, 122 t smelted HH, 96.5 t treated RHN, 7.1 t ore mined; 350 mg Ra	2.3 t ore 2.02% to Europe 7 t ore ~2% to Europe 0.5 t @ 25% (prior to 1913)	~£50 ?
1911 Jun. half year	610 t ore at surface 44 t ore to Bairnsdale, Victoria	5.1 t ore to Europe	
1909 Dec. half year	31 t ore to Europe; ~3 t to USA		
Total	>2150 t ore milled, up to 1800 mg Ra, up to 7 t U ₃ O ₈ by-product (?). Total value ~£8800	~933 t ore mined, ~2.1% U ₃ O ₈ ; ~666 mg Ra (£2338), ~3 t U ₃ O ₈ (£213). Total value ~£10,000	£18,800

^AUranium produced as sodium diuranate (mineral formula Na₂U₂O₇·6H₂O) (R. G. Thomas, 'The Processing of Radium Ores in South Australia', *Australian Chemical Institute Journal and Proceedings*, 9(6) (1942), 122–133).

^BSold within Australia.

^CApparently the ore was mined at Radium Hill and treated at Dry Creek, which was later used mainly for Mount Painter ore.

made to establish synergies with the radium mining work at Mount Painter.

Despite the effort, Radium Hill again failed to prove a financial success, with work being suspended in early 1926. Later that year, the company was merged with the radium mining interests at Mount Painter to form the Australian Radium Corporation, and by 1932 all work at Radium Hill had ceased. The ore reserves estimated in July 1930 were about 6000 tonnes of low-grade ore.⁵³

There was some fresh interest and potential funds from British entrepreneurs in 1929, although the main focus was Minerva Heights near Mount Painter. At this time the British Government was conducting an inquiry into radium supply for the Empire, through the Radium Subcommittee of its Committee on Civil Research. It concluded that the prospects

seem to be that there are but faint hopes that the Empire (so far as it has been prospected) holds any radium supplies of any importance at all. Of a very poor lot, the Mount Painter and Radium Hill deposits in South Australia show some mild promise, but are very low grade.⁵⁴

Due to a declining world market and the refusal of the Australian Government to provide a proposed £20,000 subsidy, the project did not eventuate.⁵⁵

The total value earned from Radium Hill ore was about £8800.⁵⁶ The ‘radium-rich’ mine waters were also sold at one time as a health tonic.⁵⁷ The facilities and wastes at Radium Hill were simply abandoned, with no community expectation or legislation to the contrary in place, and the site received virtually no attention thereafter.

In 1934, the Australian mining magnate W. S. Robinson, then of the Zinc Corporation, pegged the Radium Hill leases but to no avail.⁵⁸ The field lay dormant until 1940 when a Melbourne mining house, the Australian Mining and Smelting Company Ltd (AMS),⁵⁹ took up the leases at Radium

Hill from 12 November 1940 for two years at a cost of £12 per year in rent plus a minimum of £500 per year of developmental work.⁶⁰ This latter work was requested by Robinson following his visit to Ernest Lawrence at the University of California at Berkeley, who was well known by this time for his work in nuclear physics. AMS sent ore samples to the UK and USA in 1941 for testing but to no avail: no government showed any official interest.⁶¹

During a visit to Britain in October–November 1943, Robinson heard ‘Lord Cherwell (Professor Lindeman) refer to the possible developments of important uses of Uranium’ and again in Britain in May 1944 Robinson heard of ‘its chances of *very important uses by the enemy*’ (original emphasis).⁶² The critical phrase ‘very important uses’ could be interpreted in several ways. It is open to conjecture as to whether Robinson knew of uranium’s potential for an atomic bomb, or of the existence or full extent of the Manhattan Project. Given Robinson’s legendary industrial intelligence and close relationships with many key figures in both Britain and the USA,⁶³ such as British Prime Minister Winston Churchill⁶⁴ and Ernest Lawrence in California (who led the electromagnetic separation plant that produced the highly enriched uranium-235 for the Hiroshima bomb⁶⁵), it can be argued that Robinson at least knew of the potential for an atomic bomb, even if he was unaware of the true extent of Allied progress in this regard.⁶⁶ As he noted in June 1944, ‘rumours regarding the ability of one’s enemies to destroy the Universe are admittedly common in War time, but some of those circulating in London on my last visit were certainly unpleasant’.⁶⁷ Robinson’s trip in May 1944 coincided with Britain’s formally requesting Australia’s assistance in uranium procurement. Robinson quickly re-applied for leases at Radium Hill upon his return to Australia, but the application was denied since the

opportunity had been closed by the Government's awakening to the strategic use of uranium (see later section). As we shall see, only minor attention was given to Radium Hill during the exploration and research work undertaken in Australia in 1944 and 1945.

Little is known about waste management practices at either Radium Hill or Woolwich at this time. There were some basic mine safety standards, largely covering the physical aspects of mining, but there were no regulations in place for environmental management or rehabilitation, or to protect workers' health from radiation.⁶⁸ The radioactive waste legacies are further discussed below.

Mount Painter

The Gammon Ranges region of north-eastern South Australia long held hope for mining entrepreneurs. Mawson's confirma-

tion of torbenite (uranium) in samples found in 1910 by the prospector 'Smiler' Greenwood provided the latest opportunity to justify this optimism. The ensuing radium rush in the Mount Painter region between 1910 and 1914 saw a flurry of activity in both prospecting and mining, as well as at the Adelaide Stock Exchange. The principal company active in the region was the Radium Extraction Company of South Australia Ltd (RECSA), floated on 28 November 1910. Mawson was very closely involved with RECSA during this early period. A new area of high-grade ore was discovered by RECSA manager Harry Fabian in May 1911 and became the rich No. 6 Workings. Several locations were worked for uranium ore, with rich grades often found — up to 20% U_3O_8 , though most grades were around 1–5% U_3O_8 as shown in Table 1. By mid-1912, Archie Broughton had used about 3.6 kg of high-



Cairn at Radium Hill: 'This memorial commemorates the discovery of the Radium Hill Mining Field by Arthur John Smith who pegged his first claim on 24th March 1906'. Photographed by author, September 2001.

grade ore to produce about 30 g of uranyl nitrate ('yellowcake') in the University of Adelaide mineralogical laboratory, primarily to carry out research on radium production. This is most likely the first time that yellowcake was produced in Australia. Although production details are incomplete, at least 233 tonnes of ore averaging 2% U_3O_8 were mined and sold to eager markets in Europe. Thus, while Mawson's contribution was important, the Greenwoods, Fabian and Broughton really deserve the credit during this period.⁶⁹

Despite the rich grades found near the surface, the ore shoots or veins often vanished at shallow depths.⁷⁰ The remote and difficult nature of the terrain led to the use of camels to cart ore and supplies through the ranges and then by truck or rail to and from Adelaide (see Plate 2). It would appear that, except perhaps for the operators at the Stock Exchange, the individuals and companies involved, especially Greenwood and his family, did not make any fortunes.⁷¹ In 1913, with dwindling funds, some ore was shipped to the Woolwich radium refinery, from which 466 mg of hydrated radium bromide was sold.⁷² By 1914, with the First World War depleting the field of miners and with most of the ore having previously been sold to Germany, Mount Painter was without labour and markets and no longer viable. The various companies languished into liquidation by 1917, with RECSA alone having spent some £7000.⁷³

Mawson recognized the scientific quality of many of the uranium mineral specimens and ensured that samples were traded (mostly without profit) to collections at various Schools of Mines, universities and museums around the world. A sample of torbenite was presented to Rutherford on his 1925 visit to Adelaide. Mount Painter was also promoted by the South Australian and Commonwealth Governments at various international mineral exhibitions.⁷⁴

The Gammon Ranges, and the Mount Painter region in particular, are of great cultural significance to the Adnyamathanya traditional owners.⁷⁵ It is clear from my travels and meetings in the region that this early period and the subsequent uranium-mining activity in the region is of significant concern to Adnyamathanya elders.

With the increasing price and growing market for radium, interest was revived in Mount Painter in 1924 by three companies, with Archie Broughton as General Manager for all of them. Following financial difficulties, they merged in September 1926 with activities at Radium Hill to form the Australian Radium Corporation NL (ARC), based in Melbourne.⁷⁶ A second rush developed, though with even less success than the first.

Due to the lack of permanent water at the mine site, the Dry Creek mill (see Plate 2) of the Radium and Rare Earths Treatment Company was apparently used; though the arrangements are unclear, they are likely to have been related to the 1926 merger. A crude ore concentrate was produced at Mount Painter (though often with unsatisfactory results) and this, together with hand-picked ore, was carted by camel and then lorry and train to Dry Creek. It would appear that very little additional ore was mined and treated at this stage (see Table 1), with most of the ore retrieved being remnant from earlier efforts. The estimated ore reserves were as small as 1000 tonnes of low-grade ore.⁷⁷

The Commonwealth Government's decision in September 1927 to purchase 10 grams of Belgian radium, the single largest radium order ever placed to this time,⁷⁸ severely affected the prospects of the ARC, which was not given a chance to tender for the contract or even part of it.⁷⁹ Intense lobbying began, including a request that the Government should help directly with finance or at least help arrange for finance to be made available. Numerous claims were made, including

that ‘the radium was the purest in the world’ and that it might be cheaper than the Belgian product.⁸⁰ At the time, radium was selling for around £10,000 per gram and ARC presented an economic analysis that suggested they could produce radium at about £7500 per gram, and also earn a further 15–25% from the sale of uranium oxides as by-products. ARC also claimed that Dry Creek was about 90% efficient in extracting the radium from the concentrates and that, since the Belgian Government had loaned Union Minière some £2 million to facilitate its work, it was in Australia’s and the Empire’s interests to finance radium.⁸¹

The Commonwealth requested that the South Australian Director of Mines, Dr L. Keith Ward, prepare a report on the

proposed radium operations, and his confidential May 1928 report noted:

- a lack of proven high-grade ore (i.e. this had already all been mined);
- the unknown extent of low-grade ore and uncertain grades;
- the difficulty and uncertainty in concentrating the low-grade ores;
- that the Dry Creek plant was not designed to allow for these problems; and
- that there was a confidential push for an ‘Empire’ radium industry to break the Belgian monopoly.⁸²

Following a meeting at Kurrajong House in Melbourne on 16 April 1928 between Dr Flecker and Mr Coates of ARC and the Commonwealth Government bureaucrat T. M. Owen, the latter noted

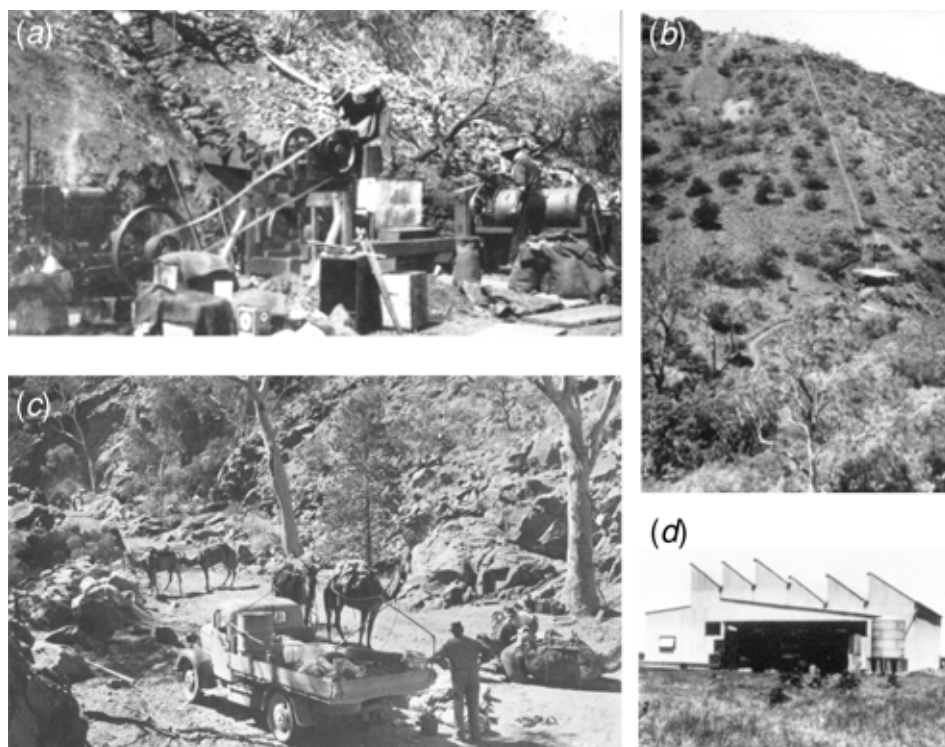


Plate 2. Mount Painter and Dry Creek. (a) No. 6 workings at Mount Painter, 1926. J. F. Drexel, *Mining in South Australia: A Pictorial History* (Adelaide, 1982); (b) No. 6 workings, 1930. (Drexel 1982); (c) Road construction and camels, August 1944. B. O’Neil, *Above and Below: The South Australian Department of Mines and Energy 1944–1994* (Adelaide, 1995); (d) Dry Creek mill, November 1925. (O’Neil 1995). Photographs used with permission from Primary Industries & Resources South Australia (PIRSA).

that ‘Neither of these gentlemen is familiar with mining in any way. They are delightfully vague as to either the tonnages that may be available at the deposits, or its grade in U_3O_8 content.’ The opinion of Dr David Rivett, Chief Executive Officer of the Council for Scientific & Industrial Research (CSIR), was also sought. He was generally unsupportive of radium mining at the time. Overall, the government recognized the high risk involved due to the lack of data on ore grades and to marketing problems arising from the effective monopoly held by the Belgians. The Commonwealth and South Australian Governments refused to assist in financing the project, insisting that if the project was as potentially profitable as ARC claimed, a bank would surely be happy to finance it.⁸³

Despite the lack of political support and the difficulty of raising finance, ARC did proceed with some mining and treated some ore concentrates at Dry Creek. About half a tonne of crude sodium uranate ($Na_2U_2O_7$) concentrate was produced, valued at £213, and 194 milligrams of radium valued at £2338. The late 1920s saw interest from the UK Government and from British entrepreneurs, including attempts by the latter to arrange £20,000 for further field work and exploration to determine the viability of commercial operations. The focus was primarily on the Minerva Heights region near Mount Painter, but no work eventuated. The funds were contingent on obtaining government support, and this was still not forthcoming despite an optimistic new assessment in May 1930 of potential ore reserves of some 22,500 tonnes with 0.4–1.0% U_3O_8 . As noted earlier, the Radium Hill and Mount Painter fields were regarded by the British Radium Sub-Committee as part of a ‘very poor lot’ of ‘very low grade’ prospects within the Empire. By 1932 all mining operations had ceased, and the ARC abandoned its leases in 1934. The total value of Mount Painter ore, based on radium content, was about £10,000.⁸⁴

In 1940, AMS took up the leases at Mount Painter in conjunction with their work at Radium Hill, but they could not overcome apparent ore treatment problems⁸⁵ and surrendered the lease by 1942.⁸⁶ The latter stages of the Second World War and the Allied project to develop the atomic bomb revived interest in uranium from Mount Painter in May 1944 (see later section).

An interesting tangent, partly related to radium–uranium mining, is the Paralana Hot Springs near Mount Painter.⁸⁷ A well-known sacred site of the Adnyamathanja,⁸⁸ it was discovered in the mid-1920s to contain radioactivity.⁸⁹ A private spa and sanitorium was established in 1924, purporting to give medical benefits to visitors similar to other spa ventures of the time. Another example mentioned previously was the ‘radium-rich’ health tonic marketed from mine waters at Radium Hill. The Paralana Spa project was owned and promoted by the same company that was active at Mount Painter, the ARC, and Sprigg states that the venture was an attempt ‘to save the faltering company’. The remote location, the uncertain nature of the supposed benefits, and especially the poor standard of its facilities, led to its quick failure in the Depression years. All subsequent attempts to develop spa or other tourist facilities at the springs have failed, sometimes spectacularly.⁹⁰

Radium Mining across Australia

There is very little published information about radium mining in other parts of Australia. Often only the briefest of references are made, for example:

- In Western Australia, some occurrences of uranium mineralization were known for many years and in 1929, mining for radium was allegedly undertaken at Holleaton, about 115 km south of Southern Cross (200 km west of Kalgoorlie).⁹¹

- In the Northern Territory, apparently in about 1908, there was ‘a radium rush of five on packhorses to Tanumbirini Lagoons of the Limmen River’, while in the early 1930s, ‘radium ores were discovered at Mount Diamond, Wandi and Tanumbirini — visions of fabulous wealth at £565,000 an ounce — but of these pitchblende and uranium ores too little was known, now, alas! Too much.’⁹²
- In New South Wales, interest was shown in Carcoar with the local *Blayney* newspaper reporting in an article titled ‘Radium at Carcoar’ on 12 August 1905, that:

A wire from Carcoar to the ‘Herald’ says that for some considerable time past it has been known that the cobalt mine situated at the south east corner of the municipal boundary contained, in addition to cobalt, an ore called copper uranite. Samples of this ore have been tested and found to contain about 1.7 percent of uranium, the parent of radium. In order to ascertain what quantities could be obtained a number of samples of ore taken from various parts of the mine were collected on Saturday, and will be forwarded to Professor David this week for test. Mining authorities state that if the results prove satisfactory the value of the property will become great, as the present price of radium is stated to be 184,000 (pounds) per oz.⁹³

According to Thomas, the total amount of radium produced across all of Australia probably did not even amount to 1 gram⁹⁴ — a rather minute quantity compared with the thousands of tonnes of ore and wastes generated. Australia’s efforts to exploit uranium ores profitably to produce radium were almost entirely confined to South Australia, though there was clearly considerably more interest than has been recognized until now.

Back to Mount Painter: Australian Uranium and the Rush for the Atomic Bomb

It is commonly stated that Australia’s attention to the strategic importance of uranium

only began in 1944 when the British urgently requested Australia to explore Mount Painter and Radium Hill for the secret Anglo–American atomic bomb effort — the Manhattan Project.⁹⁵ A closer look at archival and other material, however, clearly shows that the Commonwealth and South Australian Governments, and also some well-placed individuals and mining companies, had been aware of the emerging significance of uranium at least as early as 1941, possibly earlier, and were carefully positioning themselves accordingly.

The need for military control of uranium was communicated to the Commonwealth Government in September 1941 by the Australian physicist Mark Oliphant (then playing a seminal role in the British war effort as well as lobbying to establish the Manhattan Project). Oliphant broke the sacred rule of military secrecy by discussing the British work on the atomic bomb, code-named Tube Alloys, with Richard G. Casey, Australia’s representative to the USA in Washington DC. At Oliphant’s request, Casey forwarded the information on 17 September 1941 to David Rivett of the CSIR. Together, Rivett and Oliphant found it difficult to convince officials of the need to safeguard Australia’s uranium. Thereafter, Rivett kept in personal touch with British physicists working on the Manhattan Project and lobbied (albeit in vain) to try and gain greater access for Australian scientists to get directly involved.⁹⁶

In August 1943, just before leaving Britain for the USA to work in the Manhattan Project, Oliphant broached the subject of uranium supplies with Australia’s High Commissioner in London, Lord Bruce, who relayed this information to Australian Prime Minister John Curtin and asked him to start inquiries on the subject. From November 1943 to April 1945 Oliphant worked with Ernest Lawrence at Berkeley on the electromagnetic separation of uranium-235. (The final plant was built at

Oak Ridge, Tennessee, and supplied the fissile material for the Hiroshima bomb.)⁹⁷ Only two other Australian scientists worked on the Manhattan Project, Harrie S. W. Massey and Eric H. S. Burhop, despite efforts by Rivett for greater access and involvement.⁹⁸

There were those who considered that Oliphant was over-enthusiastic in his optimism for the success of the atomic bomb project, though Burhop wrote to Rivett shortly after starting work at Berkeley that, in his view, Oliphant was not over-enthusiastic at all and that this eventual success had several profound implications.⁹⁹

The need to consider thorium in a similar light to uranium was also clear to Rivett and others, and it was noted around this time that Australia was possibly unwittingly exporting thorium to the USA in the form of monazite found in heavy mineral sands concentrates. It was thought that it might be prudent to consider regulating such exports, subject to the opinions and interest of the British of course.¹⁰⁰

In September 1942 Rivett finally succeeded in bringing the issue of control of uranium before the wartime Production Executive Committee, which passed an order establishing Commonwealth control of uranium under the provisions of the National Security Act. There was no publicity. The Production Executive Decision (No. 133, 23 September 1942) was titled ‘Control of Uranium-Bearing Ores’ and included two main decisions:

- (1) immediate action be taken to ensure that the control of uranium-bearing ores in the Commonwealth be reserved to the Crown and not allowed to pass into the hands of private individuals or companies; and
- (2) the Division of Industrial Chemistry of the CSIR and the Minerals Survey Branch¹⁰¹ of the Department of Supply and Development be asked jointly to prepare a report on our present knowl-

edge of the occurrence of uranium-bearing ores in the Commonwealth.¹⁰²

The committee had before it a memorandum dated 17 September prepared by Minister John Dedman that clearly pointed to the military significance of uranium. Rivett had been receiving ‘certain secret information’ for some time, which he guarded as most confidential and only shared with Dedman.¹⁰³ Rivett’s source was Oliphant, who had urged this course of action during a 1942 visit to Australia.¹⁰⁴

An initial report by the CSIR was completed in early November 1942. The report was used by Rivett as the basis for requesting the South Australian Department of Mines to reserve from private interests all uranium-bearing ores in the State — that is, Radium Hill and Mount Painter (over which the AMS leases, by coincidence, had just expired). A new, seven-page report was prepared by Herbert B. Owen (Mineral Survey Branch) and Richard Grenfell Thomas (CSIR) in early 1943.¹⁰⁵ Fortunately, Thomas had worked at Radium Hill and Dry Creek from 1925 to 1928, and his experience allowed the CSIR to proceed quickly.

The South Australian Department of Mines acted without hesitation after receiving Rivett’s request of 13 November, though they apparently did not understand the basis for this action. On 19 November 1942, through the *SA Government Gazette*, ‘all uranium-bearing areas in South Australia’ (that is, Radium Hill and Mount Painter) were removed from the operation of the Mining Act and private interests. Dedman noted in July 1943 that, due to the prompt South Australian action, no further Commonwealth action to control uranium ores was necessary for the time being.¹⁰⁶

On 17 May 1944, the British Chancellor of the Exchequer, Sir John Anderson, approached Prime Minister John Curtin while the latter was visiting London for the Empire Prime Ministers’ meeting, and briefed him on the atomic bomb effort and of the dire need for uranium for ‘Empire

and War purposes'.¹⁰⁷ The British, responsible for Empire uranium under the Combined Development Trust arrangements created following the August 1943 Quebec Agreement for the Manhattan Project, requested the Australian Government to undertake an urgent and intensive investigation for uranium at Mount Painter and Radium Hill.

The full extent of Curtin's verbal briefing from Anderson on the atomic bomb is not available, only Anderson's general 'aide memoire' on Australian uranium, dated 15 May 1944 and cabled back to Australia after the 17 May meeting.¹⁰⁸ Curtin was most likely already informed on the broad aspects of atomic bomb developments by Oliphant via Rivett, but it is not clearly documented whether this British request was formally on behalf of the Manhattan Project. Given the improved Anglo-American co-operation by this time (not forgetting the Belgians and Canadians) and the already sufficient supply of uranium, it would appear that Britain was most likely acting (perhaps optimistically) in its own post-war interests and not to procure additional uranium for the Manhattan Project. The various books and archival material examined attribute the request to the UK Government alone with no mention of the USA, while American sources never mention potential Australian uranium or associated work.¹⁰⁹

From London, Curtin promptly cabled Acting Prime Minister Frank M. Forde in Australia and relayed the information he had received, stressing the urgency and his personal interest in the project.¹¹⁰ There was no time wasted, with Mawson¹¹¹ leading a party to Mount Painter within a week. Personnel from South Australian and Commonwealth departments together with various military and mining industry specialists started work in June 1944. The top-secret project was given urgent priority, giving it virtually unlimited capacity to draw on labour and expertise as required.

Curtin in late June 1944 agreed to export the uranium from the project, given the alleged urgency for the allied war effort, even though the terms, conditions and Australia's possible needs after the war did not appear to have been thoroughly considered. The Mount Painter province, with potentially higher-grade ore known to be more amenable to existing treatment processes, was given the highest priority.¹¹²

The work was highly secret, with no clear statement as to its purpose. It was probably with the wisdom of hindsight that Sprigg later wrote of how 'all that any of us could learn was that the target element, uranium, had the potential for the creation of "atomic suns". For had not European scientists in 1939 ... actually demonstrated in the laboratory the splitting of the U235 uranium atom and established a first chain reaction for the release of relatively large quantities of energy?'. Ben Dickinson, promoted to become South Australia's Government Geologist and Director of Mines in March 1944, was probably reporting more accurately when he recalled that they 'weren't told it was for the bomb. We were told it was for some obscure purpose.' Given the veil of secrecy over all things atomic at the time, it is unlikely that the intended use of any uranium produced would have been divulged to the workers in the field (especially remembering Britain's probable post-war intentions).¹¹³

The people involved in or aware of the secret uranium project included mining industry luminaries such as Maurice Mawby, W. S. Robinson and Gordon Lindesay Clark,¹¹⁴ all associated with the Zinc Corporation (later to become Conzinc Riotinto Australia and now Rio Tinto) or Western Mining Corporation (WMC), or both; government scientist-bureaucrats like Harold Raggatt, Ben Dickinson, David Rivett and Richard Grenfell Thomas; and numerous geologists and geophysicists like Reg C. Sprigg, Robert F. Thyer, D.

Edward Gardner, C. John Sullivan and Edward Broadhurst. The roles played by many of these people in later, post-war uranium exploration and in projects such as Rum Jungle, Mary Kathleen¹¹⁵ and Radium Hill cannot be underestimated. The role played by Australian mining companies has long been overlooked, with the Zinc Corporation/Rio Tinto, AMS and WMC becoming pivotal in the post-war uranium industry, primarily led by figures such as Robinson, Clark and Mawby. These companies, due to their direct involvement in the logistics and to the personal involvement of their geologists, geophysicists and managers, were able to position themselves well in advance of other mining companies. A key aspect was the emerging importance of geophysics in mineral exploration and geological survey work.¹¹⁶ Foreshadowing the future, Robinson stated in an internal Zinc Corporation letter of 22 June 1944 to A. J. Keast:

We have spent much time and several thousand pounds on our efforts to assist in making Uranium available and it would not be unreasonable to expect that, if the Government of Australia desired any outside help, The Zinc Corporation may be given an opportunity to participate.¹¹⁷

Sure enough, the Zinc Corporation was later, in August 1952, given the contract to operate the Rum Jungle uranium project on behalf of the Commonwealth Government, a contract it held until April 1971.

At Mount Painter, the Americans provided a bulldozer — the first seen by many geologists working on the project — that drove the access track through to the base camp. Although some new uranium finds were made, including around former uranium workings at East Painter, the results were mostly disappointing as rich surface veins again faded out at depth. As early as October 1944, it was realised ‘that it is most problematical as to whether we will locate any worthwhile quantity of material’, leading to the conclusion that

uranium reserves at Mount Painter were small and insufficient.¹¹⁸ The British advised the Australian Government on 22 February 1945 that, based on a review of uranium production requirements, the small potential Australian uranium production of 20 tonnes a year was no longer of any interest. By this stage the urgency had eased somewhat since the allies had been able to procure sufficient uranium from Belgian Congo, Canadian and United States mines. The Manhattan Project secured 4010 tonnes of U_3O_8 from the Belgian Congo (pitchblende ore >25% U_3O_8), and 1000 tonnes of U_3O_8 each from Canada (pitchblende ore ~10% U_3O_8) and the USA (Colorado carnotite ore <1% U_3O_8). By this time Britain also felt reasonably assured of an adequate and continuing supply of uranium under the Combined Development Trust arrangements.¹¹⁹

The estimated cost of the Mount Painter project was about £57,750 (after the sale of residual assets) and was paid for by the British; only £1000 was spent on work at Radium Hill. The total number of people who worked on the project was 31.¹²⁰

The total ore reserves did ‘not exceed 500 tons containing 0.33% UO_3 ’ (about 508 tonnes at 0.32% U_3O_8) and with an extraction efficiency of ‘60% being assumed this would yield on treatment 2000 lbs of uranium’ (about 0.91 tonnes of U_3O_8). This would give only 6.5 kg of ^{235}U — not nearly enough for one atomic bomb (let alone several). The final project report, edited by Ben Dickinson and completed in November 1945, after the bomb project had become public knowledge, also noted that the uranium could be used for an atomic bomb.¹²¹

With the nuclear attacks on Hiroshima and Nagasaki in Japan on 6 and 9 August 1945, respectively, the world would forever know (and fear) the awesome power of uranium. The fieldwork in South Australia wound down by the end of 1945, with the State Government under pro-nuclear

Premier Thomas Playford continuing to explore both Mount Painter and Radium Hill as potential uranium mines for the near future. All work was abandoned at Mount Painter by 1949, in favour of developing the Radium Hill prospect, which by this stage had enough proven resources for a potentially economic project, with some of the ore treatment problems becoming manageable.

Although it is stated by some that high-grade uranium ore from Mount Painter was supplied to the USA during 1944 and 1945, this has not been verified and there is nothing on the public record concerning the quantity exported, if any.¹²² According to the South Australian Department of Mines, only ‘small quantities of ore won for laboratory and ore-dressing research’ were supplied.¹²³ Based on the amount used by Thomas, the CSIR and others for metallurgical research, it appears that the total amount of ore used was of the order of several tonnes at most.¹²⁴ There was apparently no ore supplied from Radium Hill during the war-time exploration and research work.¹²⁵

The Legacy: Urban Radioactive Wastes

Overview

As noted previously, very little is known about the practices for waste management at the various mine and smelter sites at Radium Hill, Mount Painter, Woolwich, Dry Creek and Bairnsdale. It must be pointed out that for most of the early period of radium mining, there were no formal national or international standards for radiation protection for workers and the public. The first such standard was proposed as a voluntary code in 1934 by the International Commission on Radiation Protection (ICRP), based on the emerging evidence at that time of the risks associated with exposure to radiation. The review below collates and analyses the available information, mainly focused on Woolwich

due to the availability of a reasonable amount of data about this site.

A collage of photographs from the Radium Hill and Woolwich sites is given in Plate 1, with Mount Painter photographs in Plate 2.

Radium Hill

The overall scale of operations at Radium Hill was very small, even for the mining industry of the times — that is, compared with the numerous gold and coal fields and metal mines in Australia. There is very little known or published about practices, with the only clear information that can be ascertained being from various photographs, mostly published by the South Australian Department of Mines (see Plate 1). As was standard mining practice of the day, it would appear that ore was stockpiled before concentration, with the tailings most likely discharged adjacent to the mill (see 1912 and 1925 photographs). Beyond this, there are insufficient data on radiation levels or other issues to make any reasonably informed judgement, especially as there were no legal or other requirements for rehabilitation of mined land at this time.

An assessment of the scale and nature of the radioactive waste remaining at Radium Hill from the radium era is also problematic due to the development of a large-scale uranium mining and milling project after the Second World War. The traces of the earlier history would have been subsumed within a larger radioactive waste problem that is outside the scope of this paper.¹²⁶

Woolwich

Although the origin of the need to assess the extent of radioactive waste at Woolwich is a matter of some debate, there is a reasonable quantity of data compared with what is available for other radium-era sites. As the location of one of Australia’s first attempts at remediation of a radioactively

contaminated site and disposal of associated wastes, it is important to document it here, both for completeness and because it relates to the radium industry some decades earlier.

Gandy states, presumably on the basis of Radcliff's 1913 paper,¹²⁷ that the liquid wastes at Woolwich would probably have been discharged to the adjacent harbour while solid wastes, including impure and unwanted uranium oxide by-products, were stored or dumped nearby. Assuming radioactive equilibrium for the ore and 86% efficiency, the amount of radium dumped was calculated by Gandy as 10 GBq (280 mg), though this was not based on a complete assessment of the various wastes.¹²⁸

The Woolwich site was built on reclaimed harbour frontage land on the Parramatta River and extended up a cliff face with rough, rocky and sloping terrain (see Plate 1). The re-development for residential purposes would therefore have seen several walls and terraces constructed, the moving of much of the dumped solid waste and soils contaminated from liquid wastes from the old radium refinery, and their use as fill materials.¹²⁹ At the time of the proposed change to residential land in 1965, the site was investigated for radiation exposure levels, measuring gamma radiation doses and the uptake of radium in vegetables grown on soils at the site.

Although several locations showed high gamma dose levels and some radium uptake in plants and vegetables, the site was determined to be safe for residential use and investigations ceased in 1966.¹³⁰ The gamma radiation data are shown in Table 2. No sampling or analyses for radon gas and its decay products was performed. Slag from the adjacent tin-smelting site contaminated with ²³²Th was used in the construction of residential roads, some substantially,¹³¹ though the contamination was allegedly unknown at the time of the 1965 and 1966 radium surveys. All data is combined in Table 2.

In 1976, in the light of public debate on nuclear and radiation issues, the New South Wales Health Commission re-investigated the Woolwich site. It was now considered that the main reason for concern was possible exposure to radon and its radioactive progeny, not gamma radiation as previously thought.¹³² The house built on the site of the refinery's laboratory was the main focus of investigation, that included water and sediment sampling, radon studies and further gamma radiation readings, as set out in Table 2.

Comparing these data to available uranium ore and mill tailings data¹³³ shows that many of the soil and sediment samples from Woolwich are of equivalent activity to uranium ore and therefore a major source of radon and a significant health risk. The sampled soil was estimated to account for about 8.9 GBq of radium (~0.24 g) or about 86% of that dumped by the old refinery. A significant amount of radioactivity was thought to have been dumped in the water.

Based on the data available at the time for the house over the old laboratory, Gandy¹³⁴ calculated that the annual gamma radiation doses of about 1.8 mSv were only marginally above background and required no action, while annual radon doses were between 12 and 24 mSv, making radiation exposure significantly above the then public standard of 5 mSv per year.¹³⁵

After this work was presented, the soil samples were also found to be elevated in thorium, consistent with the elevated uranium and thorium in the monazite-bearing tin ores as well as in the uranium ore processed. This suggested that as time progressed, the radium activity would increase as the decay of thorium (²³⁰Th) would lead to more radium (²²⁶Ra) and therefore radon (²²²Rn).¹³⁶ This situation is different from common forms of radioactive waste that gradually decline in

specific radioactivity over time, whereas that at Woolwich would slowly increase. Clearly, the most rational solution was to remove the contaminated soil and wastes permanently from such valuable real estate.

In early 1978, the New South Wales Government announced its desire to remove about 3000 tonnes of radioactive wastes from the six house blocks, but no site was found for their permanent management. Efforts to relocate the waste to Manara in rural New South Wales or to South Australia were unsuccessful,¹³⁷ due to active opposition from the Bakandji Aboriginal people and the local com-

munity. Following a government directive on 22 June 1982, the New South Wales Department of Health purchased three of the residential blocks. One of these was then remediated and ‘made safe’, with the contaminated soil removed and transferred to the adjacent blocks for storage. The remaining blocks were ‘fenced off, re-vegetated and warning signs ... erected’.¹³⁸ In September 1992, a house on one of the remaining two blocks was demolished with some soil removed and sealed in three 205-litre drums that remain under the control of the New South Wales authorities.¹³⁹ No remedial decontamination

Table 2. Gamma radiation, radon activities in air and radium contamination at Woolwich

Compiled from A. P. S. E. Cardew, ‘²³²Th Contamination from Tin Smelting’, *Radiation Protection in Australia*, 2 (1982), 108–116, and G. F. Gandy, ‘Radium Contamination of Residential Areas’, *Radiation Protection in Australia*, 2 (1982), 117–129

Year	Radiation source/type	Unit	Average	Range	Background
1965	Gamma	μGy/h	0.4	0.14–1.4 (86 ^A)	~0.1
1976	Gamma	μGy/h	?	0.14–1.4	~0.1
	Radon	Bq/m ³	259	237–2916;	?
	?			7400–11,100 ^B	
	Radon progeny	WL ^C	0.13	0.002–0.32	
	Radium (soils)	Bq/kg	34,743	851–244,200	?
1977 ^E	?			(51,800,000 ^D)	
	Radium (sediments)	Bq/kg	1240	259–4070	?
	Thorium-234 (²³⁴ Th)	Bq/kg	31,890	574–271,987	?
	Radium-226 (²²⁶ Ra)		36,467 ^F	851–326,969 ^F	
	Lead-214 (²¹⁴ Pb)		36,615 ^G	740–346,320 ^G	
1977 ^H	Thorium-232 (²³² Th) – slag	Bq/kg	4206	2812–6364	?
	Thorium-232 (²³² Th) – ore	Bq/kg	–	1110 and 2516 and 12,876	
	Gamma	μGy/h	1.7	0.1–3.3	~0.2
	Gamma (1 m height)	μGy/h	1.1	0.2–2.8	
1999 ^J	²³⁸ U, ²³⁰ Th, ²¹⁰ Pb, ²²⁶ Ra	Bq/kg	–	3000–7000	–

^AGamma dose rate directly above a dump site.

^BRadon activities in air beneath the floorboards.

^CWorking Level or WL is a measure of the radioactivity of radon progeny in air. (mWL is mill WL or 10⁻³ WL.) 1 WL = 3746 Bq/m³ of radon in equilibrium with its progeny or equilibrium equivalent concentration (EEC).

^DSingle sample only, most likely an area of liquid waste disposal.

^EAverage of the AAEC analyses of both February and June 1977, total of 21 soil samples.

^FAverage excludes single value of 53,280,000 Bq/kg.

^GAverage excludes single value of 54,020,000 Bq/kg.

^HData for tin smelting site and surrounds (for comparison).

^ITin ores from Emmaville, Ravenshoe and Cairns, respectively.

^JLetter, Plues to Author, 10 March 2005.

works were undertaken, however, at the tin-smelting site.¹⁴⁰

The New South Wales regulatory authorities appear to have been monitoring the Woolwich site at Hunter's Hill since this time, as measurements of gamma dose rates and radon exhalation rates are given by Lenzen and McKenzie. The gamma radiation exposure from the residual contamination at the site was stated as about 1 $\mu\text{Sv/h}$, while radon exhalation rates were about 2.6 times higher than nearby uncontaminated rocks.¹⁴¹

According to Lenzen and McKenzie, the ICRP's recommended maximum gamma level for full-time exposure was 0.57 $\mu\text{Sv/h}$, with the re-named 'Kelly's Bush' site rated as low-risk.¹⁴² Assuming full-time occupation, this equates to 8.8 mSv/year, or 2.9 mSv/yr for typical one-third occupation. The ICRP figure of 0.57 $\mu\text{Sv/h}$ relates, however, to a public dose of 5 mSv/year, not the 1 mSv/year now in use.¹⁴³

In its formal response to a 2004 Joint Parliamentary Inquiry on Nuclear Waste, the New South Wales Government has agreed to 'complete the inventory of non-ANSTO storage sites as a matter of urgency identifying, in particular, those sites where upgrading of facilities is required'.¹⁴⁴ The Woolwich site clearly fits into this category.

As the Woolwich saga demonstrates, it takes only a small quantity of radioactive waste from uranium (or radium) mining, in the right context (e.g. in an urban area), to give rise to significant radiation exposures and on-going waste management dilemmas. The extent of the cancer risk from this additional radiation exposure is only a small increment above background radiation levels, but it does point to the need for final remediation works rather than permanent site management.

Bairnsdale

As noted earlier, the initial ore-treatment research was undertaken at Bairnsdale by

the Director of the Bairnsdale School of Mines, S. Radcliff, who was formerly at Moonta. There is no formal public record of the fate of the radioactive waste after the closure of the School of Mines.¹⁴⁵ Nothing further is known about the history of and current radiation levels at the site.

Mount Painter and Dry Creek

There is considerably less information available on the residual contamination and wastes from the various phases of Mount Painter work than for the Woolwich site. The field still retains many exposed waste dumps, shallow open cuts and underground mine workings, often used for geological research and occasionally for tourist purposes. There is no known information on the approach adopted to radioactive waste management and radiation protection during the war-time exploration work, although this period would have added to the existing workings scattered throughout the immediate region. Further uranium exploration work in the area in the late 1960s and early 1970s clearly exacerbated this legacy, as noted by Commonwealth and South Australian Parliamentary Inquiries.¹⁴⁶

A residential house, apparently in inner-suburban Adelaide, was discovered to be contaminated with radium in the early 1980s, the late resident having been a laboratory technician for radium production from Mount Painter in the 1920s (presumably at Dry Creek). The problem was identified after a 'radium needle' was found. One room where 'some laboratory work' was probably performed contained some 10 MBq of radium (~0.28 mg). It was concluded that the radium had been in solution, and had been heated in the fireplace and spilled on the floor. The contamination data is given in Table 3. After cleaning of the chimney and removal of the floorboards, the residual contamination was not thought to be a significant risk, and no further action was

taken. The radioactive wastes were dumped at an unnamed ‘industrial waste disposal site’. The Dry Creek wastes were dumped at Radium Hill during this time though there are no data on the public record concerning waste volumes or level of radioactivity.¹⁴⁷

Moonta Copper Tailings

The tailings from the former Moonta copper mine present a side story related to the urban radioactive wastes arising from Radium Hill, Woolwich and Mount Painter. As noted previously, radioactivity was confirmed by Radcliff at select parts of the Moonta copper deposit in early 1906, before Radium Hill was discovered. In the early 1980s, regional uranium exploration by WMC revealed that significant radiation emanated from the Moonta copper tailings. Certain sections of the tailings dump gave readings exceeding the Code of Practice for Radiation Protection in Uranium Mining and Milling,¹⁴⁸ based on a 40-hour working week and fifty weeks per year. According to Hill, this was recognised as a public health hazard though no radiation exposure or other data was provided.¹⁴⁹ The area was fenced and allowed to revegetate. As part of mining heritage activities, a railway was built through Moonta and has apparently been success-

ful in reducing the number of people walking over the old tailings dumps.

Conclusion

The history of early attempts to mine uranium ore in Australia primarily for its radium content, is full of unrewarded promise. Despite optimistic efforts, the mining and milling of uranium-bearing ores from Radium Hill and Mount Painter in South Australia failed to be commercially viable. Labour shortages, the tyranny of distance, ore treatment difficulties, lack of strong political support, and the tenuous nature of the radium–uranium market all conspired against success. There was no government support forthcoming for the fledgling industry, though both the South Australian and Commonwealth Governments promoted the rare ores involved at appropriate international exhibitions. This period, however, was instrumental in placing Australia in a perceived advantageous scientific position when the breakthroughs in nuclear physics in the late 1930s led to uranium becoming a strategic element for governments around the world in the 1940s.

Thus, when the British desperately wanted to procure uranium during the Second World War to secure their post-war defence interests, the famous, once-rich

Table 3. Gamma radiation, alpha activities and radon in air at ‘house’

Compiled from T. Passmore, ‘Radioactivity in a Private Residence’, *Radiation Protection in Australia*, 1 (1983), 52–54

Radiation	Unit	Average	Range	Background
Gamma – cont. room	μGy/h	0.25 ^A	(two spot samples both 1.0)	~0.08
Gamma – chimney	μGy/h	~0.4	0.11–0.90 ^B	–
Alpha contamination ^C	kBq/m ²	30–60	Up to 600	?
Radon progeny	mWL	6.9	1.5–15 ^D	0.5–8 (2) ^E

^AGamma dose rate at waist height.

^BDepending on location in or near the chimney.

^CAn indication of radium activity, which decays by alpha decay (including some of the radon progeny).

^DLargely dependent on the ventilation conditions within the room and house.

^EMeasured in a nearby similar house, the average is in parentheses.

ores of Mount Painter in Australia were an obvious choice — and Australia's mining industry and bureaucrats responded with full zeal and endeavour. It is clear that, despite the lack of uranium exports at this time, Australia was a willing, co-operative and active participant in the project that established the permanence of the nuclear weapons menace.

Australia's radium industry, relatively minor in global terms, had fomented glowing expectations around the world and laid the foundation upon which Australia, starting with the war-time exploration work, could base a post-war uranium boom. Indeed, uranium mining and export, especially to the UK and USA, has remained a dominant theme of Australia's global nuclear diplomacy since this time. The history suggests, however, that we are yet to account properly for or to manage in a sustainable way the radioactive wastes deriving from this period. There is still no appropriate facility for long-term stewardship of the Woolwich wastes; numerous waste rock dumps, exploration tracks, adits and shallow open cuts still litter the Mount Painter region; and Radium Hill was later developed into a considerably larger radioactive waste problem.

With regard to the post-war uranium industry, three key conclusions emerge: (i) the relatively minor radium industry was crucial in developing local scientific expertise; (ii) the war-time exploration work involved key figures and mining companies and, although it did not lead to uranium exports at the time, was pivotal for the rapid development of industry interest in uranium following the end of the war; and (iii) the legacy of radioactive waste has still not been solved, especially within an urban context.

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PostScript: Prefixes and Units

μ	micro	10^{-6}	Gy	Gray – gamma radiation exposure
m	milli	10^{-3}	Sv	Sievert – biological effect of radiation exposure
M	mega	10^6	Bq	Becquerel – one radioactive decay per second
G	giga	10^9	U_3O_8	Uranium oxide
WL				Working level – activity and exposure to radon progeny

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30. Radcliff had previously worked in gold mines in Western Australia (e.g. Great Boulder) and was among the earliest metallurgists in Australia to use cyanide in the milling of gold ores; see Clark, *Australian Mining and Metallurgy* (Sydney, 1904).
31. O’Neil, *In Search of Mineral Wealth*, p. 153; Radcliff, ‘Radium at Moonta Mines, South Australia’, *Transactions of the Royal Society of South Australia*, 30 (1906), 201; Mawson, ‘Uraniferous Mineral Deposits in South Australia’, p. 335.
32. O’Neil, *In Search of Mineral Wealth*, p. 153; Sprigg, *Arkaroola-Mt Painter: The Last Billion Years*, p. 229; Mincham, *The Story of the Flinders Ranges* (Adelaide, 1983), p. 237.
33. Sprigg, *Arkaroola-Mt Painter: The Last Billion Years*, p. 229.
34. O’Neil, *In Search of Mineral Wealth*, p. 153.
35. Randall and Driver-Smith, ‘Uranium Mines of the Arkaroola, Mt Painter Province’, pp. 20–21; Mincham, *The Story of the Flinders Ranges*, p. 238.
36. Mawson, ‘New Mineral Species associated with Carnotite’, p. 188; Brown, ‘Carnotite’, *Record of the Mines of South Australia* (Adelaide, 1908), p. 361.
37. O’Neil, *In Search of Mineral Wealth*, p. 153.
38. See Drexel, *Mining in South Australia: A Pictorial History* (Adelaide, 1982); Gandy, ‘Radium Contamination of Residential Areas’, *Radiation Protection in Australia*, 2/82 (1982); O’Neil, *In Search of Mineral Wealth*; Johns, ‘Uranium Exploration in South Australia’, in *Geological Aspects of the Discovery of Some Important Mineral Deposits in Australia*, ed. Glasson and Rattigan (Melbourne, 1990).
39. Gee, *Occurrence of Uranium in South Australia*, p. 11; Harrington and Kakoschke, *We Were Radium Hill: Stories and Memories of People Who Once Lived at Radium Hill* (Adelaide, 1991), pp. 1–2.
40. O’Neil, *In Search of Mineral Wealth*, p. 153.
41. Rodgers, ‘Radium Hill Uranium Deposits’, p. 165; Sprigg, ‘Geology of the Radium Hill Mining Field’, p. 7; see also Anonymous, ‘Radium Hill to Become Major Uranium Producer’, *Chemical Engineering and Mining Review* (1952), 251.
42. Radcliff, ‘Extraction of Radium’, p. 148; SA Department of Mines, *Mining Review*, No. 19, December 1913, p. 33.
43. Now apparently known as ‘Kelly’s Bush’; see Lenzen and McKenzie, ‘Comparative Measurements of ²²²Rn Exhalation from Rocks and Soil of the Sydney Area’, *Radiation Protection in Australasia*, 16(2) (1999), 16.
44. SA Department of Mines, *Mining Review* No. 17, December 1912, p. 11.
45. Exact production not recorded, estimate based on Radcliff, ‘Extraction of Radium’; Gandy, ‘Radium Contamination of Residential Areas’; Dalton, *Radiation Exposures: The Hidden Story behind the Health Hazards behind Official ‘Safety’ Standards* (Melbourne, 1991); SA Department of Mines, *Mining Review*, also compiled in Mudd, *Compilation of Uranium Production History and Uranium Deposit Across Australia* (Melbourne: SEA-US Inc, 2004). There appear to be no records or notes in NSW Department of Mines Annual Reports on the Woolwich radium facility (which appears to be an oversight given radium’s public prominence).
46. SA Department of Mines, *Mining Review*, No. 19, December 1913, p. 30. Curiously, it was noted that the difference in price between value and price paid meant that either the cost of preparing radium salts was excessively high or that miners were not receiving a fair price for their ore.
47. Mawson, ‘Uraniferous Mineral Deposits in South Australia’, p. 344; SA Department of Mines, *Mining Review*, June 1913, p. 11.
48. Cawte, *Atomic Australia*, p. 3.
49. Cardew, ‘²³²Th Contamination from Tin Smelting’, *Radiation Protection in Australia*, 2/82 (1982), 110.
50. Mawson, ‘Uraniferous Mineral Deposits in South Australia’, p. 344; Thomas, ‘The

- Processing of Radium Ores in South Australia', *Australian Chemical Institute Journal & Proceedings*, 9(6) (1942), 124.
51. Thomas, 'Processing of Radium Ores', p. 122; Drexel, *Mining in South Australia*, pp. 284, 289.
52. Maurice Mawby to G. Lindesay Clark, 20 December 1940, p. 158, Series A1146/1, Control N13/13A Part 3, National Archives of Australia (NAA).
53. Dickinson, 'Part I: General Statement. Uranium Investigation, Mount Painter, South Australia', *Report on Investigation of Uranium Deposits at Mt Painter, South Australia, June 1944 to September 1945, undertaken at the request of the British Government by the Government of the Commonwealth of Australia in conjunction with South Australian Government, November 8, 1945* (Adelaide, 1945), p. 7; SA Department of Mines, *Mining Review*, December 1932, p. 39; Memorandum for the Minister, 31 July 1930, p. 39, Series A786/2, Control J64/8, NAA.
54. R. G. Casey to S. M. Bruce, 28 February 1929, in North and Hudson, eds, *My Dear PM: R. G. Casey's Letters to S. M. Bruce, 1924–1929* (Canberra, 1980), p. 464.
55. O'Neil, *In Search of Mineral Wealth*, p. 276.
56. SA Department of Mines, *Mining Review*, No. 84, June 1946, p. 5.
57. Sprigg, *Geology is Fun*, p. 242.
58. King, 'Uranium Prospecting', *AusIMM Bulletin*, 171 Supplement (1954), 2. W. S. Robinson was heavily involved in many mining companies and was also instrumental in forming Western Mining Corporation (WMC) in 1933.
59. AMS was soon after to become a subsidiary of Consolidated Zinc Corporation or 'ConZinc', formed by the 1949 merger of the Australian mining icon, the Zinc Corporation, and the British Imperial Smelting Company. ConZinc later merged with the Australian interests of Rio Tinto Zinc (RTZ) of the UK to form Conzinc Riotinto Australia or CRA in 1962, with RTZ the majority shareholder. RTZ–CRA merged fully in the 1990s to form Rio Tinto. CRA–RTZ–Rio Tinto and their subsidiaries have been very active in Australian uranium since 1940 (e.g. at Rum Jungle, Mary Kathleen, Kintyre, Westmoreland, and more recently at Ranger/Jabiluka, etc.).
60. Maurice Mawby to G. Lindesay Clark, 20 December 1940, p. 157, Series A1146/1, N13/13A Part 3, NAA. See also King, 'Uranium Prospecting', p. 2.
61. Parkin and Glasson, 'The Geology of the Radium Hill Uranium Mine, South Australia', *Economic Geology*, 49(8) (1954), 815; Sprigg, 'Geology of the Radium Hill Mining Field', p. 8; W. S. Robinson to A. J. Keast, 22 June 1944, p. 75, Series A1146/1, Control N13/13A Part 3, NAA.
62. W. S. Robinson to A. J. Keast, loc. cit.
63. For example, Robinson, *If I Remember Rightly: W. S. Robinson Memoirs* (Melbourne, 1970); see also Ralph, 'The Origins of Western Mining Corporation and Some of the Men Who Shaped It', in *AusIMM Centenary Conference* (Adelaide: Australasian Institute of Mining and Metallurgy, 1993), pp. 367–368; Hudson, Kelly, Robinson and Way, eds, *Documents on Australian Foreign Policy 1937–49* (Canberra, 1988).
64. Following the end of the Second World War, a Holy Bible in honour of Robinson was given to Scotch College (his former school). The inscription by Winston Churchill and Brendan Bracken read, in part, 'in the six relentless years of war that preceded the victory of 1945 the services manifold of William Sydney Robinson to the British Commonwealth were beyond computation'; Dew, *Mining People: A Century: Highlights of the First Hundred Years of the AusIMM 1893–1993* (Parkville, Victoria, 1993), p. 281.
65. See Smyth, *Atomic Energy for Military Purposes: The Official Report of the Development of the Atomic Bomb under the Auspices of the United States Government 1940–1945* (Princeton, NJ, 1946); Groves, *Now It Can Be Told: The Story of the Manhattan Project* (London, 1963).
66. Further to this, the British were sure that mining companies were alert to the use of uranium as a potential source of power and were being very careful in all of their uranium procurement efforts (the price was already rising); see Gowing, *Britain and Atomic Energy*, p. 181. It is therefore hard to believe that a person of Robinson's political connections and entrepreneurial calibre was not aware of similar interests and potential imperatives for Australia's uranium.
67. W. S. Robinson to A. J. Keast, 22 June 1944, pp. 75, Series A1146/1, Control N13/13A Part 3, NAA.
68. The first radiation exposure standards for workers were recommended in 1934 by the International Commission on Radiological Protection (ICRP) (see later section).
69. Dickinson, 'Part I – Uranium Investigation, Mount Painter', pp. 5–6; Randall and Driver-Smith, 'Uranium Mines of the Arkaroola, Mt Painter Province', pp. 22–23; Sprigg, *Arkaroola-Mt Painter: The Last Billion Years*, pp. 5, 229.

70. This was a common complaint about Mt Painter ores; see Thomas, 'Processing of Radium Ores', p. 123.
71. Sprigg, *Arkaroola-Mt Painter: The Last Billion Years*, p. 231.
72. Brugger, Ansermet and Pring, 'Uranium Minerals From Mt Painter', p. 18.
73. Drexel, *Mining in South Australia*, pp. 284, 289; Sprigg, *Arkaroola-Mt Painter: The Last Billion Years*, p. 230; Randall and Driver-Smith, 'Uranium Mines of the Arkaroola, Mt Painter Province', p. 24; Coats and Blissett, *Regional and Economic Geology of the Mount Painter Province* (Adelaide: SA Department of Mines, 1971), p. 148.
74. Thomas, 'Processing of Radium Ores', p. 124; Sprigg, *Arkaroola-Mt Painter: The Last Billion Years*, p. 230; Randall and Driver-Smith, 'Uranium Mines of the Arkaroola, Mt Painter Province', p. 23; Sprigg, 'Colourful and Exotic Minerals of the Flinders Ranges', in *Natural History of the Flinders Ranges*, ed. Davies, Twidale and Tyler (Adelaide, 1996), p. 32; Brugger, Ansermet and Pring, 'Uranium Minerals From Mt Painter', pp. 18–19.
75. Sprigg, *Arkaroola-Mt Painter: The Last Billion Years*, pp. 29–48; Tunbridge, *Flinders Ranges Dreaming* (Canberra, 1988), p. 126.
76. O'Neil, *In Search of Mineral Wealth*, p. 275; Randall and Driver-Smith, 'Uranium Mines of the Arkaroola, Mt Painter Province', p. 25.
77. Drexel, *Mining in South Australia*, p. 289; pp. 19 and 39, Series A786/2, Control J64/8, NAA.
78. Richardson, *Australian Radiation Laboratory*, p. 2.
79. Personal Papers of Prime Minister Bruce; Deputation representing the Radium Company, 16 November 1927, Series CP362/2/1, Control 33; p. 35, Series CP211/2/1, Control 32/1; NAA.
80. This is probably related to the earlier report from Rutherford. See also p. 2, Series CP362/2/1, Control 33, NAA; SA Department of Mines, *Mining Review*, No. 18, June 1913, p. 11.
81. pp. 39, 41 and 42, respectively, Series CP211/2/1, Control 32/1, NAA.
82. Memorandum to Development and Migration Commission, 14 May 1928, p. 25, Series CP211/2/1, Control 32/1, NAA.
83. Notes of Interview, 16 April 1928, p. 35, Series CP211/2/1, Control 32/1; and Note for Mr Tonkin, p. 29, Mines and Mining M–Z, Radium, Series A786/2, Control J64/8; NAA.
84. Dickinson, 'Part I – Uranium Investigation, Mount Painter', p. 8; O'Neil, *In Search of Mineral Wealth*, p. 276; H. J. Lipman to J. Gunn, 12 May 1930, p. 81, and 'Radium Corporation – Additional Capital Sought', p. 163, Series A786/2, Control J64/8; NAA; North and Hudson, eds, *My Dear PM: R. G. Casey's Letters to S. M. Bruce, 1924–1929*, p. 464; Coats and Blissett, *Geology of Mount Painter*, p. 149.
85. AMS focused on physical concentration of the ore, as done previously at Radium Hill — acid leaching was already known to be feasible if one ignored the lack of water; see Sprigg, *Geology is Fun*, p. 243, or Cablegram Curtin to Forde, 17 May 1944, pp. 43–44, Series A571/141, Control 1944/1789, NAA.
86. O'Neil, *Above and Below: The South Australian Department of Mines and Energy 1944–1994* (Adelaide, 1995), p. 73.
87. For example, see Sprigg, *Arkaroola-Mt Painter: The Last Billion Years*, pp. 249–254, and Sprigg, *Geology is Fun*, pp. 114–125.
88. Tunbridge, *Flinders Ranges Dreaming*, p. 126; Sprigg, *Geology is Fun*, pp. 122–123.
89. Dalton, *Radiation Exposures*, p. 92. See also Mawson, 'The Paralana Hot Springs', *Transactions of the Royal Society of South Australia*, 51 (1927), 191; Kerr-Grant, 'Radioactivity and Composition of Water and Gas of the Paralana Hot Spring', *Transactions of the Royal Society of South Australia*, 62 (1938).
90. O'Neil, *Above and Below*, p. 375; Sprigg, *Arkaroola-Mt Painter: The Last Billion Years*, p. 250; Mincham, *The Story of the Flinders Ranges*, pp. 208–209; Sprigg, *Geology is Fun*, pp. 122–123; also *Advertiser* (Adelaide), 25 May 1946.
91. Stewart, 'An Assessment of the Search for Uranium in Australia', in *8th Commonwealth Mining and Metallurgical Congress*, ed. Woodcock (Melbourne: Australasian Institute of Mining & Metallurgy, 1965). However, according to Carter, 'Mineral Deposits of Western Australia: Uranium', in *Mining in Western Australia*, ed. Prider (Perth, 1979), claims about radium mining at Holleaton are misleading (p. 162).
92. Hill, *The Territory* (Sydney, 1970), pp. 256, 418. Attempts to verify this have not been successful, but it is reported for completeness. It is unclear if this is the Mount Diamond that was mined for copper and gold by United Uranium NL in the 1960s.
93. Courtesy of historical research by Robyn Ryan (email 8 March 2002).
94. Thomas, 'Processing of Radium Ores', p. 132.
95. For example, see Battey, Miezeitis and McKay, *Australian Uranium Resources* (Canberra: Bureau of Mineral Resources, Geology

- & Geophysics, 1987), p. 2; Dunn, Battey, Mieztis and McKay, 'The Distribution and Occurrence of Uranium', *Geological Aspects of the Discovery of Some Important Mineral Deposits in Australia*, ed. Glasson and Rattigan (Melbourne, 1990), p. 455; McKay and Mieztis, *Australia's Uranium Resources, Geology and Development of Deposits* (Canberra, 2001), p. 6. See also Uranium Information Centre, 'Uranium Exploration in Australia', Mines Paper 7, last updated 10 May 2005; <http://www.uic.com.au/explor.htm> (accessed 19 June 2005).
96. Cockburn and Ellyard, *Oliphant*, pp. 112–123; Cawte, *Atomic Australia*, pp. 2–3, 6–7; Reynolds, *Australia's Bid for the Atomic Bomb*, (Parkville, Victoria, 2000), p. 27; Gowing, *Britain and Atomic Energy*, p. 315.
97. Cockburn and Ellyard, *Oliphant*, pp. 112–123; Cawte, *Atomic Australia*, pp. 2–3, 6–7; Reynolds, *Australia's Bid*, p. 27; Gowing, *Britain and Atomic Energy*, p. 315. See also Smyth, *Atomic Energy for Military Purposes*, and Groves, *Now It Can Be Told*.
98. On 5 January 1944, Rivett wrote to F. W. G. White of the CSIR stating that Oliphant had requested an Australian or two to work on 'Tube Alloys', the British side of the atomic bomb project. Massey was already working at Berkeley; Rivett nominated Burhop who went to America in May 1944; see Hudson, Kelly, Robinson and Way, eds, *Documents on Australian Foreign Policy*, pp. 7–8, 398–399, 527, 594.
99. *Ibid.*, pp. 344–345, 398–399. Burhop also noted that any opportunity to get more people involved in the work would be beneficial.
100. A. V. Smith to D. Rivett, 17 July 1944, p. 77; H. G. Raggatt to A. V. Smith, 13 July 1944, pp. 81 and 147–148, Series A1146/1, Control N13/13A Part 3, NAA; p. 38–39, Series A571/141, Control 1944/1789, NAA.
101. Following the war this became the Commonwealth Bureau of Mineral Resources, Geology & Geophysics (BMR, now Geoscience Australia). The BMR went on to complete major uranium exploration work in the following decades.
102. Production Executive Decision No. 133, 23 September 1942, p. 259, Series A1146/1, Control N13/13A Part 3, NAA.
103. David Rivett to A. V. Smith, 23 May 1944, pp. 196 and 270, Series A1146/1, Control N13/13A Part 3, NAA. Dedman was Minister for War Organisation of Industry and the CSIR in the Curtin Labor Government (see <http://www.aph.gov.au> accessed 12 October 2004).
104. Hudson, Kelly, Robinson and Way, eds, *Documents on Australian Foreign Policy*, pp. 7, 344; Gowing, *Britain and Atomic Energy*, p. 315. Although there is no evidence that either Massey or Burhop was lobbying or informing Rivett (Cawte, *Atomic Australia*, p. 6), later in 1944 Burhop was certainly advising Rivett of the benefits of further involvement of Australian scientists.
105. *Uranium Minerals in Australia*, pp. 237–243, 254–258, Series A1146/1, Control N13/13A Part 3, NAA. Thomas had been another student of Mawson's at the University of Adelaide in the 1920s and was considered an expert on mineral processing, with a particular knowledge of uranium and monazite (indeed these were listed as some of the fifty projects he recommended for research upon joining the CSIR in 1941); see Bear, Biegler and Scott, *Alumina to Zirconia: The History of the CSIRO Division of Mineral Chemistry* (Clayton South, Victoria, 2001), pp. 5–7, 21–23.
106. L. K. Ward to A. V. Smith, 20 November 1942, p. 248; David Rivett to L. K. Ward, 13 November 1942, p. 252; and Note to Production Executive, 29 July 1943, by John Dedman, p. 232, Series A1146/1, Control N13/13A Part 3, NAA.
107. See Hudson, Kelly, Robinson and Way, eds, *Documents on Australian Foreign Policy*, pp. 324–325.
108. This 'aide memoire' is available from various sources, e.g. *Ibid.*, pp. 324–325; also pp. 43–44, Series A571/141, Control 1944/1789, NAA.
109. See Gowing, *Britain and Atomic Energy*, pp. 187, 315; Hudson, Kelly, Robinson and Way, eds, *Documents on Australian Foreign Policy*, pp. 534–535; Series A571/141, Control 1944/1789, NAA.
110. Dickinson, 'Part I – Uranium Investigation, Mount Painter', p. 1; Cawte, *Atomic Australia*, p. 4.
111. Mawson's 1944 paper was published quickly in November 1944, presumably as part of this project: 'Now that there is a revival of interest in uranium ... circumstances have arisen which deem it expedient for me to publish immediately', Mawson, 'Uraniferous Mineral Deposits in South Australia', p. 334. Ayres states that after 'World War 2 Mawson would feel understandably "sore" that "considering [he] was fundamental in establishing the presence of uranium in the Mt Painter and

- Radium Hill fields and had spent so much money and time thereon the Government ... closed the door for exploitation by anybody else” Ayres, *Mawson: A Life*, p. 41.
112. Cawte, *Atomic Australia*, pp. 4–6; O’Neil, *Above and Below*, pp. 73–75; Brugger, Ansermet and Pring, ‘Uranium Minerals From Mt Painter’, p. 20; p. 32, Series A571/141, Control 1944/1789, NAA.
 113. Sprigg, *Geology is Fun*, pp. 233–234; Sprigg, *Arkaroola-Mt Painter: The Last Billion Years*, p. 231; O’Neil, *Above and Below*, pp. 74–76; see also Reynolds, *Australia’s Bid*, p. 63.
 114. Then Deputy Controller of Mineral Production for the Commonwealth.
 115. A history of the first phase of development at Mary Kathleen, 1954–1963, is given by Harding, *Wholeheartedly and At Once: A History of the First Operation of Mary Kathleen Uranium Ltd 1954–1964*.
 116. The BMR (see earlier), replaced the MSB, and throughout the 1940s had a ‘virtual monopoly’ on geophysics expertise in Australia. The extensive geophysics work by the BMR, for uranium as well as other minerals, helped to lead the uptake of geophysicists in Australia; see Wilkinson, *Rocks to Riches: The Story of Australia’s National Geological Survey* (St Leonards, NSW, 1996).
 117. p. 76, Series A1146/1, Control N13/13A Part 3, NAA.
 118. p. 25, Dept of External Affairs to Commonwealth Government Representative, London, 13 October 1944, Series A571/141, Control 1944/1789, NAA.
 119. Sprigg, *Arkaroola-Mt Painter: The Last Billion Years*, p. 283; Cawte, *Atomic Australia*, p. 7; SA Department of Mines, *Mining Review*, No. 82, June 1945, p. 13; Eisenbud, ‘Early Occupational Exposure Experience with Uranium Processing’, in *Conference on Occupational Experience With Uranium*, ERDA 93 (Arlington, VA: US Energy Research & Development Administration, 1975), p. 18; Gowing, *Britain and Atomic Energy*, pp. 307–19. It is interesting to note that the operation of the USA–UK Combined Development Trust was extended well after the end of the Second World War, with most Australian uranium production being sold into it in the 1950s.
 120. Dickinson, ‘Part I – Uranium Investigation, Mount Painter’, p. 5; SA Department of Mines, *Mining Review*, No. 82, June 1945, pp. 11–13; Randall and Driver-Smith, ‘Uranium Mines of the Arkaroola, Mt Painter Province’, p. 29; pp. 2–4, Series A571/141, Control 1944/1789, NAA.
 121. Dickinson, ‘Part I – Uranium Investigation, Mount Painter’, p. 16.
 122. For example, a *Sydney Daily Telegraph* article on 8 August 1945 (see Cawte, *Atomic Australia*, p. 7), as well as Mason, *No Two the Same*, Publication 3/94 (Melbourne, 1994), pp. 149. Wayne Reynolds reports that he found no evidence in various national archives suggesting any export of uranium ore from Mount Painter for the Manhattan Project (email 5 April 2002)
 123. SA Department of Mines, *Mining Review*, June 1945, p. 13.
 124. See all project reports in Dickinson, ed., *Report on Investigation of Uranium Deposits at Mt Painter, South Australia, June 1944 to September 1945 undertaken at the request of the British Government by the Government of the Commonwealth of Australia in conjunction with South Australian Government, November 8, 1945* (Adelaide, 1945).
 125. Harrington and Kakoschke, *We Were Radium Hill*, pp. 1–3.
 126. See O’Neil, ‘“National Heroes, Not National Villains”: South Australia and the Atomic Age’, in *Playford’s South Australia: Essays on the History of South Australia, 1933–1968*, ed. O’Neil, Raftery and Round (Adelaide, 1996) and O’Neil, *Above and Below*. A review of the radioactive waste management problems for the 1950s–1960s uranium project at Radium Hill is given by Mudd, ‘Remediation of Uranium Mill Tailings Wastes in Australia: A Critical Review’, in *2000 Contaminated Sites Remediation Conference* (Melbourne: CSIRO Centre for Groundwater Studies, 2000). More recently, a draft management plan has been prepared by the South Australian Government, see McLeary, *Radium Hill Uranium Mine and Low Level Radioactive Waste Repository Management Plan: Phase 1 – Preliminary Investigation* (Adelaide: Primary Industry & Resources SA [PIRSA], 2004).
 127. Radcliff, ‘Extraction of Radium’.
 128. Gandy, ‘Radium Contamination of Residential Areas’, p. 118.
 129. *Ibid.*
 130. *Ibid.*, p. 119.
 131. Cardew, ‘²³²Th Contamination from Tin Smelting’, *Ibid.*, pp. 108, 111.
 132. Gilpin, ‘Uranium: Bang or Whimper?’ in *The Australian Environment: 12 Controversial Issues* (South Melbourne, Victoria, 1980), p. 128; Dalton, *Radiation Exposures*, pp. 86–87.

133. Mudd, 'Remediation of Uranium Mill Tailings Wastes in Australia: A Critical Review', in vol. 2, p. 779.
134. Gandy, 'Radium Contamination of Residential Areas'.
135. The 5 mSv per year public radiation exposure limit was recommended in 1966 by the ICRP and adopted by the National Health and Medical Research Council (NHMRC) in 1967 (email, Dr Peter Burns, ARPANSA, 27 June 2003); also ICRP, *Recommendations of the International Commission on Radiological Protection*, 1966.
136. Gilpin, 'Uranium: Bang or Whimper?', p. 128.
137. 'No Time to Waste': *Report of the Senate Select Committee on the Dangers of Radioactive Waste* (Australian Senate, Canberra, 1996), p. 54.
138. L. Plues, NSW Department of Environment and Conservation, to author, 10 March 2005; Panter, *Radioactive Waste Disposal in Australia* (Parliamentary Research Service, Department of the Parliamentary Library, Canberra, 1992), p. 5. Curiously, the Commonwealth's proposal for a national low-level radioactive waste dump near Woomera, South Australia, fails to mention the Hunters Hill radioactive waste despite its being larger in volume and of similar radioactivity to the Fishermen's Bend waste currently stored on a temporary basis at Defence facilities in Woomera (see *National Radioactive Waste Repository Draft EIS: Main Report & Appendices*, prepared by PPK Environment & Infrastructure Pty Ltd for the Commonwealth Department of Education Science & Training, Sydney, 2002).
139. Plues to author, 10 March 2005 (see previous note).
140. Cardew, '²³²Th Contamination from Tin Smelting', p. 115.
141. Lenzen and McKenzie, 'Comparative Measurements of ²²²Rn Exhalation', refer to 'Ashton, W. and others, 1986, *Kelly's Bush Landscape and Management Plan*. NSW Department of Environment and Planning by Travis Partners Pty Ltd', p. 21. The units given are mSv/h though units of μ Sv/h are more likely to be correct.
142. *Ibid.*, p. 16.
143. ICRP, *Recommendations of the International Commission on Radiological Protection* (1977); ICRP, *Recommendations of the International Commission on Radiological Protection* (1990); NHMRC, *Recommendations for Limiting Exposure to Ionizing Radiation and National Standard for Limiting Occupational Exposure to Ionizing Radiation* (Canberra, 1995). According to the ICRP, an exposure of 3 mSv/year equates to a cancer risk of 1 in 10,000 per year.
144. NSW Government Response to the Inquiry into the Transportation and Storage of Nuclear Waste, Recommendation 6; Joint Select Committee on the Transportation and Storage of Nuclear Waste, Parliament of New South Wales, 2004).
145. According to an unconfirmed news report, the radioactive waste was dumped in a nearby mine shaft and the school redeveloped as a TAFE; N. Papps, 'Secret Nuclear Hazard on City Streets', *Herald Sun* (Melbourne), 12 March 2001.
146. For example, see Brugger, Ansermet and Pring, 'Uranium Minerals From Mt Painter', or photographs in Drexel, *Mining in South Australia*. In 1970, the Australian Senate Select Committee on Water Pollution noted evidence from Dr K. R. Miles, South Australian Government Geologist, that work at Mount Painter was leading to water quality problems; Report of the Senate Select Committee on Water Pollution (Australian Senate, Canberra, 1970), Evidence and Transcript, p. 629. In 1972, a South Australian Government Environmental Inquiry noted that 'if the [recent] exploration had been conducted under the provisions of the latest legislation, much less despoilation of the landscape might have occurred'; *Report of the Committee on Environment in South Australia* (Adelaide, 1972), pp. 192–193.
147. Passmore, 'Radioactivity in a Private Residence', *Radiation Protection in Australia*, 1(2) (1983), 52, 54; O'Neil, *Above and Below*, p. 375. The dump site was most likely the radioactive waste facility at Radium Hill, gazetted in April 1981 though with no environmental impact assessment.
148. The 1980 Code of Practice was the Australian regulatory standard for radiation protection.
149. The Moonta tailings radiation problem was discussed by Hill, 'Mine Closures in South Australia', in Australian Mining Industry Council Environmental Workshop, Launceston, 1986.

Sustainability of Uranium Mining and Milling: Toward Quantifying Resources and Eco-Efficiency

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The mining of uranium has long been a controversial public issue, and a renewed debate has emerged on the potential for nuclear power to help mitigate against climate change. The central thesis of pro-nuclear advocates is the lower carbon intensity of nuclear energy compared to fossil fuels, although there remains very little detailed analysis of the true carbon costs of nuclear energy. In this paper, we compile and analyze a range of data on uranium mining and milling, including uranium resources as well as sustainability metrics such as energy and water consumption and carbon emissions with respect to uranium production—arguably the first time for modern projects. The extent of economically recoverable uranium resources is clearly linked to exploration, technology, and economics but also inextricably to environmental costs such as energy/water/chemicals consumption, greenhouse gas emissions, and social issues. Overall, the data clearly show the sensitivity of sustainability assessments to the ore grade of the uranium deposit being mined and that significant gaps remain in complete sustainability reporting and accounting. This paper is a case study of the energy, water, and carbon costs of uranium mining and milling within the context of the nuclear energy chain.

1. Introduction and Background

The nuclear industry has long been a controversial issue, commonly linked to issues such as nuclear weapons and nuclear waste. In Australia, the primary debate has often centered on uranium mining and milling as we have significant economic resources—seen by some as worthy of export for financial return or simply to maintain our position in the global nuclear fraternity.

At present there is vigorous global debate about the perceived potential for nuclear power to reduce greenhouse gas emissions—the central hypothesis put forward by pro-nuclear advocates being the apparent low carbon intensity of nuclear power compared to that of fossil fuels. From an environmental sustainability perspective, it is critical to accurately evaluate the true life cycle costs of all forms of electricity production, especially with respect to greenhouse gas emissions. For nuclear power, a significant proportion of greenhouse gas emissions is derived from the fuel supply,

including uranium mining, milling, enrichment, and fuel manufacture. However, there are only limited data reported by uranium miners with respect to greenhouse gas emissions. Further, additional issues that need to be considered for uranium mining and milling include the extent of economic resources known and the average ore grade of these resources. These aspects are critical in assessing the long-term ability of nuclear power to reduce greenhouse gas emissions.

This paper compiles and presents the available data on uranium mining and milling, with a particular emphasis on historical production trends, known economic resources, and greenhouse gas emissions, as well as water and energy consumption. This is then placed within the context of sustainability metrics applied to uranium mining and milling.

2. Methodology and Data Sources

The various aspects of sustainability investigated in this paper are assessed through the compilation of detailed data sets on (i) uranium mining and milling - historical government series/periodicals on mining; (ii) uranium resources - historical government series/periodicals on mining as well as recent company annual financial or technical reports; (iii) energy and water consumption - recent company annual sustainability or technical reports; and (iv) carbon dioxide emissions - recent company annual sustainability or technical reports.

Select sustainability data for the last two aspects are only available for a few uranium mines, namely Rössing in Namibia, McLean Lake and Cluff Lake in Canada, and Ranger, Beverley, and Olympic Dam in Australia (the latter being a polymetallic Cu–U–Au–Ag mine). There are many aspects which remain unreported since, historically, they have not been considered necessary for financial or production reporting, including chemicals used (acid/alkali, lime, solvents, ammonia), all associated transport, explosives, the embodied energy and water in infrastructure, and the like.

2.1. Data Sources: Uranium Mining and Milling. The data on uranium mining and milling are available for Canada - 1959–2003 (1), 2004–2006 (2–4); United States - 1948–2005 (5) (especially the 1992 report); South Africa - 1952–2006 (6) (including the CMSA Web site for 2006 data); Australia - 1954–2006 (7); Namibia - 1976–1989 (8–11), 1986 to 1994 courtesy of Uranium Information Centre (“Reviewing Rössing 1994”), and 1995–2006 (12) (some data estimated/cross-calculated between sources for verification); and Mongolia - 1988–1996 (13).

Additionally, data were compiled for in situ leach and byproduct derived uranium, thereby allowing a more accurate estimate of uranium production.

2.2. Data Sources: Uranium Resources. Various economic uranium ore deposits data were compiled by country, based on numerous company annual or other reports, plus the following: Australia - 1945 (14), ~1952 (15), 1958–1960 (1), 1987 (16), 2001 (17), 2005 company reports and ref (7); Canada - 1957–1963 (1), 2005 company reports (e.g., refs (2) and (3); United States - 1958–1960 (1), 1992–2003 (5), 2005 company reports; South Africa - 1958–1960 (1), 2005 company reports (incomplete country resources); Namibia - 2005 company reports (incomplete country resources); and Kazakhstan, Malawi, Mongolia, Niger, France, Zambia, Brazil, Argentina, Central African Republic and Russia - 2005 company reports (often incomplete country resources).

All data above were summed to compare calculated totals with country resources reported by the 2005 edition of ref (9). Further data have been compiled on other uranium

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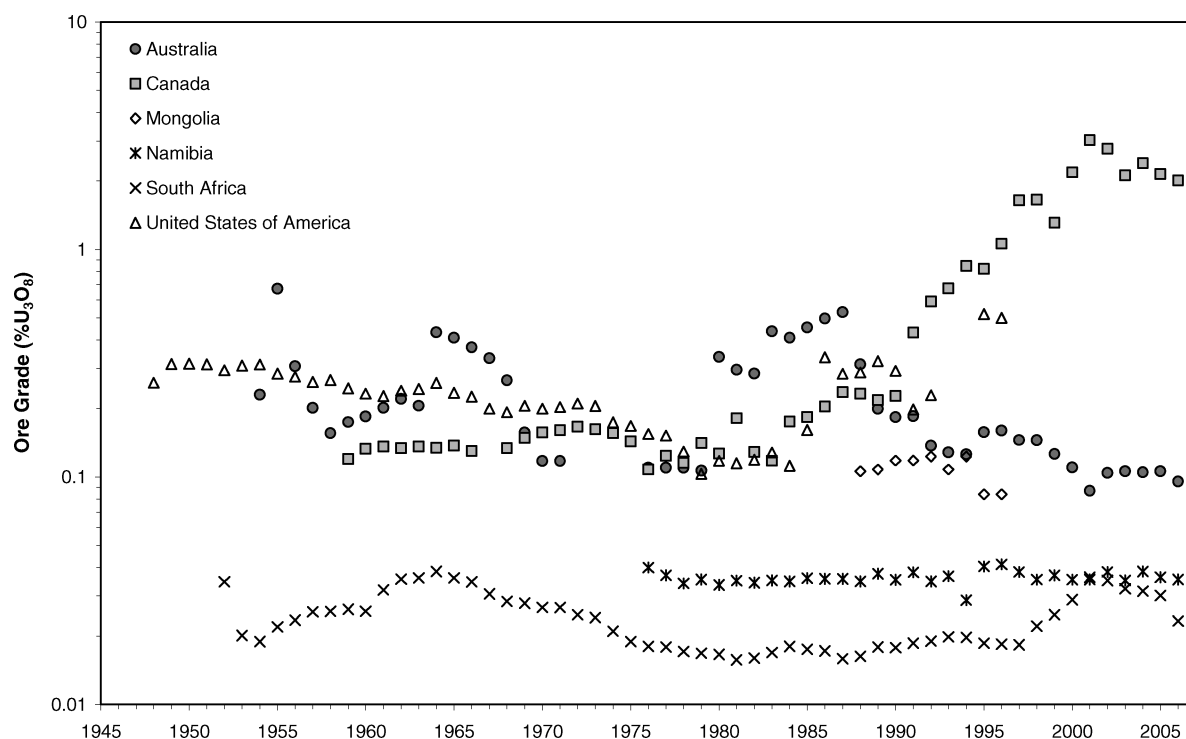


FIGURE 1. Average uranium ore grade in milling over time.

resources, such as phosphates, for comparison to conventional uranium deposits.

2.3. Data Sources: Environmental Aspects of Uranium Mining and Milling. At present, there are only limited publicly reported data on energy and water consumption in uranium mining and milling and greenhouse gas emissions. Some companies, e.g., Cameco and BHP Billiton, report company-wide totals and not site-specific data. Data available include: Rössing, Namibia - open cut mine and adjacent mill, 1995–2006 (12); Ranger, Australia - open cut mine and adjacent mill, 1983/84–1987/88 (18) (note - data are provided for 1981/82 but as the first year of operations it is excluded as an outlier) and 1996–2006 (19); Beverley, Australia - acid in situ leach project, 2003–2006 (20); Olympic Dam, Australia - underground mine, adjacent mill, and copper smelter/refinery complex, 1991–2004 (21) and 2004/05 (22) (note - Olympic Dam is a polymetallic project producing refined copper, calcined uranium oxide concentrate, and gold and silver bullion); McLean Lake, Canada - open cut mine and adjacent mill, 2002–2005 (3); and Cluff Lake, Canada - open cut mine and adjacent mill, 2002 (3) (note - closed in early 2003 and now in rehabilitation).

All data have been normalized to consumption per unit uranium oxide (U_3O_8) production. If input fuels such as diesel were reported, energy and greenhouse gas emissions were calculated using ref (23). All mines analyzed reported both direct and indirect energy and greenhouse gas emissions (or this could be calculated given available data).

To account for the fact that the Olympic Dam project is polymetallic (Cu–U–Au–Ag), data are presented in terms of attributing either all energy and water consumption and carbon dioxide emissions to uranium production or only 20%. Although assuming 100% is clearly unrealistic, the recent average ore grade at $\sim 0.08\% U_3O_8$ is higher than the Rössing uranium mine's at $\sim 0.04\% U_3O_8$. The full energy accounting for direct uranium production at Olympic Dam would need to consider a detailed analysis and breakdown of the milling, metallurgical, and smelting processes for copper, uranium, gold, and silver—which is obviously impracticable (only inputs and outputs are known, not internal aspects). The

factor of 20% is adopted as this is the long-term average proportion of revenue from uranium at Olympic Dam (7).

Beverley is excluded from ore grade graphs due to the uncertain nature of the actual ore grade being mined by acid leaching. Prior to development, uranium resources were estimated at 9.7 Mt at $0.18\% U_3O_8$, containing about 21,000 t of U_3O_8 (7).

3. Results

3.1. Global Uranium Production. The global production of uranium began in large scale following World War II, initially to supply the nuclear weapons programs of the times, but switching to the emerging civil nuclear power industry from the late 1960s. Total production has been dominated by the United States, Canada, (former Eastern) Germany, South Africa, Australia, Czech Republic, Niger, Namibia, and France as well as smaller production from several countries (2005 edition of ref (9)). Complete production data are not available for all of these countries, however, a significant portion is available, especially for several of these principal producers.

In total, the compiled cumulative data represents 1.27 Mt U_3O_8 and accounts for more than half of estimated cumulative global uranium production (~ 2.25 Mt U_3O_8) and most of the western world's total uranium production (~ 1.6 Mt U_3O_8) (2005 edition of ref (9)). The average ore grade for milling over time for the above countries is shown in Figure 1, with the estimated global data for ore milled, ore grade, and production in Figure 2. The estimated percentage of global uranium production, which the compiled data represent, is shown also, demonstrating that the data generally represent $>80\%$ of western world uranium production in the 1960s and greater than 60% since the 1970s. In situ leach mine production was excluded due to the difficulty of equivalence between solution and hard rock mining. Given the data include the current major producers, Canada, Australia, and Namibia, the data provide a reasonable representation of the global uranium industry. Two peaks of uranium production are clearly

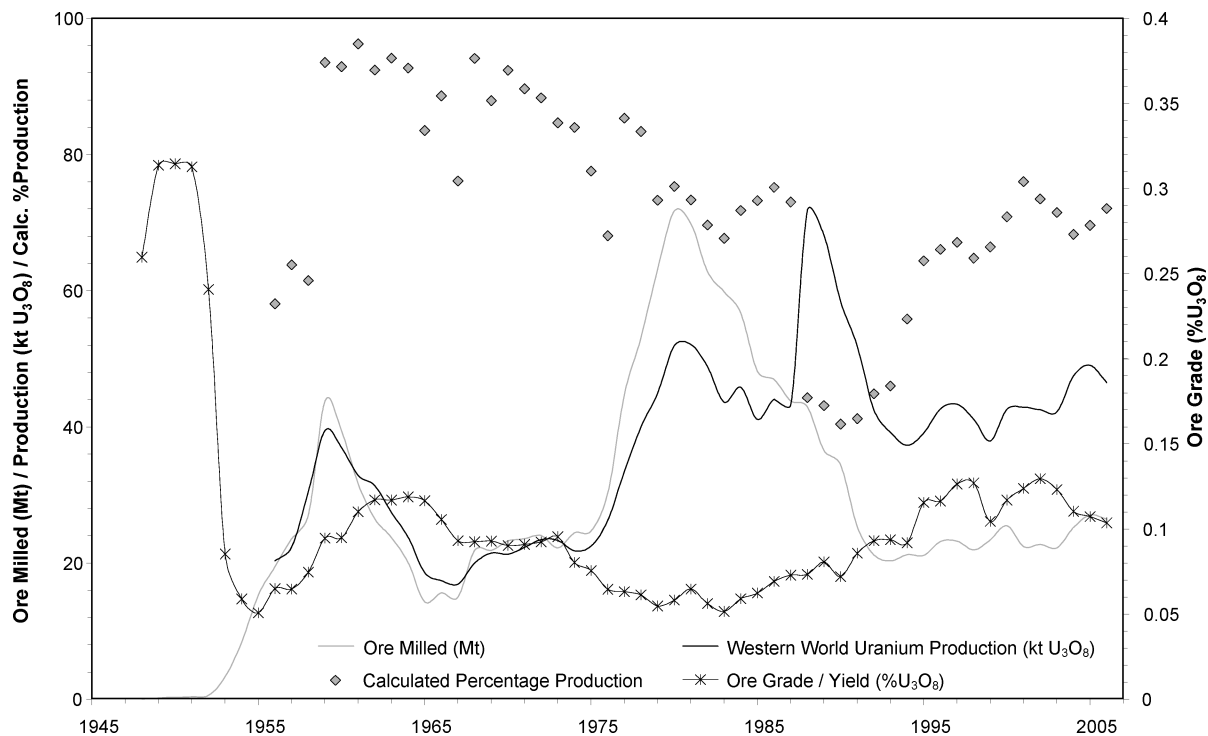


FIGURE 2. Estimated global average uranium ore grade, production, ore milled, and calculated percentage of production.

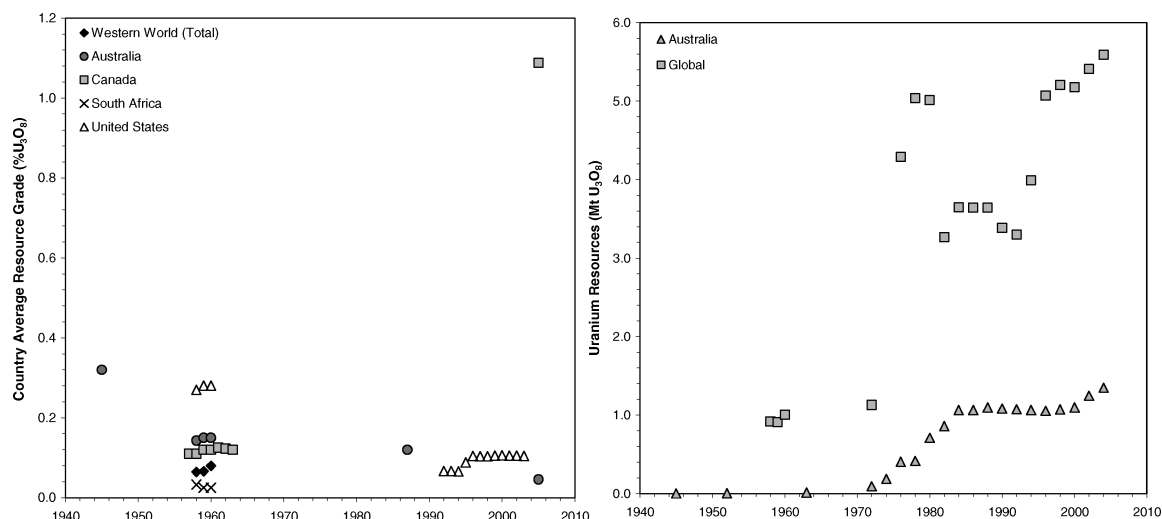


FIGURE 3. Average ore grade of select country uranium resources (left) and global and Australian known economic uranium resources (right) over time.

evident in Figure 2—the weapons phase peaking in 1959 followed by the civil phase peaking in 1988.

3.2. Global Uranium Resources. It is commonly perceived that uranium is a finite resource. The known availability of uranium has been considered to be limited in the past, with further exploration work leading to further resources being found. For example, at the start of the nuclear arms race in the 1940s, uranium was considered to be extremely scarce, yet rapid and wide-ranging exploration soon proved an abundance of uranium far in excess of that required (24).

The second period of uranium mining and milling (for civil nuclear power) also faced this same dilemma in the 1960s, but exploration again found additional uranium resources, particularly in Australia, Canada, Namibia, and Niger. The principal aspects of economic resources include the estimated contained uranium as well as the average ore grade of an individual deposit. Although country resources

over time are compiled and analyzed by ref (9), the ore grades and other salient statistics of the numerous deposits are invariably never presented.

All publicly listed mining companies, at least in western-style economies, are generally bound by voluntary industry codes and/or the law to report accurately on economic ore resources they control. Given the largely western economic control of the global uranium industry, it is therefore possible to compile an up-to-date assessment of recent uranium deposit resource statistics. This can then be compared to the limited earlier data available.

In total, the compiled data totals 3.8 Mt U_3O_8 of uranium resources and accounts for more than half of estimated total global uranium resources (5.5 Mt U_3O_8 , 2005 edition of ref (9)). The ore grade of select country uranium resources over time and global and Australian known economic uranium resources are given in Figure 3, with numerous individual

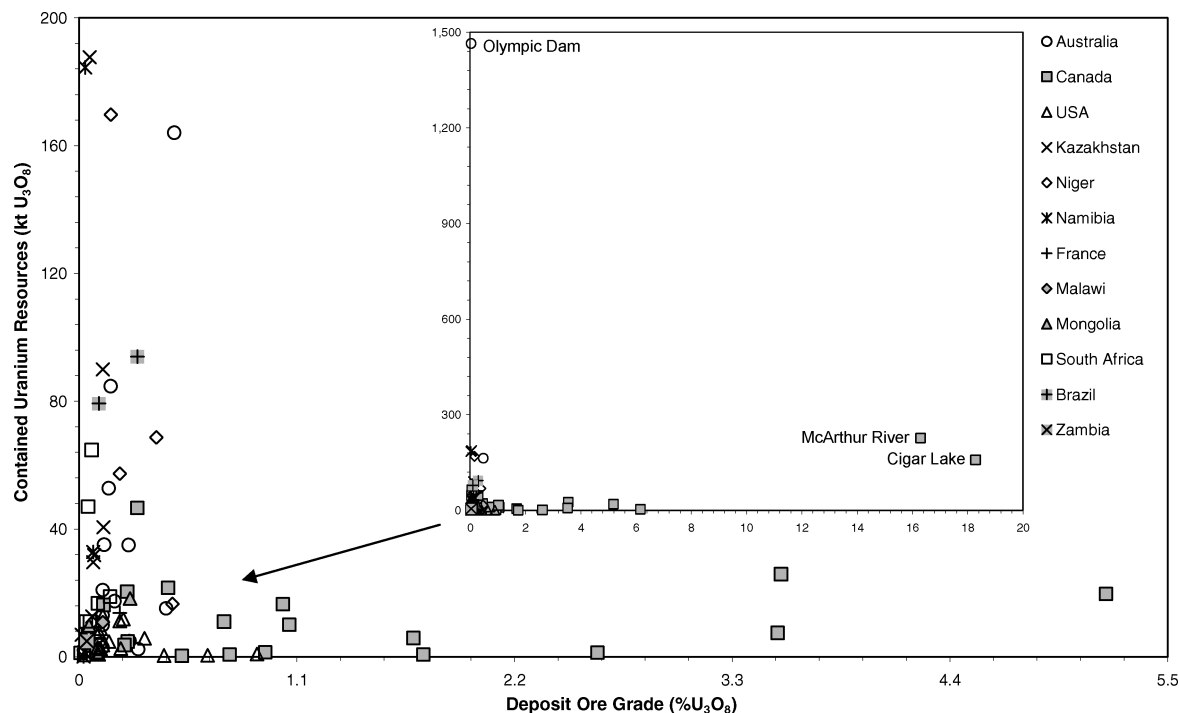


FIGURE 4. Contained uranium resources versus ore grade: individual deposits by country.

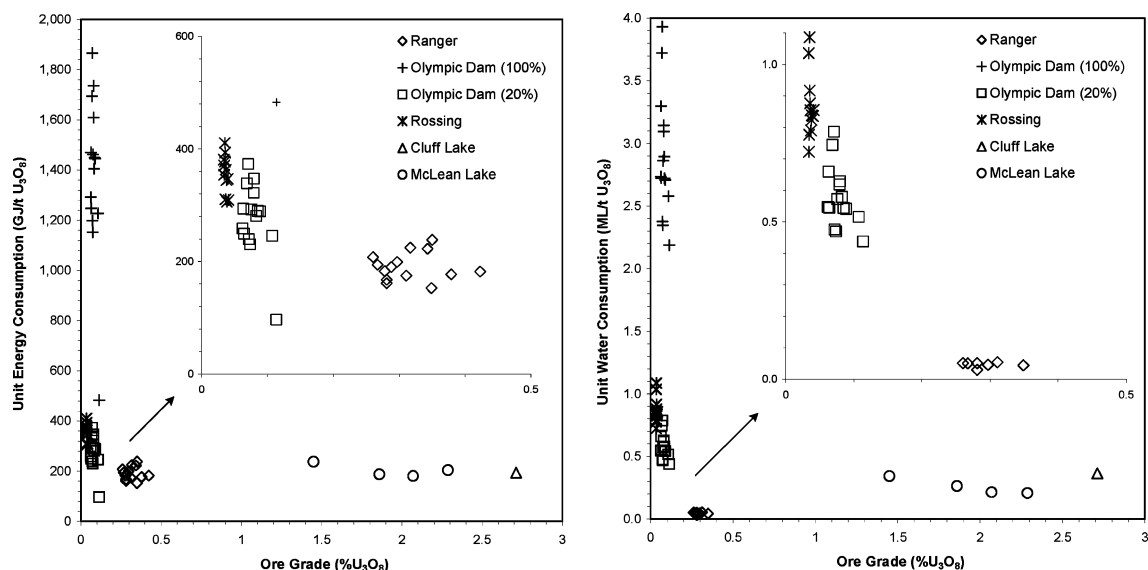


FIGURE 5. Energy and water consumption per uranium oxide produced versus ore grade.

deposits by ore grade and contained uranium compiled in Figure 4 by country.

3.3. Energy and Water Consumption in Uranium Mining and Milling. The compiled data for energy and water consumption per unit of uranium oxide production with respect to ore grade are shown in Figure 5, and with respect to time in Figure 6. As can be seen, using a 20% factor places the unit energy consumption of Olympic Dam within the same order of magnitude as Rössing. The higher water consumption of Beverley in Figure 6 is due to the fact it is an in situ leach mine. The data are summarized in Table 1.

3.4. Carbon Dioxide Emissions From Uranium Mining and Milling. The compiled data for carbon dioxide emissions per unit of uranium oxide production with respect to ore grade and over time are shown in Figure 7. As can be seen,

using a 20% factor places Olympic Dam within the same order of magnitude as Rössing. The data are summarized in Table 1.

4. Discussion

The data compiled and presented within this paper provide support for a number of key aspects of uranium mining and milling, centered around known economic resources, ore grades of resources and production, energy and water consumption per uranium oxide production, and greenhouse gas emissions (carbon dioxide) per uranium oxide production.

The extent of economic uranium resources has generally increased over time, coincident with the major periods of exploration. In Canada, the Elliot Lake region of Ontario provided most resources during the 1950s–1960s, switching to Saskatchewan from the 1970s. The extremely high grade

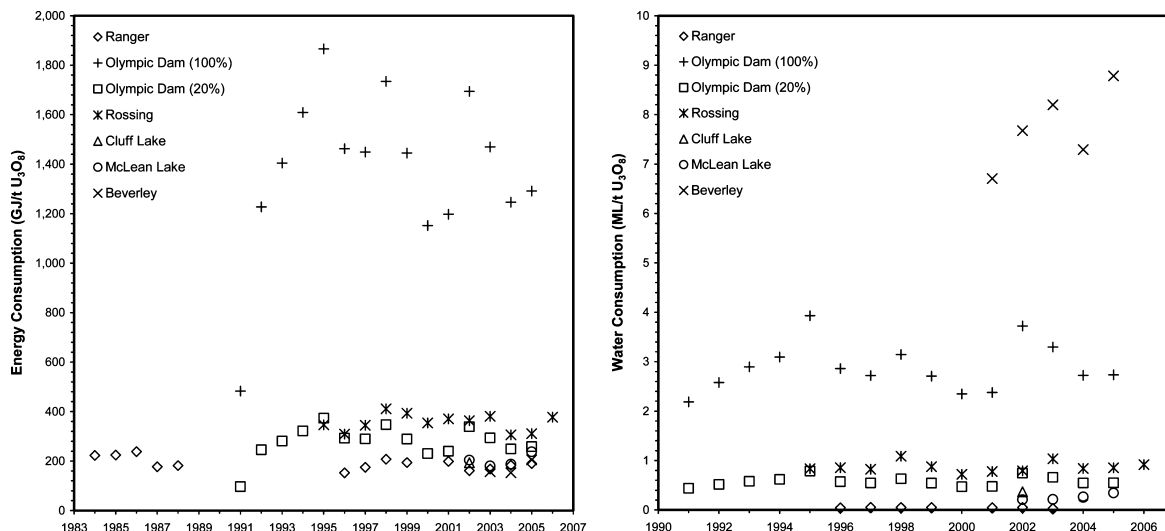


FIGURE 6. Energy and water consumption per uranium oxide produced versus time.

TABLE 1. Summary of Normalized Energy and Water Consumption and Carbon Dioxide Emissions for Uranium Mines (Average \pm Standard Deviation, Number of Years in Brackets)

uranium project	typical ore grade %U ₃ O ₈	annual production t U ₃ O ₈	consumption		emissions
			water	energy	carbon dioxide
			kL/t U ₃ O ₈	GJ/t U ₃ O ₈	t CO ₂ /t U ₃ O ₈
Ranger	0.28–0.42	~5,000	46.2 \pm 8.2 (7)	191 \pm 25 (14)	14.1 \pm 2.3 (15)
Olympic Dam (x%)			2,888 \pm 487 (15)	1,382 \pm 325 (15)	252 \pm 65 (15)
Olympic Dam (x%)	0.064–0.114	~4,300	578 \pm 97 (15)	276 \pm 65 (15)	50.4 \pm 13.0 (15)
Rössing	~0.034–0.041	~3,700	868 \pm 104 (12)	356 \pm 34 (12)	45.7 \pm 4.2 (12)
Cluff Lake	2.71	(closed)	365 (1)	194 (1)	12.1 (1)
McLean Lake	1.45–2.29	~2,750	257 \pm 62 (4)	202 \pm 25 (4) ^a	8.4 \pm 1.2 (4)
Beverley	~0.18	~1,000	8,207 \pm 1,370 (6)	198 \pm 57 (4) ^b	10.3 \pm 3.0 (4)
Niger ^c	~0.2–0.5	~3,100	no data	~204	no data
Cameco ^d	~0.9–4.0	~8,500	no data	~178	no data

^a Different data for 2000 are given by ref (26) as 313 GJ/t U₃O₈, although this is also the first year of full production and may not be representative compared to data compiled above (for years 2002–2005). ^b Different data for 2004–2005 are given by ref (26) as 187 GJ/t U₃O₈, compared to data reported by ref (20) and used in graphs and table above. ^c Data for 2000 for Areva's (formerly Cogema) two mine/mill complexes (Somair and Cominak) (26). ^d Data average over 1992–2001 for "Cameco Saskatchewan mines" (26).

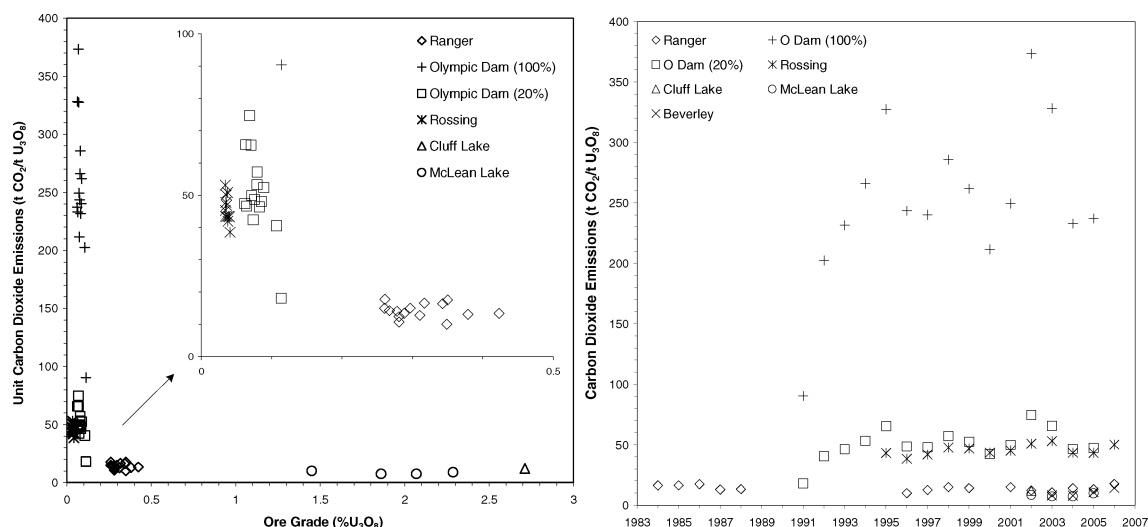


FIGURE 7. Carbon dioxide emissions per uranium oxide produced versus ore grade and time.

deposits of Cigar Lake and McArthur River were discovered in 1981 and 1988 with grades of 18.3% and 14.3% U₃O₈, respectively (prior to development) (2). Although new

prospects are being found, only the Millenium prospect from late 2002 has to date proven substantive (about 26 kt U₃O₈ at ~3.55% U₃O₈; 2005 edition of ref (2). No deposits of the

significance of Cigar Lake and McArthur River have been found since 1988.

In Australia, despite broad-ranging exploration in the 1970s with associated spectacular results, there have only been two new economic deposits discovered since 1975: the modest Kintyre in 1985 and the new Beverley 4 Mile in 2002 (although an economic mineral resource was not confirmed until early 2007). All increases in uranium resources between 1985 and 2005 have resulted from increased drilling and new assessments at known deposits, mainly Ranger and Olympic Dam. This pattern of no “world-class” discoveries greater than 50 kt U_3O_8 in the past two decades is thought to be similar in other countries (e.g., see ref (9)).

Although beyond the scope of this paper, significant additional uranium resources are likely to be available as a byproduct from phosphate ore resources (e.g., Florida), which have produced uranium in the past. It is entirely possible that with further exploration new uranium deposits could be found, however, some issues need to be considered. First, given the broad coverage of uranium exploration globally over the past 50 years, any new deposit discovered is most likely to be deeper than most current deposits. This trend is evident in Canada, where successive deposits discovered in Saskatchewan have each been deeper, and future deposits are expected to be found even deeper still (e.g. ref (25)). The deeper a deposit the more energy which could be expected to be required to mine the resource. Second, the long-term trend over the past five decades has been a steady decline in most average country ore grades (even allowing for varying economic assessments of resources). This is particularly evident in Australia, where the increasing size of the Olympic Dam deposit now dominates Australia’s total resources and average ore grade. The average country ore grade for the United States in the 1990s was typically 0.07–0.11% U_3O_8 , which is about one-third of that in the late 1950s of 0.28% U_3O_8 . Canada is the only country which has seen a substantive rise in its average ore grade, due to the rich Athabasca Basin deposits of northern Saskatchewan (e.g., McArthur River, Cigar Lake, Midwest). The average ore grade of the Elliot Lake district of northern Ontario, which generally contained more than 95% of Canada’s resources in the 1950s to 1960s, was typically 0.11% U_3O_8 —compared to the estimated average of 1.1% U_3O_8 in 2005 (based on resource data compiled for this paper). These trends in average ore grade of country resources are reflected in the ore grades of as-milled production (Figure 1). It is worth noting that despite the increasing ore grade in Canada, this has not significantly affected typical global average ore grade, which has remained between 0.05 and 0.13% U_3O_8 over the past five decades (even allowing for incomplete production and considering likely grades at remaining countries). Finally, based on data for 93 deposits/fields compiled for this paper (Figure 4), there is an indicative relationship between ore grade and contained uranium. As ore grade declines, there is an increasing possibility of substantial tonnage. In terms of major production capacity for any proposed nuclear power program, it is clear that these larger-tonnage, lower-grade deposits would need to be developed, thereby continuing to balance the rich Saskatchewan deposits into the future.

A common issue raised with uranium is the ability for a major contribution to production from byproduct sources such as phosphate and gold ores. Virtually all South African uranium has been derived as a byproduct from gold mining in the Witwatersrand Basin. In the United States some uranium was produced as a byproduct from phosphate mining until their permanent closure in 2000 (capacity of about 1,150 t U_3O_8 at that time; 2001 edition of ref (9)). The Olympic Dam project in Australia, containing copper, uranium, gold, silver, and rare earths, is the only major operating mine not solely mining a deposit for uranium,

though Olympic Dam is more correctly described as a coproduct mine due to the economic importance of uranium. Over recent years, only South Africa has continued byproduct uranium production from gold ores. A detailed examination of all editions of ref (9) shows that byproduct uranium has been a minor component of global uranium production to date (probably of the order of less than 20%). There is very little recent data on uranium resources from byproduct operations, especially ore grades and quantity, nor information available to discern or allocate energy, water, and reagent costs and pollutant emissions to the additional effort required for this byproduct uranium.

With respect to energy, gradual increasing trends are apparent for Olympic Dam, Beverley, Ranger, and McLean Lake, although Rössing shows a slight decreasing trend over time (excluding the single year for Cluff Lake). The data reported for these select mines and compiled herein are only based on direct fuel inputs, such as diesel and/or electricity. Given the data provided, there appears to be little difference in unit energy costs per uranium oxide production above an ore grade of about 0.5% U_3O_8 . Given the small number of points greater than 0.5%, however, this interpretation requires caution. A curious fact shown by the data above is that the energy cost of Beverley, an acid in situ leach project, is similar to that for Ranger, a large open cut mine/mill complex. For Beverley, a recent energy efficiency audit in 2004 showed that the well field and mill consumed 44.9% and 41.6% of electricity usage, or in terms of activities pumping consumed 80.7% of electricity usage (2004 Edition of ref (20)). The energy cost of drilling at Beverley remains unquantified and given the number of bores involved in acid leach mining and milling, it should certainly not be ignored in a true energy cost analysis.

Critically, the data for all mines does not account for the additional embodied energy required for reagents such as solvents (e.g., kerosene, amine), sulfuric acid, oxidants (e.g., hydrogen peroxide, manganese dioxide or MnO_2), lime, and so on. This would add further energy costs to uranium production. For example, data for the Ranger mine from 1988/1989 to 1996/1997 (18) suggest that each tonne of uranium oxide production requires about 320 L of kerosene, 12.7 L of amine, 460 kg of ammonia (NH_3) 1.75 t of oxidant (as t MnO_2), 15 t of acid (as t H_2SO_4), and 5.9 t of lime. For kerosene, the embodied energy is estimated as 36.6 GJ/kL (23), thereby adding about 60,000 GJ to Ranger’s energy requirements for some 5,000 t of U_3O_8 annual production. This would add approximately 11.7 GJ/t U_3O_8 or 6% to the 191 GJ/t U_3O_8 presently reported. Unfortunately more recent annual data since the 1997 mill expansion at Ranger are not available. It is clear that full life cycle accounting and sustainability reporting needs to include reagents with major embodied energy costs.

For water, gradual increasing trends are apparent for Olympic Dam, Beverley, and McLean Lake, although Ranger and Rössing show a slight decreasing trend over time (excluding the single year for Cluff Lake). There are marked differences in water consumption, due in large part to the major differences among these various projects. For example, although Ranger and Rössing are somewhat similar in terms of uranium production and scale for open cut mining, Rössing has an ore throughput about 5-fold that of Ranger as well as an ore grade some eight times lower, thereby leading to significant demands for water. The sensitivity of normalized water consumption to ore grade is apparent. Further characterizing water consumption based on water quality and the extent of recycling is not possible based on the available reported data.

The direct emission of carbon dioxide (and equivalents) is an issue of critical importance, especially in the context of the current debate over greenhouse gas emissions from

the nuclear chain. As with energy and water consumption, gradual increasing trends for normalized emissions are apparent for all mines (excluding the single year for Cluff Lake). The data in terms of carbon dioxide emissions per tonne of ore milled, although not presented within the space of this paper, show that Olympic Dam and McLean Lake are gradually declining over time while Ranger and Rössing are increasing. The declining trends are most likely related to the recent expansion of Olympic Dam and increasing throughput at McLean Lake.

In summary, the extent of economically recoverable uranium, although somewhat uncertain, is clearly linked to exploration effort, technology, and economics but is inextricably linked to environmental costs such as energy, water, and chemicals consumption, greenhouse gas emissions, and broader social issues. These crucial environmental aspects of resource extraction are only just beginning to be understood in the context of more complete life cycle analyses of the nuclear chain and other energy options. There still remains incomplete reporting however, especially in terms of data consistency among mines and site-specific data for numerous individual mines and mills, as well as the underlying factors controlling differences and variability. It is clear that there is a strong sensitivity of energy and water consumption and greenhouse gas emissions to ore grade, and that ore grades are likely to continue to decline gradually in the medium- to long-term. These issues are critical to understand in the current debate over nuclear power, greenhouse gas emissions, and climate change, especially with respect to ascribing sustainability to such activities as uranium mining and milling.

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This paper has been the result of slow accumulation of reports and data over a good while. Numerous companies and government agencies provided reports and data, which is much appreciated. Additionally, further production data for the Rössing mine was supplied by Ian Hore-Lacy (Uranium Information Centre).

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Radon releases from Australian uranium mining and milling projects: assessing the UNSCEAR approach

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Abstract

The release of radon gas and progeny from the mining and milling of uranium-bearing ores has long been recognised as a potential radiological health hazard. The standards for exposure to radon and progeny have decreased over time as the understanding of their health risk has improved. In recent years there has been debate on the long-term releases (10,000 years) of radon from uranium mining and milling sites, focusing on abandoned, operational and rehabilitated sites. The primary purpose has been estimates of the radiation exposure of both local and global populations. Although there has been an increasing number of radon release studies over recent years in the USA, Australia, Canada and elsewhere, a systematic evaluation of this work has yet to be published in the international literature. This paper presents a detailed compilation and analysis of Australian studies. In order to quantify radon sources, a review of data on uranium mining and milling wastes in Australia, as they influence radon releases, is presented. An extensive compilation of the available radon release data is then assembled for the various projects, including a comparison to predictions of radon behaviour where available. An analysis of cumulative radon releases is then developed and compared to the UNSCEAR approach. The implications for the various assessments of long-term releases of radon are discussed, including aspects such as the need for ongoing monitoring of rehabilitation at uranium mining and milling sites and life-cycle accounting. © 2007 Elsevier Ltd. All rights reserved.

Keywords: Uranium mining; Radon; Australia; UNSCEAR

1. Introduction

The exhalation and release of radon gas into the environment are the products of the radioactive decay chain of primordial uranium or thorium, specifically the isotopes ^{238}U , ^{235}U and ^{232}Th . The radon isotopes formed from these decay chains are ^{222}Rn ('radon'), ^{219}Rn ('actinon') and ^{220}Rn ('thoron'), which are the direct decay products of the radium isotopes ^{226}Ra , ^{223}Ra and ^{224}Ra , respectively, in these chains. Due to the low abundance of ^{235}U in natural uranium and the short half-life of actinon (4 s), most work concentrates on ^{222}Rn and its decay progeny since this is the dominant source of exposure. In general, most uranium deposits contain low primary thorium (^{232}Th) and hence thoron (^{220}Rn) is generally considered to be of minor radiological importance. All reference to radon and radium hereafter refers to ^{222}Rn and ^{226}Ra , respectively.

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Radon is a chemically inert noble gas with a half-life of about 3.8 days, while its decay products or progeny of various isotopes of bismuth (Bi), polonium (Po) and lead (Pb) generally forms solids at normal environmental conditions (Cothorn and Smith, 1987). The half-lives of radon progeny vary from microseconds to minutes to years. The rates of radon release are complex and depend on many factors, such as rock mineralogy and structure, the distribution of parent radionuclides (e.g. ^{238}U , and ^{226}Ra), temperature and moisture content (Barretto, 1973; Cothorn and Smith, 1987; Hart, 1986; Lawrence, 2006). The fraction of radon which is released relative to its total production is known as the emanation coefficient, and can range from 0 to 1 but is generally between 0.2 and 0.5 (Flügge and Zimens, 1939).

Due to the natural abundance of about 2.7 mg/kg uranium in soils and rocks (Langmuir, 1997; Titayeva, 1994), there is a global average radon exhalation from soils of about 0.015–0.023 Bq/m²/s (UNSCEAR, 1982). The seasonally-adjusted arithmetic mean radon and thoron exhalation from Australian soils are about 0.022 ± 0.005 and 1.7 ± 0.4 Bq/m²/s, respectively (Schery et al., 1989). The average ^{226}Ra and ^{224}Ra soil activities are 28 and 35 mBq/g, respectively (Schery et al., 1989).

Within the vicinity of a uranium deposit or project, the release rates of radon and activities in air can be elevated over natural background, depending on local conditions and/or project operations. The inhalation or ingestion of significant activities of radon and progeny has long been considered to be related to elevated incidences of lung cancer and other diseases in uranium industry workers (Dalton, 1991; Fry, 1975; NAS, 1980; NAS, 1988; Teleky, 1937).

In recent years there have been some attempts to quantify the long-term ($\sim 10,000$ years) public radiological exposure from the release of radon due to uranium mining and milling operations as part of life-cycle analyses of the nuclear fuel chain. The principal work has been undertaken by the United Nations Scientific Committee on the Effects of Atomic Radiation (UNSCEAR) in their periodic reports to the United Nations General Assembly. The main analysis of radon releases and the associated public radiological exposure over 10,000 years are given in UNSCEAR (1993), with a minor update by UNSCEAR (2000). The UNSCEAR analyses combine other stages of the nuclear fuel chain and present normalised radiological exposures per annual unit of energy generated, summarised in Table 1. The different estimates from the 1993 and 2000 reports are based on criticisms, feedback and the adoption of scenarios perceived to be more realistic for modern uranium mines. Both UNSCEAR estimates suggest that uranium mining and milling, based on the assumption of radon releases from tailings only, are the major factors in long-term public radiation exposure from the nuclear fuel chain, generally comprising between 16% and 75% of the local and global exposures from the nuclear fuel chain. The UNSCEAR (1993) estimate for global exposure

Table 1
Long-term radiological exposure of the nuclear fuel chain (UNSCEAR analyses)

Stage of the nuclear fuel chain	Collective effective dose committed per unit energy generated (person Sv/GWe year)					
	1993	2000	2000	2000	2000	2000
Period		1970–1979	1980–1984	1985–1989	1990–1994	1995–1997
<i>Local and regional component</i>						
Mining, milling and tailings	1.5	0.238	0.238	0.238	0.238	0.238
Fuel fabrication	0.003	0.003	0.003	0.003	0.003	0.003
Nuclear reactor operation	1.3	3.2	0.9	0.46	0.45	0.44
Reprocessing	0.25	8.5	1.9	0.17	0.13	0.12
Transportation	0.1	<0.1	<0.1	<0.1	<0.1	<0.1
Total	3.15	11.94	3.04	0.87	0.82	0.81
<i>Global component (including solid waste disposal)</i>						
Tailings (over 10,000 years)	150	7.5	7.5	7.5	7.5	7.5
Reactors						
Low-level waste	5×10^{-5}	5×10^{-5}	5×10^{-5}	5×10^{-5}	5×10^{-5}	5×10^{-5}
Intermediate waste	0.5	0.5	0.5	0.5	0.5	0.5
Reprocessing solid waste disposal	0.05	0.05	0.05	0.05	0.05	0.05
Globally dispersed radionuclides	50	95	70	50	40	40
Total	200.5	103	78	58	48	48

References: UNSCEAR (1993, Table 53, p. 200) and UNSCEAR (2000, Table 45, p. 284).

from tailings-derived radon was 150 person Sv/GWe year (ranging from 1 to 1000), with the UNSCEAR (2000) estimate being 7.5 person Sv/GWe year.

The radon data and assumptions used by UNSCEAR in their analyses have been questioned by Chambers et al. (1998a,b) and Frost (2000). In general, these authors argue that the UNSCEAR analyses adopt the most pessimistic values and that more realistic radon release scenarios suggest that the exposures are considerably lower. For example, Chambers et al. (1998a,b) argue that the long-term radiological exposure due to radon is 0.96 person Sv/GWe year, considerably lower than the UNSCEAR estimates.

The various analyses noted above, however, are still based on a limited survey of studies and the literature and do not take into proper account the numerous investigations which provide actual field measurements of radon releases from rehabilitated, operating and abandoned uranium projects. The UNSCEAR data used for Australia in particular are reliant on written advice from specific operations and appear to use only a minimal degree of field-measured data.

It is the normal standard of radiation dose management to follow the 'as low as reasonably achievable' or ALARA principle. That is, radiation exposure and doses should be kept to the minimum practicable. In the context of life-cycle analyses of the nuclear fuel chain, and uranium mining specifically, this therefore means the minimisation of public doses during operation and to ensure any changes from baseline radiological conditions following rehabilitation are also minimal, or even potentially beneficial (i.e. a reduction).

For this paper, radon exhalation shall refer to the radon per unit area per time ($\text{Bq}/\text{m}^2/\text{s}$) that enters the environment while radon releases shall be used to specify the mass per time (GBq/d) at which radon enters the environment.

The sources of radon from a typical uranium project are now reviewed followed by a detailed review of radon releases from the various Australian projects compared to pre-mining, where known. The comprehensive data set is then analysed to provide a more systematic basis for the figures used to assess the long-term radiological exposure due to radon as per the UNSCEAR approach. The implications for current uranium projects in Australia are then discussed.

2. Radon source terms

The principal sources of radon at a uranium mining and milling project are uranium ore (including low-grade ore), waste rock, open cuts or underground mines, processing mill, water management ponds and tailings. Sites where contamination has occurred, primarily due to radium, can also be a source of radon. For an *in situ* leach mining site, the dominant radon sources are the processing mill, groundwater bores, solution pipelines and water management ponds. Assuming a project site is effectively rehabilitated, the only change to radon releases is the removal of the mill as a major source and the long-term success of rehabilitation works on tailings, remaining ore, waste rock and contaminated areas. Any analysis of radon releases should therefore assess all of these sources and not just focus on tailings.

The main properties required to quantify radon releases include specific radium activity, material porosity and density, moisture content, and the variation of the emanation coefficient and the radon diffusion coefficient with moisture content. Based on experiments, the radon diffusion coefficient can be calculated from theoretical considerations providing that other variables are known, such as moisture content and porosity (Hart, 1986; Rogers and Nielson, 1981; Strong and Levins, 1982). An alternative approach and model are developed by Rogers and Nielson (1981) using moisture content and pore size distribution to predict radon diffusion rates and overall exhalation.

The United States Nuclear Regulatory Commission estimated the radon source terms for a 'model mill' in the Final Generic Environmental Impact Statement on Uranium Milling (USNRC, 1980). The 'model mill' processed 0.56 Mt ore per year grading 0.10% U_3O_8 to produce 520 t U_3O_8 , it had an ore pad area of 0.5 ha, with a tailings dam area of 50 ha and a dry density of $1.6 \text{ t}/\text{m}^3$ (Table 5-1, pp. 4–5). The analyses suggested that ore stockpiles and crushing facilities would release 6.9 GBq/d of radon, while tailings would release about 446 GBq/d, including a small allowance for dispersed ore and tailings of 4.9 GBq/d (Table 5-5, pp. 5–8).

In Australia, the Ranger Uranium Environmental Inquiry (1975–1977) considered that the main source of radon releases from the Ranger project would be 20–148 GBq/d from the processing mill, about 96 GBq/d from ore stockpiles, between 20 and 281 GBq/d from the open pits and 1.4–14 GBq/d from saturated or water-covered tailings (Fox et al., 1977). The most controversial aspect of radon releases was tailings. Radon data presented to the Inquiry and more recent estimates have ranged from '0' to 4440 GBq/d (Mudd, 2002). There are no published systematic measurements from the Ranger project of all radon sources in one study to verify the Ranger Inquiry predictions.

The exhalation and release of radon from different uranium deposits will vary considerably, depending on local geologic structure and environmental conditions. An important principle in the assessment of radon impacts due to

uranium mining and milling is the change from existing baseline conditions governed by the above, especially given the altered nature of the properties of mined materials compared to *in situ* geology. It is only in more recent decades, however, that pre-mining studies have been undertaken in Australia, although not necessarily as comprehensively as needed for long-term impact assessment.

3. Uranium mining and milling wastes in Australia

3.1. Overview

The mining, milling and export of uranium have been undertaken on a large scale in Australia since 1954 and have gradually expanded to a current annual production of about 11,000 t of uranium oxide (U_3O_8). Small but determined attempts to develop a radium mining industry between 1906 and 1934 failed to lead to commercial uranium production (Mudd, 2005). Most modern uranium mines have been open cut, although some have been underground plus some *in situ* leach or 'solution' mining sites. The currently operating commercial mines are Ranger (open cut), Olympic Dam (underground) and Beverley (acid *in situ* leaching). To date, there has been a total of 11 uranium mills, including pilot projects, and about 31 mines of various scale supplying ore to adjacent or nearby mills or for pilot milling and exploration work. The location of uranium mining and milling sites and other uranium deposits in Australia is shown in Fig. 1, with annual production from 1954 to 2005 in Fig. 2. The quantity of uranium production, ore grades and associated mine wastes is given in Table 2. A compilation of pertinent data for uranium deposits referred to in this paper is given in Table 3.

The management of uranium mill tailings and mine wastes in Australia has changed over the years as regulation of the radiological and environmental hazards has improved and community expectations evolve. During the 1950s in the Northern Territory, tailings and liquid wastes were generally discharged onto adjacent lowland areas which formed part of creek lines and rivers. During the intense rainfall of the tropical wet season, both erosion and water quality impacts were quite significant. In contrast, the mills in arid regions of Queensland and South Australia constructed engineered dams to retain tailings and liquid wastes. From the 1970s it has been a standard regulatory and community preference to use above ground dams for interim management only and to transfer tailings back into a mined out pit as soon as practicable after the completion of mining. Although *in situ* leach mining was tested on a pilot scale in the 1980s using acid leaching at Honeymoon and alkaline leaching at Manyingee, acid leaching has only recently been developed on a commercial scale at Beverley in 2001.

The management of low-grade ore and waste rock has received less attention despite being potentially significant radon sources. In general, these materials have been placed in piles or heaps. At some sites, due to acid mine drainage, the heaps have been rehabilitated with soil covers while at other sites they have or will be covered mainly for erosion and water quality control.

There are very few measurements of radon releases from processing mills in Australia as well as from contaminated areas, water management ponds and active mines (open cut and underground).

3.2. Average tailings data

The data in Table 2 show that the production of each tonne of Australian uranium (as U_3O_8) requires about 848 t of ore and 1152 t of combined low-grade ore and waste rock. The average ore grade is about 0.146% U_3O_8 (range 0.075% to ~2% U_3O_8) with a specific radium activity of 15.2 Bq/g (range 0.56–191 Bq/g; assuming secular equilibrium and minimal radium losses during milling and storage), while the tailings contain residual uranium of about 0.028% U_3O_8 (range 0.02% to ~0.10% U_3O_8).

An important aspect of the UNSCEAR analyses was the average area taken up by tailings, normalised to the area per annual energy output and assumed to be 1 ha/GWe year (UNSCEAR, 1993). This is important due to the slow rates of radon diffusion in tailings. For a given mass of tailings, a thicker tailings pile will allow less radon exhalation into the environment than a thinner but greater area tailings pile. A compilation of the areas and dry densities of the different tailings' piles in Australia are given in Table 4, based on existing, proposed or as-rehabilitated scenarios. The tailings data for Rum Jungle are approximate only (due to conflicting sources).

UNSCEAR adopted a tailings dry density of 1.6 t/m³. In practice, most tailings Australian sites have a density lower than this, such as the above ground dam at Ranger with a density of about 1.0 t/m³ (Li et al., 2001; Sheng

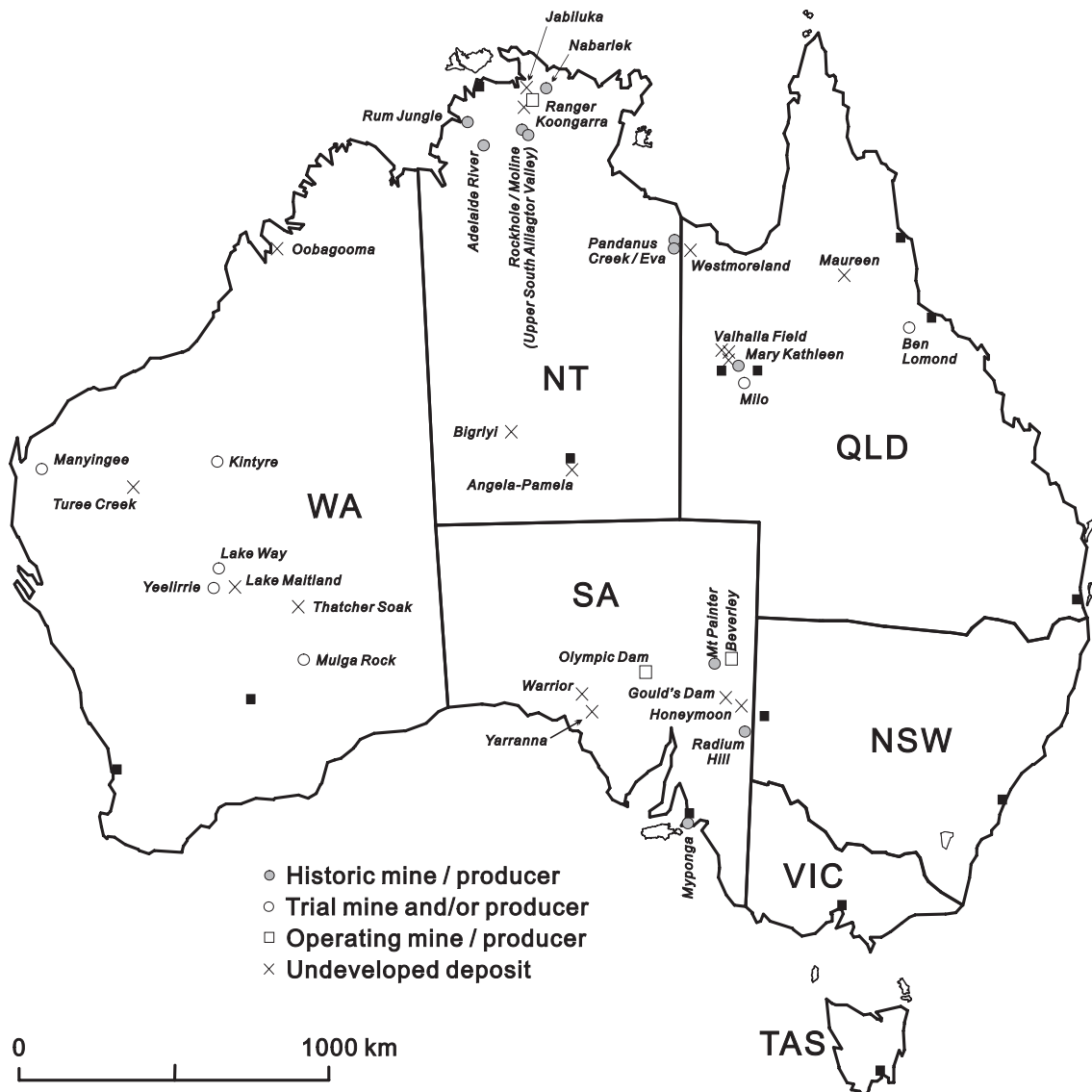


Fig. 1. Location of major uranium deposits in Australia.

et al., 1997) and Pit 1 tailings facility averaging about 1.4 t/m^3 (2005 Edition, ERA, 1984–2005). For Ranger, the tailings particle density is approximately $2.7\text{--}2.8 \text{ t/m}^3$ (Sheng et al., 2000; Sinclair, 2004). The Olympic Dam tailings' dams, however, apparently achieve a higher density ranging from 1.6 to 2.0 t/m^3 and averaging $1.7\text{--}1.8 \text{ t/m}^3$ with tailings particle density ranging from 3.2 to 3.6 t/m^3 (Johnston, 1990; Ring et al., 1998; Waggitt, 1994). The initial tailings density at Nabarlek in the early 1980s was not more than 1.0 t/m^3 (OSS, 1983) but by the time of complete site rehabilitation in 1994, a density of about 1.3 t/m^3 can be estimated based on pit volume, milling rates, and final depths of tailings, waste rock and covers. There is a general lack of tailings density data at older sites, with some of the values in Table 4 either deduced or estimated.

To date, the 123 Mt of Australian uranium mill tailings are estimated to average the UNSCEAR density of 1.6 t/m^3 at a volume of about 78 Mm^3 , and an average depth of the order of 14 m.

Based on the data in Table 4, currently proposed rehabilitation strategies and using the UNSCEAR figure of $250 \text{ t U}_3\text{O}_8/\text{GWe year}$, a normalised tailings production value of 0.95 ha/GWe year can be estimated – virtually the same as

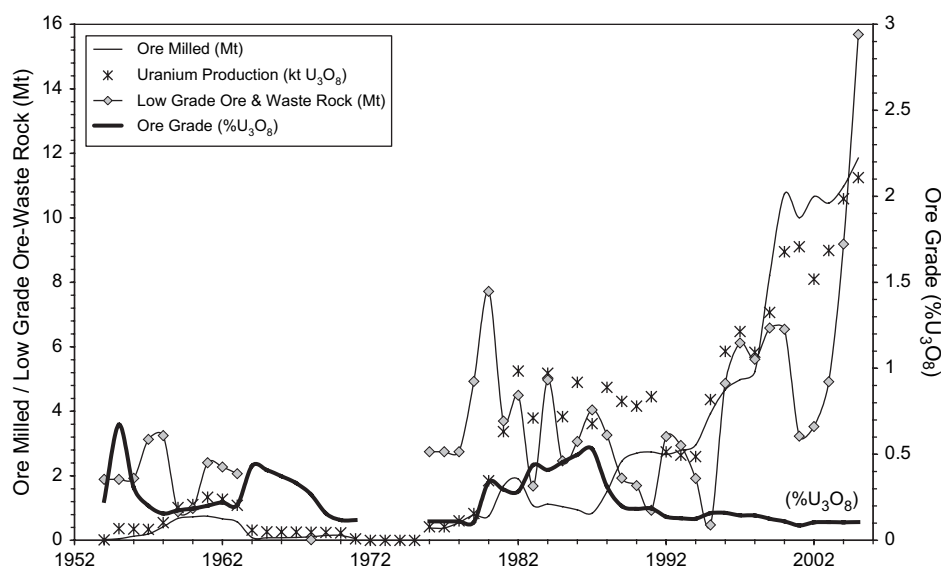


Fig. 2. Annual Australian uranium production statistics 1954–2005.

the UNSCEAR estimate of 1 ha/GWe year. Although rehabilitation works are planned for sites such as Ranger and Olympic Dam, the areal extent of the tailings repositories is difficult to predict given the potential for future expansion at Olympic Dam and evolving extensions to mine life at Ranger. These calculated values are therefore indicative only.

3.3. Average waste rock data

The total amount of waste rock, including low-grade ore, produced by uranium mining in Australia is quantified within a reasonable order of magnitude. Based on data in Table 2, about 175 Mt has been excavated to date (waste rock data for underground and most older mines are generally not available). The most significant sites for waste rock are Ranger, Mary Kathleen, Rum Jungle, Olympic Dam and Nabarlek. In the future, if the proposed expansion of Olympic Dam proceeds, this site alone may contain waste rock covering some 1600–4400 ha (depending on height, at 160 or 60 m, respectively) (BHPB, 2005).

The average uranium grades of the various waste rock piles are generally not available, though some data exist for Ranger, Nabarlek and Rum Jungle as compiled in Table 5. It can only be assumed that waste rock at other sites contains <0.02% U_3O_8 . The quantity of waste rock is primarily due to Ranger and Mary Kathleen, and to a lesser extent by Rum Jungle and Olympic Dam.

Overall, the 1152 t of combined low-grade ore and waste rock produced per tonne of Australian uranium can be expected to have a grade between 0.01% and 0.03% U_3O_8 . The average mass is about 519 kt/ha, and using a typical waste rock density of 2 t/m³, this gives an expected height of about 26 m.

4. Estimated and measured radon exhalation and releases

The measurement of radon exhalation has only been undertaken in more recent decades, commensurate with improved understanding of radon management in uranium mining and milling. Many of the recent radon studies were undertaken as part of an Environmental Impact Statement (EIS) or to support technical aspects of a project's design (e.g. radiation protection for mine workers). There is still, however, a lack of comprehensive radon exhalation and release studies at most former and current uranium project sites in Australia. Most studies only report exhalation data and do not measure (or at least do not report) other important variables such as porosity, moisture content and measured or calculated radon diffusion coefficients, or the area and grade of the active radon source.

Table 2
Principal ore, tailings and waste data for Australian uranium mines and mills to December 2005 (Mudd, 2007)

	Operation	Ore milled (t)	Ore (%U ₃ O ₈)	Prod. (t U ₃ O ₈)	Tailings (%U ₃ O ₈)	Tailings ²²⁶ Ra ^a	Low-grade ore and waste rock (t)	Other metals produced/ ores mined (±milled) ^b
Olympic Dam, SA	1988–2005 ^j	85,396,312	0.075	41,234	~0.026 ^c	7.65	~10,250,000	1957 kt Cu, 25.2 t Au, 253 t Ag
Ranger, NT	1981–2005 ^j	30,772,000	0.310	85,121	0.033	32.1	~121,150,000	—
Nabarlek, NT	1980–1988	597,957 157,000 ^d	1.84 0.05–0.1	10,955	0.036 ~0.02 ⁱ	191.1 5.2	2,330,000	—
Beverley, SA ^e	2001–2005 ^j 1998 ^h	~31,750 ML ^f 153 ML ^h	~0.18	4070 33.27 ^h	—	—	601 ML 2.686 ML ^h	—
Honeymoon, SA ^e	1982 ^h 1998–2000 ^h	(ISL ^e)	~0.12	No data 29.4 ^h	—	—	41.2 ML	—
Mary Kathleen, QLD	1976–1982	~6,200,000	0.10	4801	~0.02	10.4	17,571,000	—
Small/pilot Mines	1970–1980	Various		≥12 ^h	—	—	≥150,000 ⁱ	—
Moline, NT	1956–1964	135,444	0.46	716.0	0.070	47.5	Unknown	152.6 kt CuAu and PbZnAg ore
Rockhole, NT	1959–1962	13,418	1.11	139.7	0.066	115.3	Unknown	—
Mary Kathleen, QLD	1958–1963	2,668,094	0.172	4091.8	~0.019	16.2	4,539,652	—
Radium Hill, SA	1952–1961	822,690	0.119	—	~0.02	0.52	Unknown	—
Port Pirie, SA	1955–1962	~153,400	~0.74	852.3	~0.10	76.8	Unknown	1500 t monazite
Rum Jungle, NT	1954–1971	1,496,641	0.35	3530	0.086	33.7	~18,027,000	2.6 Mt Cu ore/87 kt Pb ore
Small/pilot Mines ^g	1950s–1960s	9225 ^g	0.92	— ^g	— ^g	~95.5	Unknown	—
Mt Painter, SA	1910–1934	~933	~2.1	~3 t ⁱ	—	—	Unknown	—
Radium Hill, SA	1906–1932	>2150	~1.4 ⁱ	~7 t ⁱ	—	—	Unknown	—
Total		~128.4 Mt	0.146%	155,595	0.028%	15.2	~175 Mt	—

^a ²²⁶Ra in Bq/g based on measured data or assuming secular equilibrium and average ore grade.

^b Such as base metal or other ores milled (e.g. copper at Moline, thorium/monazite at Port Pirie; though the Rum Jungle lead ore was not milled).

^c Adjusted for coarse backfill and copper extraction (based on 94.6% of ore milled as tailings and assuming no uranium in coarse backfill).

^d Low-grade ore experimentally heap leached.

^e ISL involves chemical solutions only (in ML) and no physical extraction of ore.

^f Includes some estimated data.

^g Ore milled at Rum Jungle ('RJ'), not included in sub-totals.

^h Pilot plant only.

ⁱ Data uncertain (approximate only).

^j Still operating at end of 2005.

Table 3
Resources and dimensions of major uranium deposits in Australia (adapted from Mudd, 2007, and additional references)

Deposit	Resources			Approximate (or average) dimensions (m)				Additional references
	Ore (Mt)	(%U ₃ O ₈)	(t U ₃ O ₈)	Depth ^a	Length	Width	Thickness	
Honeymoon, SA	2.75	0.12	3300	100–120	1000	400	4.3	SCRA (2000)
Beverley, SA ^b	~10.4	0.18	~17,900	100–120	~4000	400–750	20–30	HR (1998)
Olympic Dam, SA ^c	3970	~0.04	~1,500,000	350	~5000	~400–2300	~400	Kinhill (1997)
Ben Lomond, QLD	2.98	0.23	6800	50–75	750	150	100	McKay and Mieizitis (2001)
Ranger 1, NT ^d	19.78	0.321	63,500	1–20	500	300	~185	Kinhill and ERAES (1996), McKay and Mieizitis (2001)
Ranger 3, NT ^e	53.0	0.16	~85,000	~20–30	900	500	~25–100	McKay and Mieizitis (2001), Needham (1988)
Nabarlek, NT	0.76	1.84	10,955	2–5	230	10	85	Needham (1988)
Jabiluka 1, NT	1.36	0.25	3400	~25	350	225	Up to 35	Bathey et al. (1987), McKay and Mieizitis (2001)
Jabiluka 2, NT ^f	31.1	0.53	163,000	~80–120	1100	400	Up to 135	Bathey et al. (1987), McKay and Mieizitis (2001), Needham (1988)
Koongarra 1, NT	1.83	0.8	14,550	2–25	450	~30–100	100	Hegge et al. (1980), Needham (1988)
Koongarra 2, NT	0.77	0.3	2300	50–250	100	~30–100	Up to 200	Hegge et al. (1980)
Coronation Hill, NT	0.34	0.54	1850	~150	No data	No data	No data	McKay and Mieizitis (2001)
Lake Way, WA	5.98	0.09	5200	2–10	~3000 ^g	~2000 ^g	1.5	McKay and Mieizitis (2001)
Yeelirrie, WA	35.2	0.15	52,500	2–8	~9000 ^g	Up to 1500 ^g	3–4	McKay and Mieizitis (2001)

^a Average depth to start of economic mineralisation.

^b Adjusted from resource prior to mining, after allowing for production of 3103 t U₃O₈.

^c Resources at June 2005, excluding milled ore of 85.4 Mt at 2.62% Cu, 0.075% U₃O₈, 5.9 g/t Ag and 0.55 g/t Au.

^d Completely mined and milled.

^e Includes reserves and resources (December 2005) but not milled ore derived from Ranger 3 (~10.9 Mt at 0.20% U₃O₈).

^f Mineralisation extends to depths of 600 m, possibly deeper (possible ore zone extensions are still untested to the east and south of the deposit).

^g Mineralisation not continuous over this area.

The variation in the radon emanation coefficient with moisture content for Ranger and Jabiluka ores and laboratory tailings is shown in Fig. 3. Further studies on radon behaviour are given by Hart (1986), Lawrence (2006), Storm (1998), Strong and Levins (1982), and Todd (1998).

4.1. Pre-mining radon exhalation

The available pre-mining radon exhalation surveys are compiled in Tables 6 and 7. The pre-mining radon exhalation contours for the Koongarra and Yeelirrie deposits are shown in Figs. 4 and 5, respectively, with the pre-mining radon activity in soil at Nabarlek shown in Fig. 6. In general, it is only uranium deposits of sufficient size and which appear from a shallow depth that give rise to a significantly elevated radon exhalation at the surface (comparing Tables 3, 6 and 7). Some examples are the calcrete–carnotite deposits in Western Australia (Yeelirrie, Lake Way) and the unconformity deposits at Ranger and Nabarlek in the Northern Territory. Conversely, there is no significant mineralisation-related radon signature from Olympic Dam, Beverley, Jabiluka and others.

The use of radon techniques in uranium exploration has been performed in Australia, most notably at the Rum Jungle mineral field, NT (Stewart, 1968), at Yeelirrie, WA (Severne, 1978) and the Alligator Rivers Region, NT (Gingrich and Fisher, 1976), though it does not appear to have been widely adopted and is thus of limited use in the context of this paper.

4.2. Radon sources during open cut, underground, in situ leach mining

There are only scattered data on the exhalation and release of radon from either underground or open cut uranium mining (Table 8). The EIS estimates for some proposed mines are also included for comparison.

A detailed study of radon releases from underground uranium mines in the United States was given by Jackson et al. (1981), with further analyses by Hans et al. (1981). The dominant radon sources were ventilation shafts with

Table 4
Uranium mill tailings pile data for Australian projects to December 2005

Project	Tailings facility	Area (ha)	Mass ^a	Dry density	Volume	Depth	References
Radium Hill	No. 1 Dam	~ 8	~ 100,000 t	Unknown	Unknown	~ 2 m (?)	Hill (1986), Sheridan and
	No. 2 Dam	~ 32	723,000 t			~ 5 m (?)	Hosking (1960), Waggitt (1994)
Port Pirie	Surface dam	~ 30	151,550 t	Unknown	Unknown	~ 2 m (?)	Waggitt (1994), Wilkinson (1977)
Rum Jungle ^b	Surface deposition minus erosion ^f	34	~ 576,000 t	~ 1.7 t/m ³	~ 0.34 Mm ³	~ 1.0 m	DNT (1978), Kraatz (1998),
	In-pit (White's)	11	~ 600,000 t	~ 0.6 t/m ³ (?)	~ 1.0 Mm ³	No data	Kraatz and Applegate (1992)
	In-pit (Dyson's)	6	~ 500,000 t	~ 2.3 t/m ³ (?)	~ 0.22 Mm ³	No data	
Mary Kathleen	Surface dam	29	~ 8,900,000 t	~ 1.4 t/m ³ (?)	~ 6.4 Mm ³	~ 22 m (?)	MKU (1986), Ward (1985)
Rockhole ^c	Surface deposition minus erosion ^f	~ 2	~ 12,000 t	Unknown	Unknown	—	Waggitt (1994)
Moline ^d	Surface deposition minus erosion ^f	18	~ 202,000 t	~ 1.2 t/m ³	~ 0.188 Mm ³	~ 1.0 m	Bastias (1987), Waggitt (1994)
	Surface dam (as rehabilitated)	~ 6	~ 208,000 t	No data	No data	No data	
Nabarlek	In-pit (including heap leach wastes)	5	744,000 t	~ 1.3 t/m ³	~ 0.47 Mm ³	<65 m	Bailey (1989)
Ranger	Interim surface dam (to Pit #3) ^e	117	13,624,000 t	1.0 t/m ³	13.6 Mm ³	11.6 m	ERA (1984–2005), Li et al. (2001),
	In-pit (Pit #1)	51	~ 18,951,000 t	~ 1.38 t/m ³	~ 13.7 Mm ³	<150 m	Sheng et al. (2000), Sheng et al. (1997)
	In-pit (Pit #3) ^e	~ 75 ^e	Not applicable	—	—	—	
Olympic Dam	Current surface dam	~ 750	~ 78,500,000 t	1.75 t/m ³	~ 45 Mm ³	~ 5.9 m	Mudd (2007)
	Proposed final dam	~ 1850	Up to ~ 4.1 Gt	—	—	<30 m	BHPB (2005)
Approximate total (Dec. 2005)		1046	123.01 Mt	~ 1.6 t/m ³	—	~ 14 m	

^a Allows for extraction of uranium, base metals and removal of the coarse fraction where appropriate, though in general the reagents added during milling equals the mass removed (e.g. pyrolusite and acid).

^b Data on tailings in the pits at Rum Jungle are very poor, data as used are approximate only. The surficial tailings were dumped in Dyson's open cut during rehabilitation.

^c About half of the Rockhole tailings were removed and transported to Moline for reprocessing and emplacement in the mid 1980s.

^d The Moline tailings were excavated, reprocessed for gold and emplaced in a new engineered dam in 1986–1987, including about 6000 t of tailings transported from Rockhole. Data include base metal tailings (due to mixing with uranium tailings). After this project, a medium-size gold project was undertaken during 1988–1992 (Moline Hill, see Anon., 1988; Miller, 1990), depositing some 2.3 Mt of gold tailings over the old uranium-base metal tailings.

^e Expected quantity of tailings for Ranger's Pit #3, including the interim above ground dam, is of the order of 38 Mm³ (depending on final mine plan but excluding Jabiluka).

^f Removed during rehabilitation works.

Table 5
Waste rock data for selected Australian projects (Mudd, 2007)

Project	Deposit/mine	Low-grade ore ^a		Waste rock		Total area (ha)
		(Mt)	(%U ₃ O ₈)	(Mt)	(%U ₃ O ₈)	
Rum Jungle	White's	—	—	8.64	0.004	26.4
	Dyson's	0.0478	0.077	2.032	0.005	8.43
	Rum Jungle Creek South	0.116 ^b	0.066	4.877	0.018	21.9
	Mt Burton	0.0035	0.072	0.254	—	3.28
	Mt Fitch	—	—	0.020	—	~0.5?
	Intermediate (Cu)	—	—	1.727	0.005	6.85
Nabarlek	Nabarlek	0.157	~0.05	2.33	0.013	6
Ranger	Ranger #1 ^c	16.219	~0.075	22.338	<0.02	~200
	Ranger #3	>18.813	~0.070	>9.865	<0.02	
Olympic Dam	Olympic Dam	—	—	~10.25 ^d	—	—
Mary Kathleen	Mary Kathleen (1956–1963)	0.566	—	3.864	—	64
	Mary Kathleen (total)	—	—	~22 ^e	—	
Totals		>35.92	~0.072%	~81.832	~0.01%	~340

^a Generally defined as >0.02% U₃O₈.

^b Apparently processed at Rum Jungle between 1969 and 1971.

^c Conflicting data exist — one estimate states that for Ranger #1 a total of 19.8 Mt of ore, 4.5 Mt of low-grade ore (~0.05–0.10% U₃O₈) and 55.5 Mt of waste rock and very low-grade ore (~0.02–0.05% U₃O₈) were mined (ERA, 1999).

^d Waste rock is returned underground as backfill (though a small stockpile may exist at the surface in the short term).

^e Total of low-grade ore and waste rock from 1956 to 1982.

only a minor contribution from waste and ore stockpiles, mine water and subtracting credit for background radon. Jackson et al. (1981) estimated a normalised radon release at 1088 GBq/t U₃O₈. An important aspect of these studies is the relationship demonstrated between radon releases and cumulative production, with older mines (higher total production) showing higher radon releases relative to younger mines.

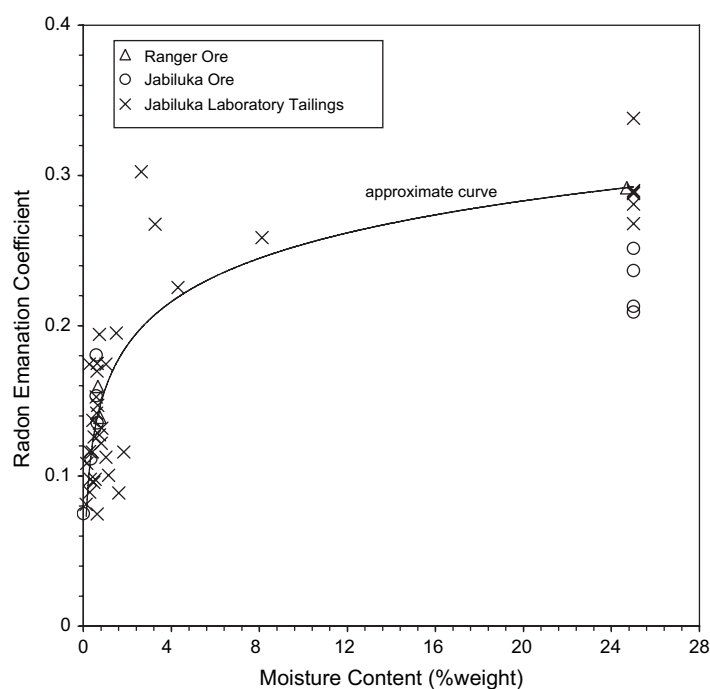


Fig. 3. Effect of moisture content on the emanation coefficient of Ranger ore and Jabiluka ore and laboratory tailings (Hart, 1986; Strong and Levins, 1982) (25% moisture assumed for saturated samples).

Table 6
Pre-mining and/or background radon exhalation and release surveys – Northern Territory

Location	Period or date of survey	Area (ha)	Exhalation (Bq/m ² /s)	Release (GBq/d)	References
Kakadu region – average ^a	Throughout 1998	—	0.030	—	Auty and du Preez (1994),
Kakadu region – range ^a	Various 1992–1998	—	0.009 → 0.057 ^b	—	Todd (1998)
Kakadu–Magela Creek	August 2003 (31 samples)	—	0.21 ± 0.02	—	Lawrence (2006)
Kakadu–Mudginberri	April and Sept. 2003 (44 samples)	—	0.035 ± 0.02	—	
Kakadu–Mirray	March 2003 (45 samples)	—	0.039 ± 0.02	—	
Kakadu–Jabiru Water Tower	March and August 2003 (46 samples)	—	0.018 ± 0.01	—	
Kakadu–Jabiru East	August 2003 (45 samples)	—	0.043 ± 0.02	—	
Jabiluka 2 ^g	Sept.–Dec. 1992	—	0.046	—	Auty and du Preez (1994)
Jabiluka Decline (east of #2)	Nov. 1992 and July–Aug. 1993	—	0.025	—	
Koongarra 1 ^g	June 1978	12.53 ^c	2.43 ^c	26.1	Davy et al. (1978)
Koongarra 2 ^g	June 1978	—	<0.05	—	Davy et al. (1978)
Nabarlek ^g	Sept. 1978	5	3.7 → 44.0 ^d	—	Clark et al. (1981)
	June 1979		11.5 → 164.0 ^{d,e}	—	
Nabarlek region	1999–2002	—	0.016 → 0.049 0.031 (average)	—	Bollhöffer et al. (2006)
Ranger 1 ^g	~ March 1978 ^f	43 ^f 91 ^f	3.8 ^f 2.5 ^f	~ 141 ^f ~ 197 ^f	Haylen (1981) ^f
Ranger 1–3 region ^g	(Calculated estimate)	245	1.78	377	Kvasnicka and Auty (1994)
Ranger 1 ^g		44	4.1	156	
Ranger 3 ^g		66	2.5	143	
Area around 1 and 3		81	1.0	70	
Australian background	—	—	0.022 ± 0.005	—	Schery et al. (1989)

^a Primarily in the near vicinity of the Ranger project area.

^b Values >0.06 Bq/m²/s were detected above known as mineralisation (e.g. Ranger 2), ranging from 0.096 to 0.280 Bq/m²/s (three points excluded from average of 18 measurements).

^c Average ²²²Rn exhalation for 5.29, 3.69, 2.57, 0.79 0.13 and 0.063 ha is 0.57, 2.02, 4.07, 8.15, 13.18 and 20.76 Bq/m²/s, respectively.

^d Range given as minimum and maximum values only (no average).

^e Vegetation cleared in preparation for mining.

^f The AAEC report on this Ranger radon survey was apparently never completed. The data quoted are cited by Haylen (1981, p. 100) (Haylen worked for the AAEC in the late 1970s as a geologist). Further reference to this AAEC study is made in radon studies at Koongarra (Davy et al., 1978, p. 5), broader radiation studies at Nabarlek (Clark et al., 1981, p. 24; Davy, 1978, p. 78), as well as Yeelirrie, WA (Brownscombe and Davy, 1978, p. 14) while NTDME (1981, p. 8) also quotes the AAEC data.

^g Above uranium deposit.

A difficult issue is estimating the actual radon released by *In Situ* Leach (ISL) mines, as currently in use at Beverley. The releases could be lower from ISL than conventional mining due to the lack of tailings and ore stockpiles, however, it is also likely that during operation the releases would be above normal baseline for the equivalent region being mined. An empirical model for estimating radon releases from ISL facilities was developed by Brown and Smith (1981), based on limited field sampling at an operational ISL mine. It was asserted that almost all of the radon released could be accounted for from the processing mill (99.95%) with a minor component from liquid waste storage ponds (0.05%). The well heads and waste scale buildup (e.g. calcite for their alkaline ISL project) were considered to be effectively ‘zero’. The normalised radon release was estimated at 54 GBq/t U₃O₈, considerably lower than the 1088 GBq/t U₃O₈ estimate for underground uranium mining. Conversely, it was also estimated by Brown (1981) that an ISL mine has a normalised release rate of 143 GBq/t U₃O₈ (the discrepancy is unexplained).

4.3. Radon from ore, waste rock and low-grade ore stockpiles

As noted earlier, there is an increasing stockpile of ore, Waste Rock and Low-Grade Ore (WR–LGO) being produced in Australia. The available data for radon exhalation and releases are compiled in Table 9.

Table 7
 Pre-mining and/or background radon exhalation and release surveys — South Australia and Western Australia

Location	Period or date of survey	Area (ha)	Exhalation (Bq/m ² /s)	Release (GBq/d)	References
Honeymoon ^d	April–June, 1980	—	0.033	—	Whittlestone (1980)
	1998	—	0.038	—	SCRA (2000)
Beverley ^d	1980	—	0.044	—	AMDEL (1982)
Paralana Hot Springs ^a	1980	—	10.6	0.54	AMDEL (1982)
Olympic Dam ^d	June 1991–May 1992	—	0.025 0.005 → 0.035	—	WMC (1992)
Yeelirrie ^d	November 1976	—	3.7	2159	WMC (1978b)
	1981	675	0.5 → 8	—	Leach et al. (1983)
Yeelirrie — regional background	Early 1980s (various)	—	0.05 → 3.5	—	O'Brien et al. (1986)
	November 1976	—	~0.74	—	WMC (1978b)
Lake Way					
Inner mine area ^{b,d}	4–17 September 1979	310	0.30	80	Casteleyn et al. (1981)
Outer mine area ^{c,d}		390	0.126	42	
Regional background		—	0.044	—	
Australian background	—	—	0.022 ± 0.005	—	Schery et al. (1989)

^a Approximately 15 km west of Beverley.

^b Distance of 0–2 km.

^c Distance of 2–3 km from centre of proposed operations.

^d Above uranium deposit.

As can be expected, there is a notably wide variation in the radon exhalation and releases from waste rock, low-grade and ore stockpiles. Some data may not be reliable, as the values seem either too high or low (e.g. trial ore stockpile at Yeelirrie). Another example is Rum Jungle, where although a rehabilitation standard of 0.14 Bq/m²/s was adopted, there was apparently no survey following rehabilitation works (1982–1986). At Yeelirrie, barometric

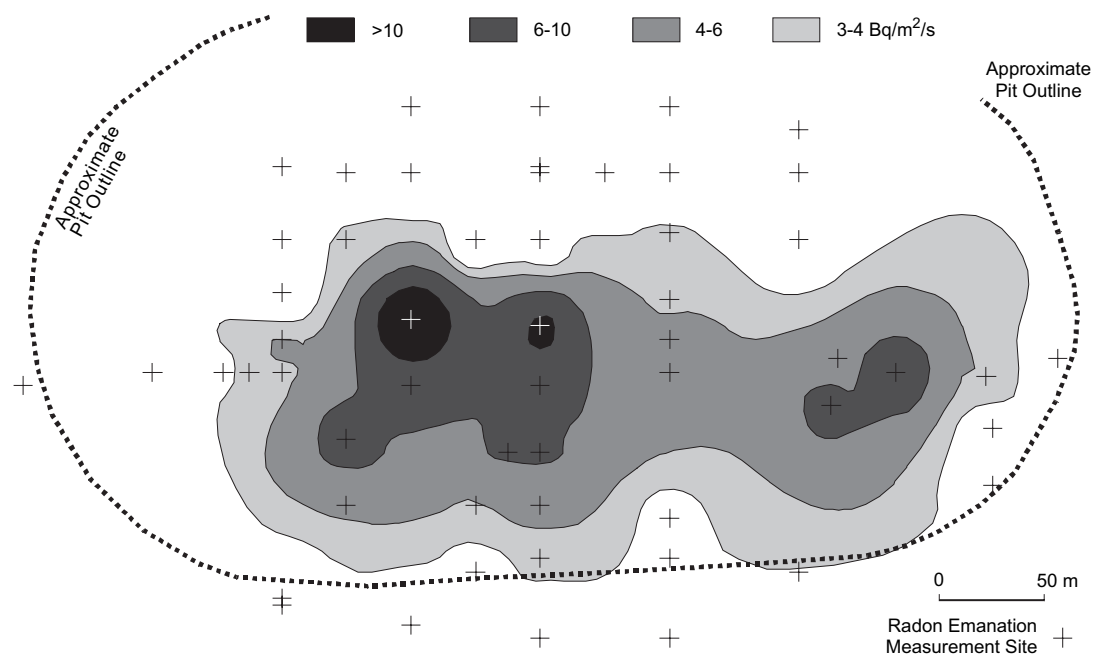


Fig. 4. Pre-mining radon exhalation measured at the Koongarra 1 uranium deposit, 1978 (mBq/m²/s) (redrawn and adapted from Davy et al., 1978).

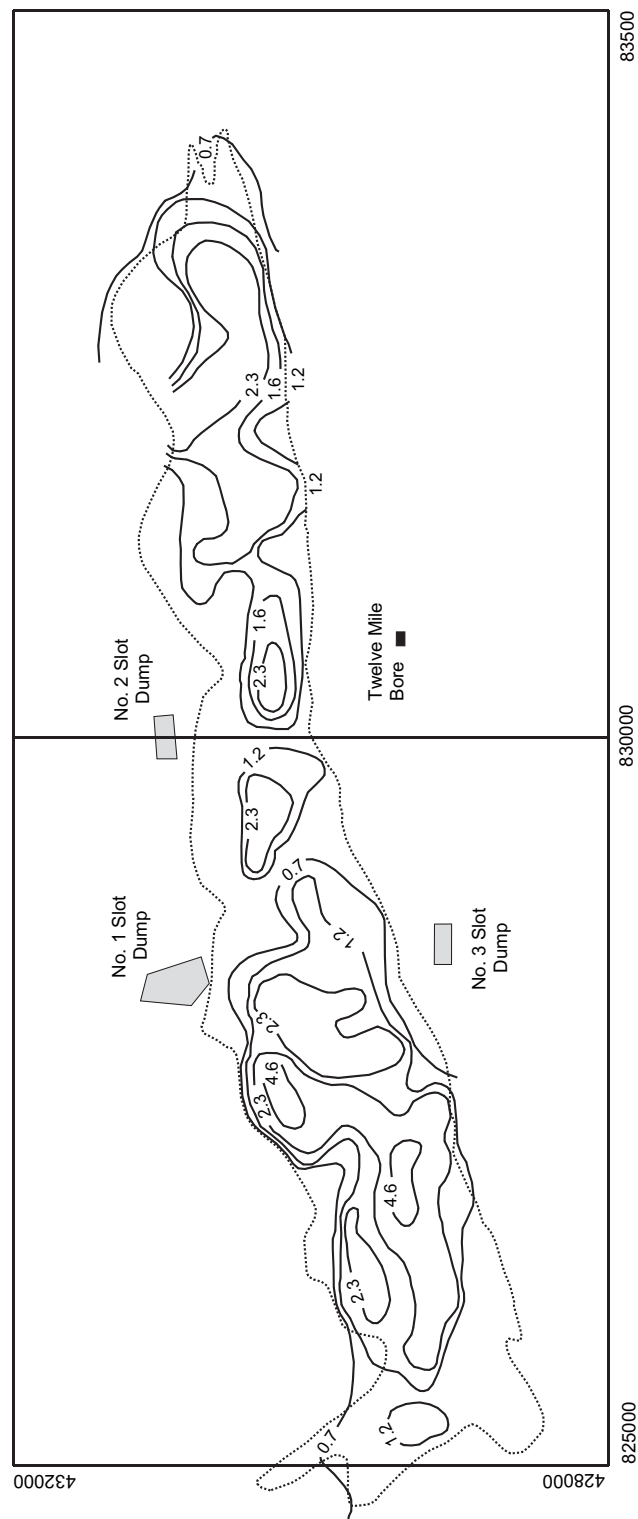


Fig. 5. Pre-mining radon exhalation measured at the Yeelirrie uranium deposit, June 1981 (Bq/m²/s) (redrawn from Leach et al., 1983).

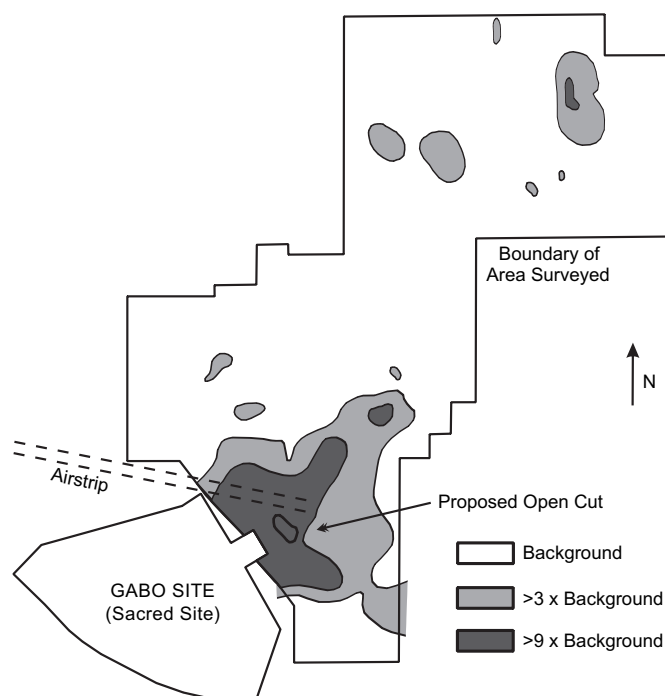


Fig. 6. Pre-mining radon activity in soil at Nabarlek (redrawn from QML, 1979).

pressure effects on radon exhalation have also been noted on early studies of the trial mine stockpiles (Brownscombe and Davy, 1978). The comprehensive study of the tropical Alligator Rivers Region by Lawrence (2006) shows clear seasonal behaviour in radon exhalation from waste rock dumps, related to the monsoonal wet season and resultant soil moisture (similarly, seasonal effects for radon activity in air have been noted earlier by Morley, 1981).

The effectiveness of rehabilitation works, such as engineered soil covers, could be expected to reduce radon exhalation somewhat though the sparse data are not convincing. For example, the study by Lawrence (2006) included radon exhalation measurements on an unnamed waste rock dump ($<0.02\%$ U_3O_8) and included a rehabilitated section. The radon exhalation was similar on both parts of the waste rock dump (see Table 9). Additionally, the study showed that radon exhalation cannot be expected to follow ore grade as the lower grade stockpile (the two stockpile, grade $0.02\text{--}0.08\%$ U_3O_8) had a higher flux than the ore stockpile (the seven stockpile, grade $>0.5\%$ U_3O_8).

The radon released from normalised WR–LGO produced per GWe year could be based on previous mining data (i.e. 280 kt at $\sim 0.02\%$ U_3O_8 and 26 m high). Further discussion of waste rock and low-grade ore stockpiles is included in Sections 4.7 and 5.

4.4. Radon from milling

During the milling of uranium ore, radon can be released from dust, ore grinding, leach solutions, calcining and product packaging areas. To date, only total estimates for radon releases from mills have been made, almost entirely for EIS purposes for recent uranium projects. There still appears to be a lack of field measurements of radon releases from processing mills to verify EIS predictions. The available data are compiled in Table 10.

4.5. Radon from uranium mill tailings

One of the most significant (and controversial) sources of radon from uranium mining and milling, both during operation and after rehabilitation, is that from mill tailings. The predictions for radon exhalation and releases have varied significantly, depending on the chosen tailings management regime, although estimates for the same regime can also differ.

Table 8

Radon exhalation and releases from abandoned, operating, rehabilitated and proposed open cut and underground mines

Site	Source and conditions	Period of survey	Grade (%U ₃ O ₈)	Area (ha)	Exhalation (Bq/m ² /s)	Release (GBq/d)	References
Ranger	Pit #1 – wall (three samples)	Oct. 2003	—	—	0.30 ± 0.05	—	Lawrence (2006)
	Pit #1 – bench (33 samples)				0.50 ± 0.05		
	Pit #3 – rocks (two samples)				1.0 ± 1.0		
	Pit #3 – pad (25 samples)				2.5 ± 0.6		
	Pit #3 – rubble (nine samples)				1.7 ± 0.7		
Jabiluka	Calculated estimate (underground mine)	~1996 (for EIS)	—	—	—	121	Howes (1997)
	Decline and mining cross-cuts	July–Aug. 1999	1.15	—	~17.3	—	Sonter (2000)
Coronation Hill	Old mining tunnel (adit)	Late 1980s	—	—	0.036 ± 0.057	—	DM (1988)
	Abandoned open cut mine				0.67 ± 0.46		
Yeelirrie	Open pits (operating) (proposed)	1978 EIS est.	—	—	~4.7	2463	WMC (1978b)
	Open pits (post-mining) (proposed)	1978 EIS est.	—	605.6	~1.2	602	WMC (1978b)
	Open pits (operating) (proposed)	1979 EIS est.	—	—	—	1918	WMC (1979)
Koongarra	Open pit mine (proposed)	1978 EIS est.	—	—	—	23–57	Noranda (1978)
Olympic Dam	Underground mine (operating)	1980–81	—	—	0.3 → 1 (avg) → 3	—	Kinhill (1982)
	Underground mine (proposed)	1982 EIS est.	—	—	—	700	Kinhill (1982)
	Underground mine (operating)	Jun 1992–May 1993	~0.083	—	—	120	Davey (1994)
	Underground mine (operating)	~1996	~0.08	—	—	121	Howes (1997)
Ben Lomond	Open pit mine (proposed)	1979 EIS est.	—	—	—	22.9	Minatome (1979)
	Underground mine (proposed)					38.4	
Ben Lomond	Pit – exposed ore (proposed)	1983 EIS est.	—	1	10	8.6	Minatome (1983)
	Pit – barren rock (proposed)		—	10	0.3	2.6	
	Underground mine (proposed)		—	—	—	3.2	

The available data for tailings-derived radon are compiled in [Tables 11 and 12](#), including the sites where some rehabilitation works have been undertaken to date. The radon exhalation contours at the former Moline and Rockhole tailings are shown in [Fig. 7](#). In 1986, half of the Rockhole tailings were excavated and transported to Moline, which were also re-excavated with all tailings emplaced within a new gold tailings dam ([Mudd, 2000](#)). There is no known radon exhalation survey at Rockhole or Moline since this time. Further to this, there are no known radon exhalation surveys for the Radium Hill tailings ([McLeary, 2004a](#)) nor publicly available for the Mary Kathleen tailings (they were undertaken but remain confidential).

The efficiency of water covers in reducing radon exhalation from tailings was a central issue during the Ranger Uranium Environmental Inquiry ([Fox et al., 1977](#)), and remains a subject of some conjecture. For example, [Chambers et al. \(1998a\)](#) state that the radon released from Ranger's tailings to be 'zero', while other estimates for water covers have ranged between 7.4 ([Fox et al., 1977](#)) and 288 GBq/d ([Davy, 1983](#)), depending on the depth of water cover assumed. In the early years of operation, [Davy \(1983\)](#) estimated that exhalation from a 2-m water cover would be 0.8 Bq/m²/s, arguing on overall environmental and economic grounds for dry tailings to achieve a radon exhalation of 0.5 Bq/m²/s. The significant difference between these estimates is due to the different regimes used for assessment and the assumptions adopted for the estimate, with some clearly being too optimistic (such as the 'zero') while others appear more reasonable. To date, however, there is no public data on the field-measured radon exhalation from water over the tailings facilities at Ranger (which currently cover about 150 ha).

Studies in Brazil have shown that approximately one third of the radon in mine water retention ponds is released to the atmosphere ([Paschoa and Nóbrega, 1981](#)). Based on laboratory column studies, [Rogers and Nielson \(1981\)](#) argued that the water covers on mill tailings facilities were a major radon source, and presented a model to estimate such releases. Using this model, as implemented by [Diehl \(2006a\)](#) and using Ranger's 1996 tailings configuration (1996 Edition, [ERA, 1984–2005](#)), a total radon exhalation of 3.01 Bq/m²/s can be calculated for a release of 296 GBq/d from the above ground tailings facility (allowing for the tailings area to be 60% under water >1 m, 10% saturated and 30% unsaturated) (additional data for the calculation sourced from [Hart, 1986](#); [Kvasnicka, 1986](#)).

Table 9

Radon exhalation and releases from abandoned, operating, rehabilitated and proposed ore stockpiles and waste rock stockpiles

Site	Source and conditions	Period of survey	Grade (%U ₃ O ₈)	Area (ha)	Exhalation (Bq/m ² /s)	Release (GBq/d)	References
Rum Jungle	White's waste rock dump (12 points)	Dry season 1981	0.01	26.37	1.1	25	Mason et al. (1982)
	RJCS waste rock dump (36 points)	Dry season 1981	0.054	15	2.7	35	Mason et al. (1982)
	Proposed rehabilitation	—	—	—	0.14	—	Allen and Verhoeven (1986)
Nabarlek	Ore stockpile (prior to cover)	~ Oct. 1979	1.86	2.9	130	326	Leach et al. (1982)
	Ore stockpile (after cover)	~ Nov. 1979	—	—	38	95	Leach et al. (1982)
	Waste rock dump (20 points)	Dry season 1981	0.013	—	0.26	—	Mason et al. (1982)
Ranger	Waste rock dump (WRD) (unspecified)	~ 1989	—	—	—	18.0	Kvasnicka (1990)
	Waste rock dump (unspecified)	Jan.—May 1995	—	—	0.47	—	Kvasnicka and Auty (1996)
	Waste rock dump (unspecified)	Sept. 1996	—	—	0.519	—	Todd (1998)
	Tailings dam wall (low-grade ore)	Dry season 1981	0.013	—	0.21	—	Mason et al. (1982)
	Laterite stockpile — pad (20 samples)	August 2002	—	—	5.2 ± 0.6	—	Lawrence (2006)
	Laterite stockpile — push (seven samples)	August 2002	—	—	81 ± 15	—	Lawrence (2006)
	Laterite stockpile — rim (13 samples)	August 2002	—	—	38 ± 5	—	Lawrence (2006)
	Ore stockpile 2 — pad (15 samples)	Sept. 2002	0.02–0.08	—	10 ± 2	—	Lawrence (2006)
	Ore stockpile 2 — rim (10 samples)	Sept. 2002	0.02–0.08	—	7.3 ± 2.2	—	Lawrence (2006)
	Ore stockpile 7 — pad (nine samples)	July 2002	>0.5	—	3.1 ± 0.7	—	Lawrence (2006)
	Ore stockpile 7 — rim (eight samples)	July 2002	>0.5	—	0.95 ± 0.35	—	Lawrence (2006)
	Ore stockpile 7 — push (five samples)	July 2002	>0.5	—	1.7 ± 0.7	—	Lawrence (2006)
	WRD — pad (20 samples)	July 2002	<0.02	—	0.53 ± 0.1	—	Lawrence (2006)
	WRD — rehabilitated (21 samples)	July 2002	<0.02	—	0.94 ± 0.1	—	Lawrence (2006)
	WRD — overburden (four samples)	July 2002	<0.02	—	0.97 ± 0.17	—	Lawrence (2006)
Coronation Hill	Nearby adjacent areas	Mid 1980s	—	—	0.18 ± 0.28	—	DM (1988)
	Approximate background	—	—	—	0.062 ± 0.007	—	—
Koongarra	Ore stockpile (proposed)	1978 EIS est.	—	—	70–184	—	Noranda (1978)
	Waste rock stockpile (proposed)	—	—	—	9–26	—	—

(continued on next page)

Table 9 (continued)

Site	Source and conditions	Period of survey	Grade (%U ₃ O ₈)	Area (ha)	Exhalation (Bq/m ² /s)	Release (GBq/d)	References
Yeelirrie	Stockpiles (various) (proposed)	1978 EIS est.	0.44	417.8	~ 1.6	566	WMC (1978b)
	Stockpiles (post-mining) (proposed)	1978 EIS est.		417.8	~ 0.9	339	WMC (1978b)
	Waste rock (trial mine stockpile)	Nov. 1976		Small	0.0015	—	WMC (1979)
	Stockpiles (various) (proposed)	1979 EIS est.		~ 400	2.82	975	WMC (1979)
Olympic Dam	Ore stockpile (proposed)	1982 EIS est.	~ 0.08	—	—	8.6	Kinhill (1982)
Ben Lomond	Overburden (proposed)	1979 EIS est.	0.0008	—	—	0.7	Minatome (1979)
	Waste rock (proposed)		0.0033	13.6	—	3.6	
	Ore stockpile — mill (proposed)		—	—	—	1.2	
Ben Lomond	Waste rock (proposed)	1983 EIS est.	—	10	0.5	4.4	Minatome (1983)
	Low-grade ore (proposed)		—	5	4	17.2	
	Ore stockpile — mill (proposed)		—	1	10	8.6	

Table 10
Estimated or measured radon releases from uranium processing mills

Site	Current status	Date of survey/estimate	Release (GBq/d)	Capacity (t U ₃ O ₈ /year)	References
Ranger	Operating commercially	1974 and 1975 EIS estimates	44	3000	RUM (1974, 1975)
		1977 Ranger Inquiry estimate	20 → 148	3000	Fox et al. (1977)
		1989 and 1992 Research estimates	147	3000	Kvasnicka (1990, 1992)
		1993 Research estimates	150	3000	Akber et al. (1993)
Beverley	Operating commercially	1998 EIS estimate	~101	~1000	HR (1998)
Honeymoon	Commercial mill proposed	2000 EIS estimate	484	~1000	SCRA (2000)
Olympic Dam	Operating commercially	1982 EIS estimate	16.4 ^a	3000	Kinhill (1982)
		June 1992 → May 1993	57 ^b	1351 ^c	Davey (1994)
Yeelirrie pilot mill	Care and maintenance	1978 EIS estimate	0.19	~12	WMC (1978a)
Yeelirrie	Undeveloped	1978 EIS estimate	311	2500	WMC (1978b)
Koongarra	Undeveloped	1978 EIS estimate	46 ^a	1375	Noranda (1978)

^a Includes evaporation ponds.

^b Assuming all radon is released during grinding and leaching.

^c Approximate actual production during period of measurements.

Of interest at Olympic Dam is the effect of shrinkage cracks on radon exhalation, with a field study given by Storm et al. (1997) and Storm (1998). Based on this data, cracks can significantly increase the radon exhalation, and though the full extent awaits further field or laboratory studies, it could be as high as an order of magnitude. The proposed radon exhalation for rehabilitated tailings storage facilities at Olympic Dam, according to the 1982 EIS (Kinhill, 1982), was 1 Bq/m²/s. This compares to the regional background radon exhalation of about 0.025 Bq/m²/s (WMC, 1992). The 1997 Expansion EIS (Kinhill, 1997) discussed the need to reduce radon exhalation at the time of rehabilitation, however, no rate or quantitative objective was presented.

It can be seen in Tables 11 and 12 that both predicted and measured radon exhalation vary considerably. The direct comparison of much of this data is hampered by the different field measurement techniques and lack of full reporting (or measurement) of data relevant to quantifying radon behaviour (especially moisture content).

Another important issue to note is the change in radon exhalation at Nabarlek following rehabilitation. Prior to mining, radon exhalation was of the order of 4–44 Bq/m²/s (Table 6), whereas they presently average 1 Bq/m²/s following rehabilitation (Table 11). This is clearly the product of improved environmental planning and design at modern uranium mines. At Nabarlek, the high grade ore body outcropped at the surface but during mining the ore was buried in the bottom sections of the mined out pit and only contaminated soils and waste rock were emplaced in the upper sections of the pit, which was capped using waste rock and some soils (see Klessa, 2001). If there was no signature from the tailings (or other radium-containing materials), the radon exhalation should be within regional background. As such, the rehabilitated radon exhalation of 1 Bq/m²/s shows a signature from radium-bearing materials near the surface. This is most likely to be related to the waste rock and radium-rich evaporation pond sediments emplaced in the upper section of the pit. A recent issue identified at Nabarlek, however, is a small region (0.44 ha) showing a strong radiation exposure within a land unit known as 'Erosion Unit 7' (Bollhöffer et al., 2006; Hancock et al., 2006). This region shows a high radon exhalation of 6.5 Bq/m²/s and is thought to be due to erosion of a thinner soil cover in this area and exposure of the underlying contaminated soils scraped from the evaporation ponds during rehabilitation works, although the strong disequilibrium between ²³⁸U and ²²⁶Ra could suggest mill tailings. As noted by Bollhöffer et al. (2006), it is important to understand radon exhalation in terms of the radium activity as well as physical properties such as porosity, grain size and rock coverage.

There are continuing management issues at most tailings sites, e.g. Rum Jungle (Pidsley, 2002), Nabarlek (Bollhöffer et al., 2003; Iles, 2005), Mary Kathleen (Lottermoser et al., 2003), Radium Hill (McLeary, 2004a), Port Pirie (McLeary, 2004b) and Rockhole (Cochrane, 2000). There is nothing publicly available to ascertain the current status of neither the Moline tailings nor the Yeelirrie pilot mill tailings just north of Kalgoorlie. In order to improve the prospects for future tailings management, a more coherent picture and quantitative framework are clearly required based on well defined and reported field-measured data (and not merely assumed or asserted values, such as 'zero').

Table 11

Radon exhalation and releases from abandoned, operating, rehabilitated and proposed uranium tailings piles – Northern Territory and Queensland

Site	Source and conditions	Period of survey	Area (ha)	Exhalation (Bq/m ² /s)	Release (GBq/d)	References
Rum Jungle	Unrehabilitated tailings	1977–78 ^a	~ 35	2.1	64	Davy et al. (1978), Ritchie (1985)
	Proposed rehabilitation target	—	—	0.14	—	Allen and Verhoeven (1986)
Nabarlek	Unrehabilitated dry tailings (lab)	1980s	—	32.2	139	Kvasnicka (1986)
	Final in-pit tailings (calculated)	1988 and 1996	—	3.63/4.71	—	Storm and Patterson (1999)
	UNSCEAR (1993) advised data	—	5	2.1	9.1	UNSCEAR (1993)
	Predicted rehabilitated tailings	—	—	~ 10 ⁻²²	—	Storm and Patterson (1999)
	Rehabilitated tailings (actual)	Aug.–Sept. 1999	4	1.03 ± 0.80	3.6	Martin et al. (2002)
	Rehabilitated tailings (actual)	1999–2002	4	0.97	3.4	Bollhöffer et al. (2006)
Nabarlek	Radioactive anomalous area ('Erosion Unit 7') ^b	Oct. 2002	0.44	6.51 ± 6.83	2.5	Bollhöffer et al. (2006), Hancock et al. (2006)
Rockhole	Unrehabilitated tailings	June 25–27, 1982	~ 2	~ 6 (average) < 5 → 21.1	10.4	Bastias (1987)
Moline	Unrehabilitated tailings	June 19–23, 1982	~ 18	~ 2 (average) < 1 → 17.9	31	Bastias (1987)
Ranger	Unrehabilitated dry tailings (lab)	1980s	—	10.4	—	Kvasnicka (1986)
Koongarra	Proposed operational tailings	1978 EIS est.	—	—	260	Noranda (1978)
Ben Lomond	Proposed operational tailings	1979 EIS est. ^c	6.8	24.5	144.1	Minatome (1979)
		1983 EIS est.	24	0.3	6.2	Minatome (1983)

^a Based on unpublished data quoted in the references (no date given). Number of sampling points was 24 with an average ²²⁶Ra activity of 26.5 Bq/g.

^b The source of the radioactivity in 'Erosion Unit 7' is considered to be tailings and contaminated soils scraped from the former evaporation ponds (Bollhöffer et al., 2006, pp. 321–322).

^c Estimated ²²⁶Ra activity of 17.1 Bq/g.

4.6. Radon from radium-contaminated areas

The radon exhalation and releases from areas of radium contamination remain poorly quantified. In general, the main areas which have received significant radium due to uranium projects are downstream of Rum Jungle and water management areas at Nabarlek and Ranger. The Magela Land Application Area (MLAA) at Ranger, which receives mine site runoff waters from Retention Pond 2 (RP2) elevated in magnesium, sulfate, uranium and radium, has had approximately 8.6 GBq of radium applied over about 51 ha between 1985 and 2004 (land application presently continues) (compiled and estimated from ERA, 1984–2005). Early research into the soils of the MLAA suggests that the radium is adsorbed within the topmost 5–10 cm of soil (Akber and Harris, 1991; Willett et al., 1993). This suggests an approximate increase in soil radium activity of about 100–200 mBq/g (assuming 1.6 t/m³ for topsoil), a range consistent with soil monitoring of the MLAA (pp. 80–84, 2002 Edition, ERA, 1984–2005). A recent field study of the MLAA showed a radon exhalation of 0.112 Bq/m²/s (Akber et al., 2004), with further details in Lawrence (2006). Given the MLAA area of about 75 ha, this gives a radon release of up to 7.3 GBq/d.

4.7. Total project radon releases

The total radon releases released by uranium projects across Australia are generally poorly understood with respect to changes from pre-mining or baseline conditions and relative to production levels. This is also complicated by the fact that the largest producer of tailings, Olympic Dam, produces uranium as a co-product with copper, gold and silver.

The total radon release for the Olympic Dam project, based on computer modelling of measured radon decay products, has been estimated as 518 GBq/d by Crouch et al. (2005). This value is somewhat lower than those in previous tables, though it should also be noted therein that actual measurements are often different to predicted values (including both higher or lower values).

Table 12

Radon exhalation and releases from abandoned, operating, rehabilitated and proposed uranium tailings piles — South Australia and Western Australia

Site	Source and conditions	Period of survey	Area (ha)	Exhalation (Bq/m ² /s)	Release (GBq/d)	References
Port Pirie	Unrehabilitated tailings	Survey 1 year	17.1 ^a 4.5 ^b	1.9 1.5 → 5.6 (avg) → 7.4	27.8 19.2	AAEC (1980)
Port Pirie	Covered tailings	Survey 1 year	17.1	0.12	1.8	Crouch et al. (1988), Hill (1986), Spehr (1984)
Olympic Dam	Proposed tailings (operating)	1982 EIS est.	400	0.6	207	Kinhill (1982)
	Covered Tailings	1982 EIS est.	400	1	346	Kinhill (1982)
	Operating Tailings	Jun 1997–Mar. 1998	380	1.24 → 3.5 (avg) → 8.2	1150	Storm (1998)
	Trial Covered Tailings	Mar. 1998	—	0.88	—	Storm (1998)
Lake Way	Proposed tailings (post-mining)	1981 EIS est.	—	0.75	—	BLA (1981)
Yeelirrie	Proposed tailings (operating) ^c	1978 EIS est.	330.3	~2.0	586	WMC (1978b)
	Proposed tailings (post-mining)	1978 EIS est.	330	~11.4	3261	WMC (1978b)
	Proposed tailings (operating)	1979 EIS est.	330	38.5	10,980	WMC (1979)

^a Total area.

^b Cells 2 and 3 only (majority of tailings).

^c Includes radon sourced from pit dewatering operations (0.3 ha) pumped to the tailings dam for evaporation.

A realistic site for total release estimates is Ranger, since estimates for most components of radon releases are available. A preliminary compilation for total radon releases at Ranger is given in Table 13. It is noteworthy that the various estimates over time by different authors are quite variable, and perhaps even counter-intuitive to what could be expected. For example, a comparison of the pre-mine estimates with operational pit radon releases would

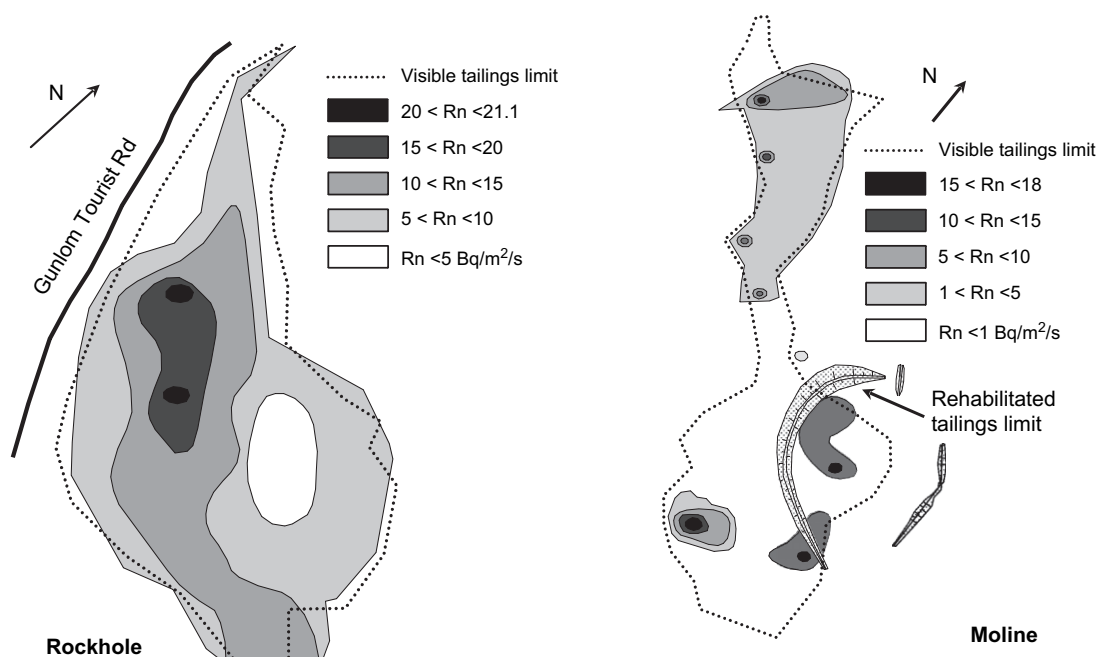


Fig. 7. Radon exhalation contours for uranium mill tailings before rehabilitation at Rockhole and Moline, June 1982 (no scale available) (redrawn from Bastias, 1987).

Table 13

Radon release estimates over time for the Ranger uranium project (GBq/d) (adapted from Mudd, 2002)

Year	Tailings management	Mill	Ore SP	WR	Pits	Tailings	Total	Reference
Pre-mine	—	0	0	0	372	5 ^a	377	Auty and du Preez (1994)
1975	>2 m water cover	44	19 ^b	—	32	<0.37	96	RUM (1975)
1977	—	20–148	~96 ^b	—	20–281	14–144	150–669	Fox et al. (1977)
1981	Bare tailings (<12% moisture)	—	—	—	—	4105	—	Haylen (1981)
1981	Covered tailings (1 m clay, 2 m soil)	—	—	—	—	48	—	Haylen (1981)
1980s	Sub-aqueous deposition	—	—	—	—	197	—	Davy (1983), Author ^c
1989	Sub-aqueous and aerial deposition	147	318	18	34	148	665	Kvasnicka (1990)
1992	—	147	318	8	44	96	613	Kvasnicka (1992)
1993	Sub-aerial deposition	150	325	15	26	94	610	Akber et al. (1993)
1990s	Sub-aqueous and aerial deposition	—	—	—	—	77	—	Davy (1983)
1990s	Sub-aqueous and aerial deposition	—	—	—	—	296	—	Author ^c
2000s	Sub-aqueous and aerial deposition	150 ^c	80 ^{c,1}	163 ^{c,2}	54 ^{c,3}	299 ^{c,4}	~750 ^{c,5}	Author ^c

^a Assuming a pre-mining exhalation of 0.05 Bq/m²/s.

^b Includes waste rock. WR, waste rock; SP, stockpiles.

^c Values calculated/adopted from previous tables, as well as including new data from Lawrence (2006); ¹5 Bq/m²/s over 18.5 ha; ²1 Bq/m²/s over 188.4 ha; ³1 Bq/m²/s over 62.0 ha; ⁴above ground dam and Pit #1; ⁵includes small allowance for land application areas (as noted in Section 4.6).

suggest that open cut mining is actually leading to a lower radon release whereas logic would expect an elevated release due to the significantly increased surface area open to the atmosphere. The estimate of 4105 GBq/d from tailings by Haylen (1981) is an extreme estimate in comparison to others in Table 13 but is kept for completeness.

A compilation of the total of radon sources and uranium production levels is given in Table 14, to allow an estimate of the radon release relative to uranium production (i.e. GBq/t U₃O₈). The estimates, after allowing for data gaps, show that the radon release per tonne of uranium is quite variable and commonly between 30 and 100 GBq/t U₃O₈. For comparison to earlier ISL data (54–143 GBq/t U₃O₈), the Beverley acid leach mine releases approximately 37 GBq/t U₃O₈. There is little apparent difference between ISL, open cut and underground mining for Australian-produced uranium.

The UNSCEAR analyses (and others critiquing them) have only assumed radon is released in the long-term from mill tailings. This fails to account for what is often the biggest source by mass and area — waste rock and low-grade ore, as well as other components which can sometimes provide significant radon releases, such as contaminated areas and abandoned mines. From an environmental and radiological perspective, it is the long-term success of rehabilitation and the cumulative changes from baseline which should be used as the basis for standards and assessing the local and global radiological consequences of uranium projects.

At current uranium projects, radon progeny is monitored in the surrounding environment, public radiological doses are estimated and provided these meet the relevant statutory requirements (i.e. <1 mSv/year), no further work has been considered necessary. This approach is inadequate, however, when setting rehabilitation standards and estimating long-term global doses as the releases are needed relative to the sources and operations at a specific uranium project. That is, we need to have a reliable estimate of the total radon released from the various source terms. Additionally,

Table 14

Predicted radon releases per unit Australian uranium production (GBq/t U₃O₈)

Project	Production (t U ₃ O ₈ /year)	Radon release (GBq/d)							Unit release (GBq/t U ₃ O ₈)	
		Mine	WR	Ore SP	Mill	Tailings	Minimum	Maximum	Minimum	Maximum
Beverley	1000				101		101	101	36.9	36.9
Ranger	5000	20–281	8–18	19–325	0–150	<0.4–299 ^a	47.4	1073	3.5	78.3
Olympic Dam	4500	120–700			16–57	207–1150	343	1907	27.8	154.7
Yeelirrie	2500	600–2500	340–1000		311	586–11,000	1837	14,811	268.2	2162.4
Koongarra	1375	23–57			46	260	329	363	87.3	96.4
Ben Lomond	500	10–38	1–17			6–144	17	199	12.4	145.3
Nabarlek	1360		95			5–139	100	234	26.8	62.8

^a Excluding the estimate by Haylen (1981) for bare, dry tailings as Ranger's tailings have never been operated in this manner.

it should be recognised that the changes in radon releases from uranium projects are cumulative across the industry (including reductions). Radiological monitoring and assessment should be also designed and undertaken in such a way as to include the ability to compare against pre-mining (or natural) conditions and how rehabilitation plans can be designed to achieve, at the very least, this level of performance. In this way, the cumulative changes across the industry can be argued to meet the ALARA principle and minimise doses.

5. Discussion

There are two major difficulties with estimating total radon releases from Australian uranium projects: (i) the lack of comprehensive data over time (including comprehensive pre-mining studies) and (ii) differing methods and focus giving rise to inconsistent measurements and reporting. Aspects of these problems include either no measured or reported radium activity, moisture content, density or porosity. It is noted by [Bollhöffer et al. \(2003\)](#), in discussing the different radon exhalation values at the rehabilitated Nabarlek site, that discrepancies in measurement techniques and sample locations can affect overall results. Further significant issues are the geology and mining conditions for each deposit and the fact that almost all studies lack consistency on measuring or reporting moisture data. Given the critical importance of moisture and climatic differences, this remains a vexed issue. It is likely that these factors could explain, at least partly, some of the data variability within the tables.

Overall, this makes the direct comparison and use of the data somewhat problematic. Therefore, the detailed data compiled within this paper should be taken as indicative only. It should be emphasized that an assessment or calculation of radon releases from proposed and operating uranium projects should include all source term components (e.g. mine, mill, waste rock, stockpiles, tailings and mine site water ponds). The use of accurate field-measured data should be given the highest priority for studies on operating sites. For proposed sites, advantage can be taken of pilot milling and metallurgical research on ores to establish tailings' parameters, exploration data from drill cores, and so on. The practice of simply assuming data and other properties (as appears to be commonly undertaken in Australia at least) should be discouraged. The UNSCEAR analyses ([UNSCEAR, 1993, 2000](#)) both used assumed or approximated data for Australia – despite the available data from Australia (ignoring the somewhat disperse and often obscure location of some of the radon data).

It can be noted in the tables that for some older sites, both rehabilitated and abandoned, there is evidence of ongoing erosion problems leading to locally elevated radon exhalation (e.g. Nabarlek). Although measurements may be taken at a point in time, it is important to continually monitor and re-assess the radon sources of all sites, especially where population is nearby (e.g. Port Pirie) or some form of further land use is expected (e.g. Nabarlek).

In comparison to the UNSCEAR data, it would appear that Australia's equivalent tailings data are similar in dry density at 1.6 t/m^3 and also area at 0.95 ha/GWe year . To produce the $250 \text{ t U}_3\text{O}_8$ for 1 GWe year requires about 212 kt of $0.146\% \text{ U}_3\text{O}_8$ ore, with radium 15.2 Bq/g and a tailings thickness of about 14 m. In addition, some 288 kt of waste rock and low-grade ore is produced at an approximate average of $0.02\% \text{ U}_3\text{O}_8$, with radium 2.1 Bq/g and covering about 0.55 ha to a height of about 26 m. The radon releases can be predicted for these wastes using an online version of the US Nuclear Regulatory Commission's 'RAECOM' radon model ([Rogers et al., 1984](#)), as implemented by [Diehl \(2006b\)](#). The results are shown in [Table 15](#). The UNSCEAR data, $3 \text{ Bq/m}^2/\text{s}$ from 1 ha of tailings only, give a radon release of 2.6 GBq/d – compared to a possible range of radon releases for Australian-produced uranium of 2.9 – 12.6 GBq/d (tailings plus waste rock and low-grade ore). The total radon release depends on the combination of moisture content and emanation coefficient adopted, however, in any case the radon releases from Australian-produced uranium are likely to be higher than that assumed by UNSCEAR data. Rehabilitation works could reduce the long-term radon release but the field evidence is not convincing (e.g. erosion problems at Nabarlek leading to locally higher radon exhalation).

Given the widely varying conditions and compiled data, however, a standardised rate per GWe year is clearly not realistic; instead, site-specific and comprehensive field studies should be used. As noted previously, however, the UNSCEAR-style approach above ignores the additional sources from uranium projects, such as waste rock and contaminated areas, which can also be significant sources as shown in [Tables 9 and 14](#). The long-term radon releases from waste rock and/or contaminated areas would clearly depend on the extent and effectiveness of rehabilitation works, with the sites for which actual post-rehabilitation radon exhalation data exist being restricted to Port Pirie and Nabarlek. In order to keep within the ALARA principle, it is therefore important to ensure that changes in radon releases are minimised – including waste rock, tailings and other potential sources.

Table 15

Predicted normalised radon exhalation and releases from Australian uranium for a standard reactor year (1 GWe year)

Waste	Moisture condition	Moisture (%dry weight)	Emanation coefficient	Exhalation (Bq/m ² /s)	Release (GBq/d)
Tailings	Dry	0.1	0.1	7.82	6.4
Tailings	Moist	8	0.25	14.17	11.6
Tailings	Saturated	31.3 ^a	0.3	2.80	2.3
Waste rock and low-grade ore	Dry	0.1	0.1	1.23	0.58
Waste rock and low-grade ore	Moist	8	0.25	1.94	0.2
Waste rock and low-grade ore	Saturated	26.8 ^b	0.3	0.15	0.07

^a Saturated moisture based on calculated porosity of 0.50, based on the estimated average tailings particle density ('specific gravity') of about 3.2 (see tailings in Section 3.2).

^b Saturated moisture based on calculated porosity of 0.43, based on the assumed waste rock particle density ('specific gravity') of about 2.8.

6. Conclusions

This paper has presented a detailed compilation and analysis of radon exhalation and releases from Australian uranium mining and milling projects. The primary purpose was to estimate normalised tailings and waste rock data and radon exhalation and release rates for a standard reactor year of uranium production to assess the efficacy of the UNSCEAR approach to long-term radon release from uranium mining (and consequent global population radiological doses). Overall, the UNSCEAR data for solid waste parameters are reasonable though it ignore potential major sources such as waste rock and low-grade ore. The extensive Australian data compiled for radon exhalation and releases for the various components of uranium mining and milling demonstrate wide variation and data quality, and show that waste rock and low-grade ores can be significant sources of radon. Importantly, the evidence on the effectiveness of rehabilitation works in reducing radon exhalation and releases is not convincing, especially when comparing cumulative changes from pre-mining conditions. Further work is required to ascertain whether this is due to design conflicts between revegetation or radon exhalation reduction requirements for engineered covers. When adopting more realistic data for tailings and waste rock, the UNSCEAR approach appears to underestimate the radon released from a standard reactor year of uranium production, though this needs to be moderated by the uncertain long-term effectiveness of engineered rehabilitation works. This paper has also shown that there is potential for uranium mining and milling to increase long-term radon releases into the adjacent environment relative to baseline or pre-mining conditions. In summary, these issues remain to be recognised in the broader debate about life-cycle analyses of uranium mining and nuclear power.

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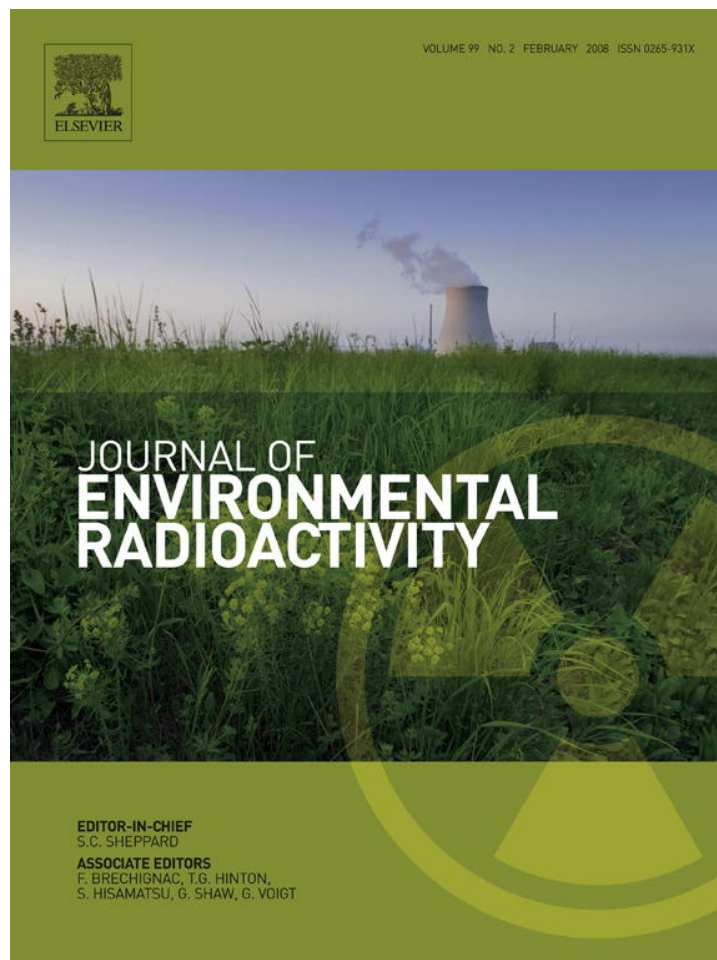
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Radon sources and impacts: a review of mining and non-mining issues

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Abstract Radon is a ubiquitous natural carcinogen derived from the three primordial radionuclides of the uranium series (^{238}U and ^{235}U) and thorium series (^{232}Th). In general, it is present at very low concentrations in the outdoor or indoor environment, but a number of scenarios can give rise to significant radiological exposures. Historically, these scenarios were not recognised, and took many centuries to understand the links between the complex behaviour of radon and progeny decay and health risks such as lung cancer. However, in concert with the rapid evolution in the related sciences of nuclear physics and radiological health in the first half of the twentieth century, a more comprehensive understanding of the links between radon, its progeny and health impacts such as lung cancer has evolved. It is clear from uranium miner studies that acute occupational exposures lead to significant increases in cancer risk, but chronic or sub-chronic exposures, such as indoor residential settings, while suggestive of health risks, still entails various uncertainties. At present, prominent groups such as the BEIR or UNSCEAR committees argue that the ‘linear no threshold’ (LNT) model is the most appropriate model for radiation exposure management, based on their detailed review and analysis of uranium miner, residential, cellular or

molecular studies. The LNT model implies that any additional or excess exposure to radon and progeny increases overall risks such as lung cancer. A variety of engineering approaches are available to address radon exposure problems. Where high radon scenarios are encountered, such as uranium mining, the most cost effective approach is well-engineered ventilation systems. For residential radon problems, various options can be assessed, including building design and passive or active ventilation systems. This paper presents a very broad but thorough review of radon sources, its behaviour (especially the importance of its radioactive decay progeny), common mining and non-mining scenarios which can give rise to significant radon and progeny exposures, followed by a review of associated health impacts, culminating in typical engineering approaches to reduce exposures and rehabilitate wastes.

Keywords Radon · Radon progeny · Uranium mining · Indoor radon · Health impacts · Radiation exposure

1 Introduction

Radon is a radioactive member of the noble gases, and is derived from the decay of primal uranium or thorium. As an element, radon was first confirmed and studied over the period 1898–1903, and involved

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many prominent scientists such as Ernest Rutherford, Frederick Soddy, Marie and Pierre Curie, Friedrich Ernst Dorn and others. Despite its somewhat late discovery by science compared to many other elements, radon and its associated health impacts have been felt for many centuries. For example, by the sixteenth century in the Erzgebirge ('Ore Mountains') of central Europe, miners often complained of 'lung-wasting disease' (i.e. lung cancer) but a cause remained elusive and mysterious. It was from mines in this region that uranium was first isolated in 1789 by Martin Klaproth, although health studies suggesting links between uranium, radon exposure and lung cancers would not emerge until a century later.

Throughout the twentieth century, extensive monitoring and research has allowed a more wide-ranging picture of radon and its radioactive decay progeny to emerge, as well as a more comprehensive understanding of potential health issues linked to radon exposure. Radon is considered to be responsible for about half of natural radiation exposure (e.g. UNSCEAR 2000), and therefore possibly a major contributor to background health impacts such as lung cancer (in the absence of other risk factors such as smoking) (e.g. Haque and Kirk 1992; NAS 1999a; Pearce and Boyle 2005; Rosario and Wichmann 2006). The link between sources, exposure and impacts is not always clear or decisive, but the modern approach adopts a generally cautious stance on radon exposures (commonly by minimisation). Since radon is derived directly from the decay of radium, its behaviour can often be governed as much by its parent radium as much by the primary source of uranium (or thorium).

This paper presents a broad review of radon issues. Firstly, it briefly reviews the history of radon, from initial discovery to the more complex understanding of its role in radiological exposures and health impacts. Secondly, it covers the principal physical, chemical and radiological properties of radon, as required for source, transport, exposure or remediation studies. A compilation of natural or background radon is then presented. Next the paper reviews common scenarios for radon sources and exposures, covering non-mining situations such as residential (indoor) radon, caves, and earthquakes, moving to various mining-related issues for uranium, mineral sands, phosphate, oil and gas, gold and coal mining and some miscellaneous problems. This leads to a

discussion of radon exposure and health issues, ending up in different remediation strategies commonly employed to address radon sources and minimise potential exposures. The paper is therefore intended to be a broad but thorough coverage of the principal radon sources, exposures and impacts.

2 Brief history

The effects of radon had been felt by Erzgebirge miners since at least the sixteenth century, as noted by Agricola in his seminal 1556 work *De Re Metallica* (Agricola 1556), though a cause–effect relationship for the sicknesses remained obscure. In 1789, German chemist Martin Klaproth first isolated uranium minerals from these mines (Schneeberg and Joachimsthal) (Habashi and Dufek 2001). Uranium was mined essentially for boutique purposes, such as dyes and ceramic glazes. The discovery and proof of the phenomenon of radioactivity from uranium was made by French physicist Henri Becquerel by mid-1896—starting a rapid revolution in the field soon to become known as nuclear physics (Gowing 1964; Weeks and Leicester 1968; Boorse et al. 1989). Soon afterwards, Becquerel's Polish assistant and research student, Marya Skłodowska (soon to become famous as Marie Curie), and her French husband Pierre Curie isolated the main sources of the radioactivity between 1898 and 1902 as the new elements polonium and radium; also demonstrating in 1898 that thorium was radioactive (Gowing 1964; Habashi 2001).

Around this period, between 1898 and 1903, many scientists were independently researching uranium, thorium and the new phenomenon of radioactivity. In 1899, Ernest Rutherford carefully demonstrated in his laboratory at McGill University in Canada that thorium (ie. ^{232}Th) led to an 'emanation' of radioactive particles—what we now know as 'thoron' or radon-220 (^{220}Rn) (Wilson 1983). Over 1900 to 1902, Friedrich Ernst Dorn in Germany, Frederick Soddy joining Rutherford in Canada and Marie Curie in France all noted that radium emanated radioactive 'particles'—what we now know as radon-222 gas (^{222}Rn). Although Dorn is often given credit as the first to identify the existence of radon, Marshall and Marshall (2003) recently revisited the historical papers and places involved and argued convincingly that due credit for radon's discovery should be given

to Rutherford (and possibly Soddy). The role of water in boosting emanation was also observed. The emanation from actinium ('actinon' or ^{219}Rn) was discovered independently in 1904 by Freidrich Fiesel and André Debierne. Rutherford and Soddy published their seminal papers in 1903 that radioactive decay led to new elements being formed, with the radioactive decay following a simple geometric law leading to a characteristic property or 'half-life' for each radioactive element (Cothorn and Smith 1987). The science of nuclear physics was thus born and continued to evolve rapidly over the coming decades (culminating in the atomic bomb in 1945).

The use of radium emanation as a potential health treatment was proposed soon after its discovery and the medical use of radium, and later radon, quickly accelerated as radium slowly became available (Caufield 1989; Mogren 2002). Throughout the 1910s radium-laced waters were sold as health tonics, radioactive spas were promoted for asserted health benefits and radium even began to be used in luminescent paints for clock dials and even soldiers in the trenches of World War I (Clark 1997). Demand for medical radium escalated dramatically, reaching an incredible price of \$100,000 *per gram* (Habashi and Dufek 2001). Sources of radium were considered extremely rare, and, following in great tradition, a uranium mining boom began to procure prized radium, though the global market was quickly controlled by a handful of mines or even individual countries such as the United States, Belgium and Canada (e.g. Landa 1993; Habashi 2001; Mogren 2002; Mudd 2005). The 1930s saw the emergence of research suggesting links between radon exposure and health impacts such as lung cancers, initially from studies of the Joachimsthal miners but also from strong evidence of health impacts among radium painters (Cothorn and Smith 1987; Jacobi 1993; Clark 1997). This also coincided with increasing understanding of the potential health impacts of excessive radium exposure. In 1934, the International Commission for Radiological Protection (ICRP), proposed the first standard for exposure to radiation—ushering in the era of health physics to minimise and manage impacts. The ICRP standards have evolved over the decades, generally always decreasing as further research comes to light on the relationship between radiation exposure and health impacts.

At the start of World War II, radioactive decay chains were well defined for uranium (^{238}U , ^{235}U) and thorium (^{232}Th), analytical testing was of increasing accuracy and able to detect very low activities, and potential scenarios for exposures and health impacts were beginning to be understood—though much remained to be studied for the latter area of radon.

3 Properties of radon

3.1 Basic properties

Radon is the heaviest member of the noble gas family and is colourless, odourless, relatively chemically inert, naturally radioactive, and has the highest melting point, boiling point, critical temperature and critical pressure of noble gases (Cothorn and Smith 1987). It is soluble in water, with solubility decreasing with increasing temperature, as shown in Fig. 1. An important property of radon is its higher solubility in organic solvents compared to water, a property used in various analytical or field techniques (e.g. Al-Azmi et al. 2004; Fan et al. 2007; Schubert et al. 2007). In general, radon behaves as an inert gas, though it can form clathrates and complex fluorides (no successful formation of oxides or other halides is known) (Cothorn and Smith 1987). The chemistry of

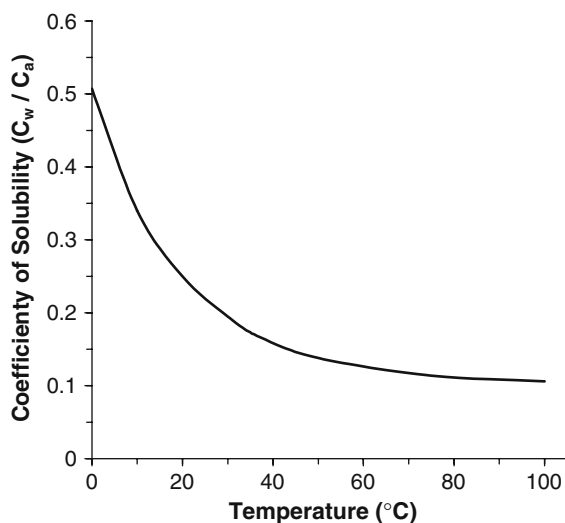


Fig. 1 Solubility of radon in water with respect to temperature (where C_w and C_a are concentrations in water and air, respectively) (adapted from Cothorn and Smith 1987)

Table 1 Uranium (^{238}U), Actinium (^{235}U) and Thorium (^{232}Th) decay chain and weighted average alpha energies (MeV) leading to ^{222}Rn , ^{220}Rn and ^{219}Rn ^a

	Half-life	α (MeV)		Half-life	α (MeV)		Half-Life	α (MeV)
^{238}U	4.51 Gy	4.16	^{235}U	710 Gy	4.20	^{232}Th	14.1 Gy	4.00
^{234}Th	24.1 d		^{231}Th	25.5 d		^{228}Ra	5.77 y	
^{234}Pa	70.2 s		^{231}Pa	32.5 ky	4.97	^{228}Ac	6.12 h	
^{234}U	247 ky	4.76	^{227}Ac	21.6 y		^{228}Th	1.91 y	5.40
^{230}Th	75 ky	4.67	^{227}Th	18.5 d	5.97	^{224}Ra	3.64 d	5.67
^{226}Ra	1600 y	4.77	^{223}Ra	11.4 d	5.83	^{220}Rn	54.9 s	6.29
^{222}Rn	3.82 d	5.49	^{219}Rn	4.01 s	6.76	^{216}Po	150 ms	6.78
^{218}Po	183 s	6.00	^{215}Po	1.8 ms	7.39	^{212}Pb	10.6 h	
^{214}Pb	0.467 h		^{211}Pb	0.602 h		$^{212}\text{Bi}^a$	1.01 h	6.05
^{214}Bi	0.328 h		^{211}Bi	129 s	6.57	^{208}Tl	186 s	
^{214}Po	164 μs	7.69	^{207}Tl	286 s		^{206}Pb	Stable	
^{210}Pb	22.3 y		^{207}Pb	Stable				
^{210}Bi	5.01 d					$^{212}\text{Bi}^a$	1.01 h	
^{210}Po	138 d	5.30				^{212}Po	0.304 μs	8.78
^{206}Pb	Stable					^{208}Pb	Stable	

^a Approximately 64% of ^{212}Bi decays by α and 36% by β ; other minor decays not included

y, years; d, days; h, hours; s, seconds; G, billion (10^9); k, thousand (10^3); m, thousandths (10^{-3}); μ , millionths (10^{-6})

References: UNSCEAR 1993; Titayeva 1994; IAEA 2003; Appleton 2005; BNL 2008

radon remains relatively understudied compared to other noble gases (Malli 2001).

The element radon has 86 protons and a variable number of neutrons in its atomic nucleus, due to the radon being derived from a different parent radionuclide decay series. The three primary sources for natural radon are the parent isotopes of the two uranium series (^{238}U and ^{235}U) and the thorium series (^{232}Th); with the decay chain sequences and alpha energies shown in Table 1. These chains give rise to the specific radon isotopes of ^{222}Rn , ^{219}Rn and ^{220}Rn , respectively, and their decay products, commonly referred to as ‘progeny’. Although there are 33 radon isotopes known with 110–142 neutrons (Ekström and Firestone 2008), only radon (^{222}Rn), actinon (^{219}Rn) and thoron (^{220}Rn) are relevant in natural or industrial contexts. The extent to which radon, actinon and thoron isotopes (used hereafter to refer to their respective radon isotopes) are present in a given situation will depend on whether the decay chain is in secular equilibrium (i.e. undisturbed) and the primary concentration of uranium and thorium.

All radon isotopes give rise to progeny of polonium, bismuth, tellurium (actinium series only) and

lead with half-lives ranging from fractions of a second (e.g. ^{214}Po) to several years (e.g. ^{210}Pb). These relatively short half-lives give rise to the progeny all having a very high specific radioactivity, and includes numerous alpha, beta and gamma decay steps (see later sections). The behaviour of radon and progeny is therefore critical to understand in order to predict radiation exposures.

The process of alpha decay leads to the recoil of both the alpha particle (which is a charged helium atom, $^4\text{He}^{2+}$) and the progeny. For radon and progeny, this can be crucial in understanding the release into the environment, and will depend on the location of the parent isotope, crystal pore structure and the presence of water, shown in Fig. 2. The recoil distance a radon (^{222}Rn) atom can travel in different media is about 20–70 nm for solids, 100 nm in water and 64,000 nm in air (Tanner 1980; Greeman and Rose 1995).

A final critical aspect of radon and progeny behaviour is the attached–unattached fraction issue. Radon is a relatively inert noble gas but its progeny are all considerably more chemically reactive, leading to an important property for progeny of attaching

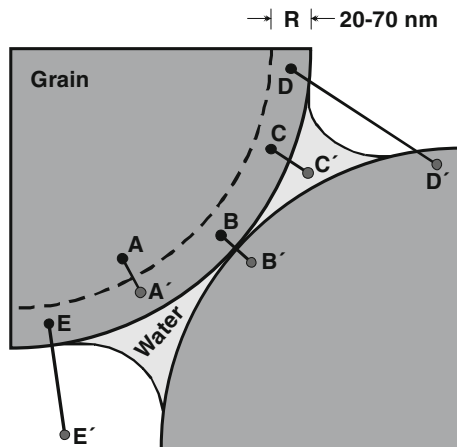


Fig. 2 Radon atom recoils: A–A' inside same mineral grain; B–B' from one mineral to adjacent mineral; C–C' from mineral to water; D–D' from mineral through air to adjacent mineral; E–E' from mineral to air (adapted from Cothorn and Smith 1987; Lawrence 2005)

to aerosols and particulates in the atmosphere. This leads to the fundamental distinction between attached and unattached progeny, a critical aspect in estimating the biological effects of progeny and radiation exposure in lungs (see later health section) (e.g. USDoE 1990; Lugg and Probert 1997; NAS 1999a).

3.2 Units

Due to the complex and rapid evolution in the scientific understanding of radon and progeny, a variety of units have historically been used, especially with respect to assessing exposures and health impacts. For completeness, these include (common symbol, units):

- *Curie*: specific activity of one gram of pure radium-226 (symbol *Ci*);
- *Becquerel*: one radioactive decay per unit time (symbol *Bq*, 1 *Ci* = 3.7×10^{10} *Bq*, 1 *pCi/l* = 37 *Bq/m³*);
- *Electron volt*: the product of the charge of an electron and one volt (symbol *eV*; 1 *eV* = 1.6×10^{-19} *J*);
- *Potential alpha energy*: the total alpha energy emitted by a radon atom as it undergoes complete decay, ideally giving a measure of the energy released if that atom decayed completely inside a lung (symbol *PEA* or ϵ_p , units *MeV per atom*). In practice, since the half-life of ^{210}Pb is 22.3 years

and it would most likely be excreted by lung fluids away from the lung before decay, only the alpha energies from ^{222}Rn to ^{214}Po are included;

- *Potential alpha energy concentration*: the cumulative or total potential alpha energy in a given air volume (symbol *PEAC* or c_p , units *MeV/l* or *J/m³*);
- *Equilibrium equivalent concentration*: due to disequilibrium between radon and progeny, this is defined as the activity of the parent radon gas in secular equilibrium which has the same potential alpha energy concentration as the non-equilibrium sample (symbol *EEC*, units *Bq/m³*). An equilibrium factor is also defined as the ratio of EEC to the equilibrium PAEC (symbol *F*);
- *Working level*: similar to PAEC, the working level was, initially, somewhat arbitrarily defined as 100 *pCi/l* (3.7 *Bq/l*) of air for each of the alpha decays from ^{222}Rn to ^{214}Po , chosen on the belief that this should minimise potential health impacts such as lung cancer (units *WL*, note 1 *WL* = 1.3×10^5 *MeV/l*). This was subsequently relaxed to any combination of radon and progeny which leads to 1.3×10^5 *MeV* of potential alpha energy, and it implicitly assumes a secular equilibrium situation (e.g. 1 *WL* = 3,700 *Bq/m³* ^{222}Rn ; similar activities can be calculated for ^{220}Rn and ^{219}Rn at equilibrium);
- *Potential alpha energy exposure*: given that working level is the activity concentration at a specific time, the cumulative *WL* over time gives a measure of the total alpha energy exposure (symbol *E*, units *WLM*). Commonly, for convenience in worker health studies, a monthly unit was adopted, giving exposure of 'working level months' (assuming say 2000 h of work per year or ~170 h per month);
- *Absorbed dose*: this is a measure of the radiation energy absorbed directly by cells (symbol *D*, units *J/kg*, named *Gray* or symbol *Gy*);
- *Dose equivalent*: in order to compare effects of radiation, an equivalent unit is required which allows for the differing biological effects and sensitivity of alpha, beta and gamma radiation (symbol *H*, units *Sievert* or *Sv*). The Sievert represents the effective biological impact from radiation exposure after taking into account weighting factors for organ sensitivity, radiation type and other factors.

3.3 Measurement

There are a variety of techniques available to measure radon and progeny. In essence, they can be grouped into three principal techniques: (i) grab sampling; (ii) continuous; (iii) integrative over time (Cothorn and Smith 1987; Harley 1992). Some equipment can be used under either group, such as scintillation cells or ionisation chambers being used for grab sampling or continuous monitoring. Ideally, measurement techniques should establish the respective activities of all radon and progeny isotopes, thereby facilitating the most accurate biological dose models for exposure assessments. However, in practice, the inherent complexity of radon and progeny behaviour means a compromise is required between practicality and theoretical considerations (Cothorn and Smith 1987). The common equipment used for each group includes (e.g. Cothorn and Smith 1987; Harley 1992; IAEA 1992b; Lawrence 2005):

- (i) *Grab sampling*: scintillation cells, ionisation chambers, two filter method;
- (ii) *Continuous*: scintillation cells, ionisation chambers, passive barrier with progeny collection on scintillator, two filter method;
- (iii) *Integrative*: passive barrier with progeny collection on thermoluminescent dosimeter (TLD), activated charcoal, solid state nuclear track detectors (also called “track etch” detectors).

It is important that techniques and programs for radon assessment include calibration and quality control, as there can be wide variability in radon measurements—even using standard equipment and techniques (Djeffal et al. 1992). The advent of powerful portable computing and electronics is making more complex field instrumentation practicable (e.g. Todd 1998; Martin et al. 2004; Lawrence 2005), such as linking continuous radon and thoron devices to weather stations. A radon–thoron emanometer for use in measuring field exhalation rates is shown in Fig. 3.

3.4 Exhalation behaviour

The release or escape of radon isotopes and their progeny from its parent radionuclide is a complex process and dependent on many factors. The process, when first recognised by Rutherford, the Curies and



Fig. 3 Radon and thoron emanometer and field laptop (Lawrence 2005)

others, was quickly termed ‘emanation’. For this paper, emanation is used to refer to release from a mineral particle into adjacent pore space, while ‘exhalation’ refers to release into the surface environment.

The location of the uranium or thorium and the respective radium isotopes are fundamental, such as the mineral and its crystal structure (e.g. surface coating of uraninite on a silica grain versus immobilised inside a monazite mineral). Other factors which can influence the exhalation rate include moisture content, barometric pressure, preferential pathways (e.g. cracks, fractures), temperature, particle size and morphology, radium distribution (e.g. diffuse or concentrated, especially with respect to mineral surfaces), seasonal and vegetation effects (e.g. Dyk and Tan 1978; Tanner 1980; Hart 1986; Kvasnicka 1986; IAEA 1992a, b; Schumann and Gundersen 1996; Storm 1998; Storm and Patterson 1999; Lawrence 2005; Schmidt and Regner 2005). It

is unfortunate that many studies on radon fluxes from uranium mining and milling projects, in Australia at least, have not measured or reported most of the above factors, making only cautious comparisons possible (Mudd 2008b).

The fraction of radon which is released relative to its total production is known as the emanation coefficient, and can range from 0 to 1 but is generally between 0.2 and 0.5 (Tanner 1980; Greeman and Rose 1995; Schumann and Gundersen 1996). The emanation coefficient, in turn, is very dependent on moisture content (e.g. Strong and Levins 1982), with an example shown in Fig. 4. In high-grade ores, radiation damage can also increase the emanation coefficient, though this effect is not always present (IAEA 1992b).

Overall, these factors can lead to significant variation in radon exhalation rates and radon and progeny activities in air, such as diurnal variation (e.g. Jackson et al. 1981; Robé et al. 1992; Seftelis et al. 2007), seasonal variation (e.g. Magalhães et al. 2002; Schmidt and Regner 2005; Ruano-Ravina et al. 2008; Zhuo et al. 2008) or possible storm variation due to barometric pressure and/or moisture effects (e.g. Lawrence 2005). It is therefore critically important to understand the principal factors contributing to radon and progeny behaviour at any given site, especially over various time scales.

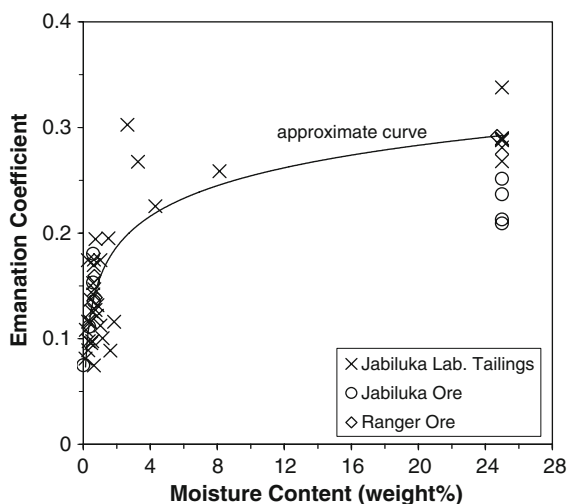


Fig. 4 Effect of moisture content on emanation coefficient for radon from Ranger ore and Jabiluka ore and laboratory tailings (adapted from Strong and Levins 1982; Hart 1986) (25% moisture assumed for saturated samples, based on estimated porosity and density data)

Table 2 Diffusion coefficients and diffusion lengths for radon in various media (Cothorn and Smith 1987)

Media	Diffusion coefficient (cm ² /s)	Diffusion length (m)
Air	10 ⁻²	2.4
Water	10 ⁻⁵	
Sand	3 × 10 ⁻²	1.5
Argillite	6 × 10 ⁻⁵	
Concrete	2 × 10 ⁻⁵	0.04–0.26
Mineral crystals	10 ⁻⁹ to 10 ⁻²⁰	

As a gas, radon is able to diffuse through different materials, with the diffusive flux proportional to the concentration gradient (Cothorn and Smith 1987). The diffusion coefficient (*D*) will vary according to the media, and can depend on the presence of water, crystal or mineral structure, temperature, radiation damage and particle size distribution (Cothorn and Smith 1987). Typical values for diffusion coefficients in various media are given in Table 2, showing that radon is only likely to migrate reasonable distances in air, water or more porous soils (since inside crystals the time taken for diffusion is longer than the half life).

The steady state exhalation of radon is commonly modelled using Fick's first law of diffusion in one dimension (e.g. Rogers and Nielson 1981; Rogers et al. 1984; Hart et al. 1986; IAEA 1992b; Ferry et al. 2001, 2002; Dinis and Fiúza 2008). The input data required commonly includes particle size distribution (e.g. sand–silt–clay fractions), dry density, radium activity, soil moisture retention characteristics, diffusion parameters, emanation coefficient (wet and dry) and soil thickness and porosity. The model can then be validated against measured field or laboratory data and used to design and predict the performance of engineered systems to minimise radon fluxes.

The techniques used to measure radon and progeny activity may be able to discern the unattached fraction, depending upon whether they are instantaneous or integrative.

4 Background radon

Uranium and thorium are widespread and exist in very low natural concentrations in soils and rocks,

typically about 3 mg/kg for uranium and 10 mg/kg for thorium (e.g. UNSCEAR 1993; Titayeva 1994; Langmuir 1997). This gives rise to a background radon exhalation from the earth's surface; some thoron exhalation studies are known but remain uncommon. Examples of some country studies include:

- *Australia*: seasonally-adjusted arithmetic mean radon and thoron exhalation from Australian soils is about 22 ± 5 and $1,700 \pm 400$ mBq/m²/s, respectively; the average ²²⁶Ra and ²²⁴Ra soil activities are 28 and 35 Bq/kg, respectively (Schery et al. 1989);
- *China*: area-weighted annual average radon exhalation from Chinese soils is about 30 ± 9.4 mBq/m²/s while ²²⁶Ra soil activity ranged from 17.5 to 115.5 Bq/kg (20 samples only) (Zhuo et al. 2008) (UNSCEAR give Chinese average ²²⁶Ra soil activity as 37 ± 22 Bq/kg; UNSCEAR 1993);
- *France*: radon exhalation from French soils ranges from 10 to 50 mBq/m²/s; the average ²²⁶Ra soil activity is 28 Bq/kg (Robé et al. 1992).

The global radon exhalation from soils ranges from 15 to 23 mBq/m²/s (UNSCEAR 1982). Although there are limited studies on background thoron fluxes, values of approximately 1–2 Bq/m²/s appear typical (UNSCEAR 1993). The radon exhalation gives rise to a general atmospheric background radon which can vary significantly—regionally, seasonally or even diurnally. As such, it is critical to note the weather conditions, time and season of measurements. Some examples of background radon activities in ambient outdoor air are shown in Table 3. In general, an outdoor radon activity of the order of 5–10 Bq/m³ is typical, with an equilibrium factor for progeny ranging from about 0.5 to 0.7 and probably averaging about 0.6 (though more extreme values between 0.2 and 1.0 are known) (UNSCEAR 2000). Background radon activities have also been used to model global atmospheric transport processes, in order to provide a validation of such models independent of climatic inputs (e.g. Zaborowski and Whittlestone 1996).

Radon activities in surface waters and groundwaters are, as one could expect, extremely variable. In general, groundwaters are higher in radon than surface waters (e.g. UNSCEAR 1993; NAS 1999b; Santos et al. 2008), primarily due to slower

Table 3 Ambient outdoor radon activities in air in the United States and Mexico (Bq/m³) (adapted from Gesell 1983; Segovia et al. 2007)

Region	Period of measurement	Average radon
Grants Mineral Belt, New Mexico	November	22.8
Grand Junction, Colorado	Annual	27.8
Laguna, New Mexico	June	18.5
Cincinnati, Ohio	Annual (morning)	16.3
Cincinnati, Ohio	Annual (afternoon)	5.1
Argonne, Illinois	Late spring/summer	11.1
Socorro, New Mexico	Annual	8.9
Chester, New Jersey	Annual	8.1
Lloyd, New York	Summer only	7.4
Lloyd, New York	March/April	3.0
Washington, DC	Annual (afternoon)	4.5
Hawaii	May/June	1.0
Wales, Arkansas	Annual (afternoon)	0.7
Kodiak, Arkansas	Annual (afternoon)	0.4
Mexico	Variable	13–23

movement rates and slightly higher radium from dissolved radium plus radon emanation from aquifer sediments. A major study of 100 public water supply systems in China showed a range from 0.04 to 100 Bq/l radon (Ren et al. 1996). In Mexico groundwaters and water supplies are commonly around 2.2 Bq/l radon but can reach a maximum of 34.2 Bq/l radon (Segovia et al. 2007).

The presence of locally elevated radon activities in surface waters can be used to detect possible groundwater discharge and thereby groundwater–surface water interaction (e.g. Cook et al. 2003; Schubert et al. 2006; Mullinger et al. 2007) or submarine groundwater discharge (e.g. Dulaiova et al. 2007; Lamontagne et al. 2007; Santos et al. 2008).

There is some limited information on the ambient or background radon activities in marine waters. Dissolved radon, commonly around 1 mBq/l, is in deficit relative to radium near the surface (~50 m) due to diffusive losses to the atmosphere, while radon is commonly in excess at the sediment–water interface (Cochran 1992). These results, the radon deficit or excess, can be used to assess oceanic mixing processes. A recent study of the Mediterranean Sea

showed generally low radon activities in its marine waters, ranging between 1 and 20 mBq/l from top to bottom, with wide variation over depths to 3 km (Vaupotič et al. 2008).

As noted for radon exhalation, issues such as seasonal and/or diurnal variation, sampling techniques, and so on need to be carefully considered in understanding background radon processes and activities for any given location or segment of the environment.

5 Non-mining radon issues

In its simplest context, it is possible for radon to accumulate to significant levels wherever there is a radium source and diffusion and transport processes are similar to the half-life for radon. It is principally the longest-lived radon (^{222}Rn) isotope involved, and not actinon or thoron, since these latter isotopes have very short half-lives and it is uncommon for situations to develop that allow build-up and exposure (e.g. UNSCEAR 1993).

To date, significant activities of radon have been observed in cave systems and residential dwellings. Due to their prime significance in population doses, only residential dwellings is reviewed in detail herein. A brief summary of other non-mining radon issues is subsequently presented.

5.1 Indoor radon

Although the potential for biologically significant radon exposures due to mining had been recognised since the early twentieth century, it was not until the 1970s that clear evidence came to light that it was possible for naturally-derived radon (i.e. excluding contaminated sites) to accumulate in residential dwellings, non-uranium mines or other situations to radiologically significant levels (Lugg and Probert 1997; IAEA 2003). Since this time there have been a large number of studies around the world investigating ambient radon activities in residential dwellings, including epidemiological studies for possible related health impacts.

According to an extensive compilation presented by UNSCEAR (2000), given in Table 4, an indoor radon activity of between 30 and 40 Bq/m³ is typical, with an apparent relationship to latitude shown in

Fig. 5. The data suggests that closer to the equator has lower ambient indoor radon, most likely due to greater ventilation associated with higher temperatures, although the scatter at higher latitudes suggests that other factors can also be important.

The accumulation of radon inside residential dwellings is a complex combination of factors and processes, sometimes competing against each other. The major factors involved in determining radon and progeny activities inside a residential dwelling include geology, climate, building materials, design and construction (especially single or multi-storey), building age, barometric pressure effects, and finally lifestyle (e.g. UNSCEAR 1993, 2000; Lugg and Probert 1997; Rosario and Wichmann 2006; Barros-Dios et al. 2007; Denman et al. 2007). Air pressure differences can suck radon into a dwelling or suppress it from entering (e.g. UNSCEAR 2000). Lifestyle aspects often relate to how a dwelling is utilised and can exacerbate or minimise radon issues. For example, Australia is generally considered to have generally low indoor radon due to an open and outdoor lifestyle, compared to cold climate countries where residences are often enclosed for most of the year. Some of these factors are related, such as climate, building design and lifestyle, however, they are not always related in the same manner in different parts of the world.

Some regions naturally contain elevated uranium and/or thorium in soils and rocks, such as granites up to 40 mg/kg uranium, and this can lead to significant radon emanating into and accumulating in dwellings. Based on studies in the UK, Czech Republic, Germany and elsewhere, the most common geological situations giving rise to elevated U/Th are related to granites (Appleton 2007). Alternately, the earthen materials used in construction may contain elevated U/Th, leading to elevated radon (e.g. alum shale in Sweden).

In a handful of mining towns dwellings were sometimes built over uranium mill tailings (or other tailings with elevated radium), or even using tailings in building materials, leading to major radon and progeny exposures for residents in any case (in some cases higher than underground uranium miners). Examples include:

- *Grand Junction and Mesa Counties, Colorado, USA*: approximately 312,000 tonnes of uranium

Table 4 Global summary of indoor radon (^{222}Rn) activities (Bq/m^3) (UNSCEAR 2000), including additional Mexican data from Segovia et al. (2007) and Paarl, South African data from Lindsay et al. (2008)

Region	Country	Arithmetic mean	Geometric mean	Maximum	Geometric SD
Africa	Algeria	30		140	
	Egypt	9		24	
	Ghana			340	
	South Africa (Paarl)	~37–132		465	
North America	Canada	34	14	1720	3.6
	United States	46	25		3.1
Central America	Mexico	83	75	448	
South America	Argentina	37	26	211	2.2
	Chile	25		86	
	Paraguay	28		51	
East Asia	China	24	20	380	2.2
	Hong Kong	41		140	
	India	57	42	210	2.2
	Indonesia	12		120	
	Japan	16	13	310	1.8
	Kazakstan	10		6000	
	Malaysia	14		20	
	Pakistan	30		83	
	Thailand	23	16	480	1.2
	Armenia	104		216	1.3
West Asia	Iran	82		3070	
	Kuwait	14	6	120	
	Syria	44		520	
	Denmark	53	29	600	2.2
North Europe	Estonia	120	92	1390	
	Finland	120	84	20000	2.1
	Lithuania	55	22	1860	
	Norway	73	40	50000	
	Sweden	108	56	85000	
	Austria		15	190	
West Europe	Belgium	48	38	12000	2.0
	France	62	41	4690	2.7
	Germany	50	40	>10000	1.9
	Ireland		37	1700	
	Luxembourg	110	70	2500	2.0
	Netherlands	23	18	380	1.6
	Switzerland	70	50	10000	
	United Kingdom	20		10000	
	Bulgaria		22	250	
	Czech Republic	140		20000	
Eastern Europe	Hungary	107	82	1990	2.7
	Poland	41	32	432	2.0
	Romania	45		1025	
	Slovakia	87		3750	

Table 4 continued

Region	Country	Arithmetic mean	Geometric mean	Maximum	Geometric SD
South Europe	Albania	120	105	270	2.0
	Croatia	35	32	92	
	Cyprus	7	7	78	2.6
	Greece	73	52	490	
	Italy	75	57	1040	2.0
	Portugal	62	45	2700	2.2
	Slovenia	87	60	1330	2.2
	Spain	86	42	15400	3.7
	Australia	11	8	420	2.1
Oceania	New Zealand	20	18	90	
Median		46	37	480	
Population-weighted average		39	30	1200	

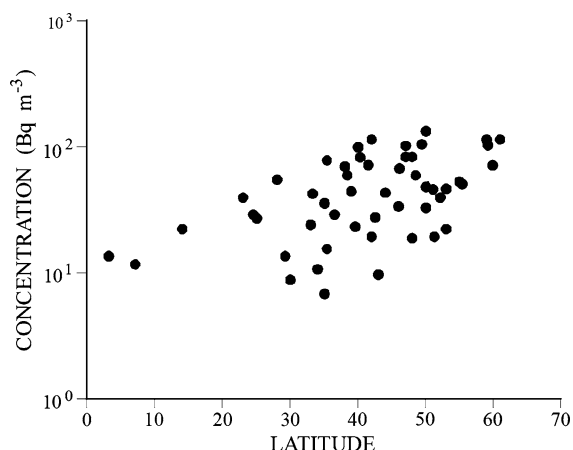


Fig. 5 Average indoor radon activity versus latitude (UNSCEAR 2000)

mill tailings were used in construction materials throughout the vicinity for more than 4,000 houses, schools, churches, public and commercial buildings (the material was provided freely by the mill) (Hazle et al. 1982; Rael 1999);

- *Canonsburg, Pennsylvania, USA*: the site of both a radium refinery in the early twentieth century and later a uranium mill from 1942 to 1957, Canonsburg also saw some mill tailings taken from the site for construction purposes (USDoe 2001);
- *Eastern Germany (former GDR)*: waste rock from the former Crossen uranium mine was used in buildings in eastern Germany (Küppers and

Schmidt 1994); another survey of 1700 homes in eastern Germany returned activities up to 15,000 Bq/m³ with one extreme value of 115,000 Bq/m³, with more than 50% of homes at Schneeberg greater than the local action limit of 250 Bq/m³; (Vandenhove et al. 2006);

- *South Africa*: similarly to the USA and Germany, local communities in the south-west Karoo Province used stockpiled uranium ore for road construction or farmhouse foundations, leading to indoor radon activities of 351–835 Bq/m³ and exposures of 6.0–14.2 mSv/year (Scholtz et al. 2005);
- *Hunters Hill, inner suburban Sydney, Australia*: the site of an old radium extraction refinery (1911–1915) and adjacent tin smelter (1895–1964) was redeveloped for residential housing though the site has yet to be satisfactorily remediated (Mudd 2005).

In areas around the world known for elevated indoor radon, there are often specific building codes, regulations or guidelines to ensure that building designs and construction minimise radon build-up and associated radiation exposures (e.g. USEPA’s “Citizen’s Guide”, which suggests an action level of 148 Bq/m³ for indoor radon; USEPA 2007).

Indoor radon and progeny activities are rarely in equilibrium, typically showing an equilibrium factor of about 40% (NAS 1999a). The differences between the use of short- and long-term detectors for the study of indoor radon been shown to be minimal, though it

is considered more thorough to use long-term detectors (Ruano-Ravina et al. 2008).

A case study of indoor equilibrium-equivalent radon and thoron progeny activities in 10 rammed earth and 10 conventional dwellings at Margaret River, just south of Perth in Western Australia, was given by Walsh and Jennings (2002). The study showed that mean indoor radon and progeny activities were 24 and 9.3 Bq/m³ EEC, respectively, while thoron and progeny activities were 3.9 and 0.8 Bq/m³ EEC, respectively, leading to a combined radiation dose for each dwelling type of 4.1 and 2.2 mSv/year, respectively—significantly above average Australian background radon exposures of about 0.7 mSv/year (see Webb et al. 1999). A more extreme example from Slovenia involved a dwelling with a radon exposure of 9–35 mSv/year (the cause of the high radon is not stated) (Zmazek and Vaupotič 2007).

An online radon and progeny dose calculator has been implemented by Diehl (2008a), based on unit conversion factors (e.g. alpha energy to WL to WLM). Based on a typical indoor radon activity of 40 Bq/m³, equilibrium factor of 0.4 and 60% occupancy over 70 years, it is possible to estimate a natural indoor radon exposure of about 0.53 mSv/year or 0.13 WLM/year.

5.2 Miscellaneous radon issues

A number of other scenarios are known to involve elevated radon activities, radiation exposures or uses. These include:

- *Mineral spas and thermal waters*: such waters often contain appreciable radon (by choice or otherwise). Examples include the Polichnitos hot springs in Greece with radon commonly between 110 and 220 Bq/l (Vogiannis et al. 2004), thermal springs of northern Venezuela with 1–578 Bq/l radon (Horváth et al. 2000), and Paralana spring in South Australia at 1,800–5,800 Bq/l radon (ambient air is highly variable but can range from 1,705 to 10,952 Bq/m³ directly over the two spring pools; radium is 14–17 Bq/l in pool waters) (HR 1998; Brugger et al. 2005).
- *Earthquakes or other land movements*: it has long been recognised that elevated radon activities in soils occur just before an earthquake (e.g. Singh

et al. 1999; Yang et al. 2005; Amponsah et al. 2008), generally considered to be related to the rapid migration of gases just before the earthquake. In some cases, however, seismic activity appears to give rise to reduced radon activities in soils, as measured in Japan (Yasuoka et al. 2005) and Taiwan (Kuo et al. 2006). Recently, radon in soil and/or groundwater has been shown to be linked to tectonic controls adjacent to landslide activity in the Himalayas of northern India (Ramola et al. 2007), as well as subsidence due to former underground iron ore mining in the Luxembourg Basin in Europe (Kies et al. 2006). Field studies at Yerevan, Armenia, have shown that the increases in radon activities due to seismic activity are important in public radiation exposure doses (Saghatelian et al. 2005).

- *Caves*: underground caves, most commonly in limestone, can also give rise to scenarios of elevated radon and progeny (e.g. (Szerbin 1996; Madden 1997). For example, numerous caves were assessed for radon activities in the mid-1990s in Australia by Solomon et al. (1996). Monitoring data in winter and spring showed radon activities of 500 ± 40 and 795 ± 50 Bq/m³, respectively, with a maximum of 6,330 Bq/m³. In winter 19% of measurements were above the action level of 1000 Bq/m³, rising to 29% for spring. Seasonal variability in radon activities does appear to be common in caves, including equilibrium factors.

Further papers on non-mining radon issues can be found in several recent conferences on naturally occurring radioactive materials ('NORM') held by the International Atomic Energy Agency in Vienna in September 1999 (IAEA 2002), May 2004 (IAEA 2005) and December 2004 (IAEA 2006), amongst many other conferences and considerable literature.

6 Mining radon issues

Mining is the most common industry where the potential for acute exposures can occur, principally in uranium mining but also for other commodities. This section will review several sectors of the mining industry which are known to encounter significant radon and related exposure issues.

6.1 Uranium mining and milling

The mining and milling of uranium ore can lead to high radon exposures, sometimes extremely high if minimal or no precautions are adopted. A typical uranium mine is similar to any other mine, consisting of an underground or open cut mine, economic ore, low grade ore and/or waste rock stockpiles, a processing mill (such as grinding, leaching, solvent extraction, chemical precipitation and calcining) and finally a tailings dam. Other infrastructure may also be required, such as water management facilities, a power station, dedicated road and air transport infrastructure and possibly a nearby town to accommodate workers and their families. A minor amount of uranium is also produced by non-conventional in-situ leach or ‘solution mining’, whereby acidic or alkaline chemical solutions are injected into and extracted from the porous uranium orebody using boreholes (i.e. essentially a chemical solution mining exercise in groundwater; see Mudd 2001a, b).

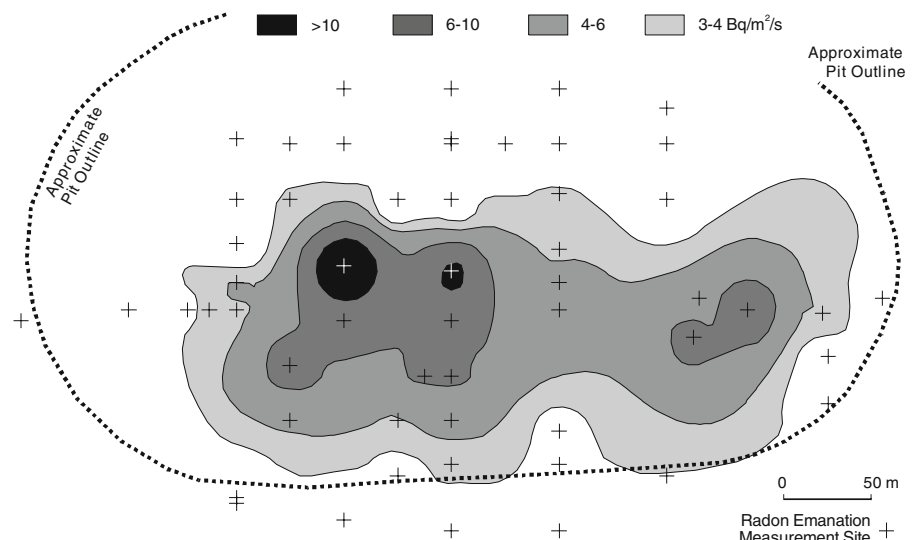
It is possible to seek elevated radon fluxes during uranium exploration programs (e.g. Stewart 1968; Smith et al. 1976; Severne 1978), however, whether there is a notable expression of radon from a uranium deposit at the surface is highly variable. A recent analysis of Australia showed that uranium ore-related radon fluxes were detectable for the Ranger, Yeelirrie, Nabarlek, Koongarra and Lake Way uranium deposits as these all outcrop or subcrop (sometimes over large areas as at Ranger and Yeelirrie), while

other buried uranium deposits such as Olympic Dam, Beverley, Honeymoon and Jabiluka showed no clear deposit-related radon signature (Mudd 2008b). The baseline radon flux contours for the Koongarra 1 uranium deposit are shown in Fig. 6 (Koongarra remains undeveloped).

Mining is commonly the occupation with the highest potential for radon exposures, principally in underground mining, while mill workers and other roles encounter lower radon activities and exposures. The nature of the exposure is complex, and depends on uranium ore grade, deposit mineralogy and geochemistry, ventilation regime (especially open cut versus underground), temperature and the extent and nature of particulates in the mine atmosphere (since this critically affects the attached–unattached progeny fraction and lung dosimetry; Wasiolek and James 2000). Additionally, the older the mine the higher its radon emissions are likely to be (Jackson et al. 1981). Blasting appears to temporarily increase radon and progeny activities as well as alter the equilibrium factor (Ertle et al. 1981; Warneke and Sontner 1989).

The uranium mines of the first half of the twentieth century, operated primarily for radium extraction, included a mix of open cut and underground mining, though it appears that underground mining was more dominant (similar to gold and other mining of the day). Efforts to minimise radiation exposures due to radon and progeny were minimal, as there was still only limited understanding of the links between high

Fig. 6 Pre-mining radon exhalation measured at the Koongarra 1 uranium deposit, 1978 (redrawn from Davy et al. 1978)



exposures and health impacts such as lung cancers (Jacobi 1993).

Following the advent of the Cold War nuclear weapons race from 1945, uranium mining became a major boom industry across the world, especially in the United States, Canada, South Africa, eastern Germany (the former German Democratic Republic, GDR) and many states of the Soviet Union. The first two decades were primarily concerned about urgent uranium production for the nuclear weapons programs of the day—although there was arguably limited but evolving understanding of radiological exposure issues, this came second to uranium production (Jacobi 1993). By the late 1960s, however, radiological exposure issues and standards had become more critical, with increasing attention being given to ventilation to address dust, radon and progeny issues with respect to health impacts (see next section).

There is an extensive range of technical reports, journal papers and conference proceedings with data on radon and progeny activities for uranium mining since the late 1940s. The major groups include the International Atomic Energy Agency (IAEA), International Commission on Radiological Protection (ICRP), Biological Effects of Ionising Radiation (BEIR), United Nations Scientific Committee on the Effects of Atomic Radiation (UNSCEAR), numerous government research agencies and regulators as well as technical societies, amongst others.

To illustrate the changes in radon exposures over time, a case study for Australian uranium mining and milling was developed and is shown in Table 5. This case study could be expected to be similar or representative of many other uranium mining countries.

Another major source of radon emissions or exposures from a uranium mining-milling project is tailings, the finely ground rock remaining after chemical processing and uranium extraction. Uranium mill tailings typically retain most of the radium from the original ore (and other radionuclides, including residual uranium), and thus constitute a major radon source term (Clements et al. 1978; IAEA 1992a). Tailings are invariably pumped as a slurry to an engineered storage dam, with the facility decommissioned and rehabilitated upon mine closure. The management of tailings is therefore critical with respect to radon (IAEA 1992b). A brief compilation

of radon fluxes from uranium mill tailings is shown in Table 6.

One approach which has been favoured by some in the uranium industry is the use of water covers, though their effectiveness remains the subject of some conjecture (Mudd 2008b). In Australia water covers have been viewed favourably, especially in the tropics for the operating Ranger and closed Nabarlek uranium projects, however, field evidence of the effectiveness of water covers in reducing radon fluxes and loads is lacking, with theoretical estimates varying widely (Mudd 2008b). Field studies in Brazil have shown that approximately one third of the radon in mine water retention ponds is released to the atmosphere (Paschoa and Nóbrega 1981). Based on laboratory column studies, Rogers and Nielson (1981) argued that the water covers on mill tailings facilities were a major radon source, and presented a model to estimate such releases (implemented online by Diehl 2008c).

The rehabilitation of uranium mill tailings generally involves dewatering (to the best extent practicable) followed by construction of an engineered soil cover over the tailings. The soil cover is designed with multiple layers to ensure some moisture retention and thereby retard radon diffusion, leading to lower radon exhalation at the surface (e.g. Rogers et al. 1984; IAEA 1992b). In Australia, the major public inquiry into the Ranger uranium project (1975–1977) (Fox et al. 1977) recommended final in-pit tailings disposal and management, primarily due to concerns over long-term radon exhalation close to indigenous communities after rehabilitation (Haylen 1981). If the tailings were deep in the former pit and below the water table, then the radon exhalation at the surface would be minimised after final rehabilitation.

A comprehensive review of radon exhalation and loads from uranium mill tailings, economic ore, low-grade ore and waste rock stockpiles and processing mills for numerous Australian uranium projects is presented by Mudd (2008b). The normalised radon released per tonne of uranium oxide production (ie. GBq/t U_3O_8) is variable, with estimates for Australia commonly ranging from 37 to 155 GBq/t U_3O_8 , with one extreme estimate at 2,162 GBq/t U_3O_8 (Mudd 2008b). The US Nuclear Regulatory Commission estimated the normalised radon release for a ‘generic’ uranium mill of about 318 GBq/t U_3O_8 (USNRC 1980). Based on written advice from various uranium

Table 5 Summary of radon activities over time for some Australian uranium mines¹ (compiled from Stewart 1963^a, Rosen 1987; Sonter 1987; Sonter and Hondros 1988; Warneke and Sonter 1989; Woodward et al. 1991; Kinhill 1997; HR 2003–2007; Mudd 2008a)

1950–1960s (general)	Typical ore Grade (%U ₃ O ₈)	Radon Average	(Bq/m ³) Maximum
Open cut mines	0.17–0.35%	40	190
Underground mines—low grade	~0.1%	3,000	9,300
Underground mines—high grade ^b	0.3–2.5%	‘0’–85,000	1,110,000 ^c
Chemical treatment plants	~0.7%	–	<190
Code of Practice (1955)			3,700
	Grade (%U ₃ O ₈)	Radon (Bq/m ³)	Progeny (m WL)
<i>Radium Hill</i> (underground mine)			
Before March 1955	~0.1%	2,100–18,000	600–1,800
March 1955–1961	~0.1%	100–7,900	100–550
<i>Ranger</i> (Nov. 1984–Oct. 1985) (open cut mine)			
Controlled and supervised areas	~0.3%	3,223 samples <3,100	Maximum 86; 1,020 samples <17
Non-controlled and non-supervised areas	~0.3%	1,709 samples <3,700; maximum 1,295; 1,230 samples <185	Maximum 17.6; mean mine office 4.9; other areas 1.4–4.5
Limits—Controlled / supervised areas	–	100,000	330
Limits—Non-controlled/Non-supervised areas	–	–	10
<i>Olympic Dam</i> (mid-1980s to late 1990s) ^d			
Underground ore stockpiles (in drive)	~0.1%	~200	~110–160
Underground drives and tunnels	~0.1%	–	~50–140
Underground open stopes (mid-1980s)	~0.1%	~5,200	~10,000–20,000
Underground stopes (late 1980s)	~0.1%	~2,000 to ~5,000	~100–300
Underground stopes—after blasting (late 1980s)	~0.1%	~2,000 to ~18,000	~200–1,200
Underground radon progeny exposures (1983–86)	Range: 0.2–1.0 WLM; Average ~0.35 WLM		
Underground mine (‘Purple Stope’, mid-1990s)	~0.08%	–	5.3–73
<i>Beverley</i> (2000s)			
‘Environmental’ radon ^e	–	~20 to ~195	~1 to ~7

¹ Uranium projects of the 1950–1960s included Rum Jungle (open cut), Upper South Alligator Valley (open cut, underground), Mary Kathleen (open cut), Radium Hill (underground), Port Pirie (metallurgical plant)

^a Stewart (1963) used units for radon of both ‘μC/l’ and pCi/l. Based on the paper and its timing, it appears that units of pCi/l are intended and have been assumed above

^b A small stope in a high grade underground mine (unnamed) averaged 5,550 Bq/m³ (ranging up to 22,200 Bq/m³), and progeny concentrations were 0.43 × 10⁵ MeV/l under natural ventilation and 0.125 × 10⁵ MeV/l under forced ventilation of 28.3 m³/min (Stewart 1963)

^c Described as an ‘exceptional value’ by Stewart (1963)

^d These values are estimates and not actual monitoring data (see Sonter 1987)

^e Location of monitoring point in relation to the in-situ leach processing plant unstated, though for some years it is stated as the nearby accommodation camp (see HR 2003–2007)

Note: No data on ambient radon activities appears to be published within active mining areas for the Nabarlek and Mary Kathleen uranium projects

Table 6 Brief survey^a of radon fluxes from uranium mill tailings around the world (IAEA 1992b)

Mine	Description	Ore grade (%U ₃ O ₈)	Radium (Bq/kg)	Radon Flux (Bq/m ² /s)
Olympic Dam, Australia	Moisture ~ 19%	0.08%	8,000	1.3
Lacnor, Elliot Lake, Canada (closed)	Frozen surface	~ 0.12%	600 to 12,700	0.03–1.52
	Vegetated surface			0.31–4.96
	Dry surface			0.20–0.84
	Damp surface, no vegetation			0.18–6.67
	Saturated surface			0.01–0.11
Key Lake, Canada	Very wet	~ 2.95%	100,000 to 315,000	0.78, 0.88
	Dry			1.11–10.2
	Damp		Average to 300,000	2.4, 8.6
	Frozen			0.14
Hamr, Czech Rep.	Very wet	0.12%	–	0.22
Rössing, Namibia	<1% moisture	0.035%	1,000 to 4,000	0.9
	1–10% moisture			0.65
	>10% moisture			0.4
Andujar, Spain	Uncovered	0.12%	12,800	10
	Clay cover			0.08
	Soil, sand, clay cover			0.024
Western areas, USA	Dry, no cover	0.1–0.3%	15,000 to 30,000	10–40
	Dry, clay–silt cover			0.3–0.7

^a A detailed compilation for Australian uranium mill tailings is given by (Mudd 2008b)

mines, UNSCEAR estimated normalised radon releases for numerous uranium mines in Canada, Australia and eastern Germany (GDR) ranging from 1.2 to 1800 GBq/t U₃O₈, averaging approximately 13 GBq/t U₃O₈ (UNSCEAR 1993). There does not appear to be any major difference in normalised radon releases between underground, open cut or in-situ leach projects, although the UNSCEAR estimates and others available appear to be very crude and approximate.

Uranium mining gives rise to major sources of radon loads and activities, which in turn can give rise to major exposure scenarios for workers or nearby communities. Modern uranium mine design and operation requires substantive effort to address radon issues, and requires constant monitoring and vigilance to ensure compliance with radiation exposure standards.

6.2 Mineral sands mining and milling

Mineral sands mining processes beach or placer sands for the recovery of various heavy minerals such as rutile (TiO₂), ilmenite (FeTiO₃), zircon (Zr(SiO₄))

and monazite (a rare earths-phosphate mineral, e.g. (Ce, La)PO₄) (Lottermoser 2007). The proportion of each mineral in the heavy mineral fraction is variable across deposits. Although monazite typically comprises 1% of the heavy mineral fraction in Australian mineral sands resources, monazite itself contains radiologically significant impurities of thorium (²³²Th, up to several percent) and uranium (Mason et al. 1988). Monazite from Guarapari, Brazil, contains 8–12% ThO₂ (Cockell et al. 2007).

The separation processes applied to heavy mineral sands leads to concentration of the radioactive monazite into a specific concentrate, potentially giving rise to significant radiation exposure issues due to thoron, dust, gamma radiation (due to ²¹⁴Bi) as well as radon. A compilation of the thorium and uranium content of Australian and Brazilian mineral sands fractions is given in Table 7.

In Western Australia in the early 1990s, about 30 million tonnes of material was mined annually, typically containing up to 60 mg/kg thorium and 20 mg/kg uranium (Hewson and Upton 1996). After processing, individual waste streams could contain thorium ranging from 100 to 30,000 mg/kg

Table 7 Concentration ranges of thorium (^{232}Th) and uranium (^{238}U) in Australian and Brazilian heavy mineral sands fractions (Bq/kg) (UNSCEAR 1993; Malanca et al. 1998)

	Australia ^{232}Th	Australia ^{238}U	Brazil ^{232}Th	Brazil ^{238}U
Ore	60–200	40	–	–
Heavy mineral concentrate	1,000–1,300	<100	2,900–60,000	480–4,000
Ilmenite	600–6,000	<100–400	1,765	461
Leucoxene	1,000–9,000	250–600	–	–
Rutile	<600–4,000	<100–250	–	–
Zircon	2,000–3,000	200 to 400	473	4,409
Monazite	600,000–900,000	10,000–40,000	187,00–196,800	12,090–13,760
Xenotime	180,000	50,000	–	–
Average soil or rock	40	40	–	–

(i.e. 0.01–3% thorium), though the more highly radioactive waste streams were generally smaller in mass. At Byron Bay, on the central coast of New South Wales, Australia, the tailings from mineral sands mining and processing, including low grade monazite concentrates, were used in various low-lying areas of the township for urban re-development, leading to above background radiation exposures in some residences (Gandy and Colgan 1983). A similar case study is known for Capel in Western Australia (King et al. 1983).

6.3 Phosphate mining and milling

Phosphate ore often contains elevated levels of uranium compared to typical soils (thorium is similar to soils), though it is variable from 30 Bq/kg for Kovdor, Russia, to 1,500 Bq/kg in Florida, USA (equivalent to 2.4–121 mg/kg, respectively) (UNSCEAR 2000). The extraction of uranium as a by-product from phosphate mining has been minor but perhaps important, probably of the order of less than 20% of cumulative global uranium production (Mudd and Diesendorf 2008).

Radon issues primarily arise with the processing wastes, mainly phosphogypsum, as well as any use of by-products or other wastes (including liquid wastes). The ^{238}U decay chain is usually close to secular equilibrium, with the radium (^{226}Ra) activity of phosphogypsum typically about 900 Bq/kg (depending on the origin of the ore) (UNSCEAR 1993). The management of associated radon issues in phosphate mining and processing will therefore depend on various site specific factors, such as ore grade,

climate, waste management practices, solid waste use, and so on (e.g. phosphogypsum use in building materials). Estimates of the radon released by typical phosphate projects are 820 GBq/year for a 0.7 Mt/year phosphoric acid facility and 221 GBq/year for a 375 kt/year fertilizer facility (UNSCEAR 2000).

6.4 Oil and gas extraction and processing

The extraction and processing of oil and gas represents a potentially major global radon source term, due mainly to the volumes produced and consumed. The activity of radium, and thereby radon and progeny, is highly variable in oil–gas projects across the world and can be expected to be closely linked to the source petroleum field, though published data is not widespread. During processing, the major risks are the exhalation of radon and buildup of radium in pipe scale. Radium scale is particularly important due to the decay chain leading to bismuth (^{214}Bi), which is a strong gamma emitter. Estimates of the radon released by typical oil–gas facilities are 540 GBq/year for a 3.5 Mt/year oil facility, 500 GBq/year for a 72 Gm³/year gas facility, while a 400 MWe gas-fired power plant is estimated to release 230 GBq/year of radon (UNSCEAR 2000).

Reed et al. (1991) presented data on radium buildup in North Sea and Louisiana oil–gas production facilities:

- *Pipe scales*: 26,000–286,000 Bq/kg;
- *Oil production separator sludges*: 1,000–823,000 Bq/kg;
- *Gas separator sludges*: 2,000–19,000 Bq/kg;
- *Gas separators*: 200–55,000 Bq/kg.

In Poland, oil–gas brines contain a maximum of 258 Bq/l radium (^{226}Ra) (Skowronek et al. 2005). In offshore petroleum exploration near Darwin, Australia, pipe scale contained ^{226}Ra – ^{228}Ra activities up to 600,000 Bq/kg (Cassels and Waite 2001). A study of oil–gas facilities in Nigeria has shown elevated gamma radiation ranging from 0.1 to 15 $\mu\text{Sv/h}$, with radium activity measured in scale up to 200,000 Bq/kg (Elegba and Funtua 2005).

6.5 Gold mining and milling

Commonly, there is only background concentrations of uranium or thorium in most gold ores, with consequent radiation exposures being low and minor in comparison to other occupational health issues in gold mining (e.g. silicosis). There are some exceptions, however, most notably being the gold–uranium ores of South Africa.

Since the discovery and development of the gold fields of South Africa in the late nineteenth century, more than six billion tonnes of gold ore has been mined and processed to produce $\sim 51,000$ tonnes of gold (Mudd 2007), including about 765 million tonnes of ore grading $\sim 0.022\%$ U_3O_8 which was processed to produce about 175,000 tonnes U_3O_8 (Mudd and Diesendorf 2008). The low-grade uranium content of most of the gold–uranium tailings, left almost entirely as tailings on the surface and adjacent to major populations, has led to a major and ongoing radiological exposure issue (Lindsay et al. 2004; Tsela and Zituta 2006). Additionally, there have been major issues with regards to radiation exposure for gold mine workers, including underground miners and those in the mills.

A recent study of Ghana found mean radon activities in underground gold mines of 350–445 Bq/m³, leading to exposures of about 1.83 mSv/year (mining only) (Darko et al. 2005). Surface outdoor radon activities ranged from 24 to 41 Bq/m³ in the mill, tailings dams, stockpiles and shafts/declines, leading to exposures of about 0.13–0.17 mSv/year. Uranium (^{238}U) and thorium (^{232}Th) concentrations were low and essentially background.

6.6 Coal mining and combustion

As with oil and gas, the presence of elevated uranium (or thorium) is highly variable across coal fields

around the world. There are a very small number of coals in the world which contain uranium at potentially economic grades, such as the former Freital coal mine in eastern Germany at 0.1–0.3% U_3O_8 , the dormant Okrzeszyn coal mine in Poland at 0.04–0.12% U_3O_8 , or the dormant Belskoie coal mine in Russia at 0.04–0.12% U_3O_8 (IAEA 1996). Although uranium-rich coal deposits have not been widely processed in the past for their uranium content, when combusted in coal-fired power stations they can cause major releases of radionuclides to the surrounding region as well as enriching the residual fly ash in radionuclides.

In Poland, hard rock coals contain a maximum of 159 Bq/kg radium (^{226}Ra), while associated waste rock contains 122 Bq/kg radium (Skowronek et al. 2005). Pond sediments and pipe scales at coal mines contained radium up to 157,000 Bq/kg radium. A survey of underground coal mines in the United Kingdom gave radon activities of 27–1,244 Bq/m³, with radon progeny being 4–40 mWL, showing significant variability between and within mines (Page and Smith 1992). In New South Wales, Australia, coal ash has ^{226}Ra ranging from 88 to 370 Bq/kg, ^{238}U from 70 to 167 Bq/kg and ^{228}Th from 91 to 261 Bq/kg; ^{222}Rn very low at 2.1–6 mBq/m²/s while ^{220}Rn was 20–86 mBq/m²/s (Zahorowski et al. 1994)—all considerably lower than average Australian background fluxes. At the Figueira coal mine in southern Brazil, within a geologic province containing known uranium deposits, radon activities averaged about 1,700 Bq/m³ (range 200–6,100 Bq/m³), leading to exposures estimated at 2.1 WLM (range 0.2–7.2 WLM) (Veiga et al. 2006). As such, any assessment of radon and progeny issues associated with coal mining needs to be undertaken on a site-specific basis.

6.7 Other mining

A recent survey of a large, long abandoned tin mine in Cornwall, UK, from which a very small amount of uranium ore was also extracted a century ago (for radium), showed radon activities reaching as high as 3,932,920 Bq/m³ some 52 m from the tunnel entrance (claimed as one of the highest radon measurements ever recorded in Europe)—with radon activity at 1 m height still being 2,154,560 Bq/m³ (Gillmore et al. 2002). The authors, adopting an approximate

equilibrium factor of 0.5, estimated that the radiation exposure for a 2 h visit at the upper value of 4 MBq/m³ would be 62 mSv *per 2 h visit*.

The mining and processing of mercury ore over many centuries in Slovenia has led to a situation with significantly elevated radon activities in the town of Idrija due to various scattered mine wastes and slags. One apartment measured an indoor radon activity of between 7,300 and 15,000 Bq/m³ while outdoor radon activity averaged nearly 100 m³ in the town centre (Križman et al. 1996).

Another major ore type which may face radon issues is rare earths, commonly due to monazite but sometimes due to uranium or thorium being present also. In Australia, two important rare earth deposits at Nolan's Bore in the central Northern Territory and at Mt Weld in central Western Australia both contain radiologically significant uranium and thorium concentrations (e.g. Nolan's Bore has 18.6 Mt of ore grading 3.1% rare earth oxides, 0.021% U₃O₈ and ~0.5–0.7% ThO₂; AR 2007, 2008). Historically, monazite has been an important source of rare earths and, as noted earlier, involves significant radiological issues.

7 Brief review of health issues

As noted in the historical review, health problems in areas of high radon activities have long been known, such as the Erzgebirge. The link between radon and health impacts, however, has only been suggested since the early twentieth century—though not widely scientifically accepted until the 1960s (Jacobi 1993; NAS 1999a). By the 1930s, based on the lung cancer rates in German-Czech mines, radon was being suggested as the main cause of the health impacts, but the quantitative evidence was not sufficiently clear, especially the role of radon progeny. In the 1950s, work done by William F Bale and John Harley in the USA demonstrated that the major radiological dose was actually delivered to lung tissues by the progeny and not the radon gas (Jacobi 1993). Combined with epidemiological studies emerging by the early 1970s, it became clearer that high radon and progeny activities were associated with impacts such as lung cancers. In the same decade it also became clear that residential dwellings could also allow radon and progeny to accumulate to exposures

sometimes as high as uranium miners. Based on combined studies and evidence, the US EPA now states environmental radon as the largest cause of lung cancer in non-smokers in the USA (e.g. USEPA 2007). Radon is commonly described as the most extensively studied carcinogen (e.g. Appleton 2005).

There are two principal exposure scenarios for radon: (i) high activities and exposure with good relationships to health impacts (e.g. uranium mining), or (ii) low activities and exposures and somewhat uncertain links between exposures and health impacts (e.g. natural background radon). The evidence for radon's carcinogenic nature is derived from molecular, cellular, animal and human (epidemiological) studies accumulated over many decades (NAS 1999a). This section will present a brief review of the mechanisms for radon-induced carcinogenesis, followed by a review of health issues for high exposures and finally low exposures. Given the ongoing controversy over low radon and progeny exposures, this section is intended as a guide only. For further details, see the major studies, such as BEIR-VI (NAS 1999a) or BEIR-VII Phase 2 (NAS 2006).

7.1 Mechanisms for radon-induced carcinogenesis

The radioactive decay of radon through its progeny to a stable lead isotope involves several alpha and beta decays as well as significant gamma radiation. A single radon atom can therefore impart notable energy at the molecular or cellular level, having the potential to cause major genomic changes in a cell resulting in mutations or other transformations (NAS 1999a).

For the most common inhalation exposure situations, as noted previously, it is the progeny which is the major cause of the imparted energy (see Table 1). The issue is therefore the extent of the radon that decays while inside the lung, and the depth to which the active progeny can reach inside the lung. Finer particles can reach deeper into the lung, and hence the attached–unattached fraction is critical to consider for lung dosimetry. Radon, as a noble gas with a 3.82 day half-life, is most likely to be exhaled before decaying.

According to the most recent 'Biological Effects of Ionising Radiation' (BEIR) committee study into radon exposure and cancer ('BEIR-VI'; NAS 1999a), there is convincing evidence that most cancers are of

Table 8 Summary of global epidemiological studies for uranium or other mine-workers (BEIR VI study) (NAS 1999a)

Cohort	Mineral	Period of follow-up	Number of miners	Person years	Average cumulative exposure (WLM)	Exposure rate (WL)	Lung cancer deaths	Age at first exposure	Year of first exposure	Length of follow-up	Excess relative risk per WLM (ERR/WLM)
Yunnan, China	Tin	1976–1987	13,649	134,842	286.0	1.7	936	18.8	1955.6	10.2	0.16 (0.1–0.2)
Bohemia, Czech Republic	Uranium	1952–1990	4,320	102,650	196.8	2.8	701	30.1	1951.0	25.2	0.34 (0.2–0.6)
Colorado Plateau, USA	Uranium	1950–1990	3,347	79,556	578.6	11.7	334	31.8	1953.0	26.3	0.42 (0.3–0.7)
Ontario, Canada	Uranium	1955–1986	21,346	300,608	31.0	0.9	285	26.4	1963.8	17.8	0.89 (0.5–1.5)
Newfoundland, Canada	Fluorspar	1950–1984	1,751	33,795	388.4	4.9	112	27.5	1954.1	23.3	0.76 (0.4–1.3)
Northern Sweden	Iron Ore	1951–1991	1,294	32,452	80.6	0.4	79	27.4	1934.1	25.7	0.95 (0.1–4.1)
New Mexico, USA	Uranium	1943–1985	3,457	46,800	110.9	1.6	68	28	1965.6	17	1.72 (0.6–6.7)
Beaverlodge, Canada	Uranium	1950–1980	6,895	67,080	21.2	1.3	56	28	1962.6	14	2.21 (0.9–5.6)
Port Radium, Canada	Uranium	1950–1980	1,420	30,454	243.0	14.9	39	27.6	1952.3	25.3	0.19 (0.1–0.6)
Radium Hill, South Australia	Uranium	1948–1987	1,457	24,138	7.6	0.7	31	29.2	1956.0	21.9	5.06 (1.0–12.2)
Central & Western France	Uranium	1948–1986	1,769	29,172	59.4	0.8	45	29.5	1956.8	24.7	0.36 (0.0–1.2)

monoclonal origin. When combined with mechanistic processes for alpha decay and the probability of damaging cells, the BEIR-VI committee concluded that there is more evidence supporting a linear no threshold ('LNT') model for radon exposures. That is, any increase in radon exposure linearly increases the chance of cancer. The BEIR-VI committee also noted, however, that it could not exclude the possibility that a threshold dose occurred at low exposures. The BEIR-VI committee's view on the adequacy of the LNT model is supported by the United Nations Scientific Committee on the Effects of Atomic Radiation (UNSCEAR) (see UNSCEAR 2000). The more recent and broader BEIR-VII study into low level radiation exposure re-affirmed the LNT model for radiation exposures (NAS 2006).

The model of cause-effect between radon and progeny exposures and effects such as lung cancer is at the heart of the debate about radon. A number of critical factors need to be considered in quantifying this relationship. Firstly, the combination of cigarette smoking and radon exposures is argued by BEIR-VI as synergistic, that is the combined effect of these two actions is greater than the individual sum alone. Secondly, the exposure conditions are different between uranium miners and residential dwellings, such as concentrations and equilibrium factors (exposures in miners are about one order of magnitude higher or more than indoor exposures; NAS 1999a). Finally, issues such as gender or age can also be important in the effects of exposure.

7.2 Health impacts at high exposures

The exposures of uranium miners, especially underground miners during the 1940s to late 1960s, was particularly high. The BEIR-VI committee reviewed all available epidemiological studies on uranium and non-uranium miners, with the key results given in Table 8. The combined results were used to develop the quantitative risk models in NAS (1999a), and give an indication of the 'excess relative risk' (ERR) per exposure for the various studies. The lung cancer rate relative to mean radon exposure is given in Fig. 7.

7.3 Health impacts at low exposures

The impacts of low radon exposures remain somewhat controversial. At ambient activities commonly

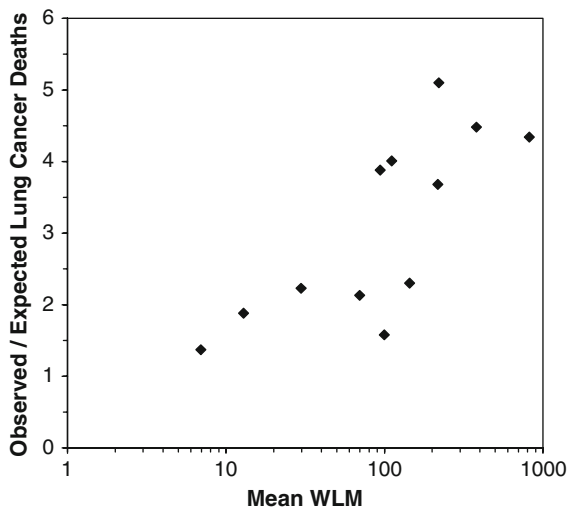


Fig. 7 Lung cancers versus radon exposure (adapted from NRPB 2000, cited by Appleton 2005)

encountered in indoor or outdoor settings, exposures are considerably lower than miners, although they can reach high levels at the lower end of typical miner exposures. Some of the major problems with many background radon versus miner studies is the different study designs, the different radon measurement techniques employed, varying radon activities encountered, variable information on confounding factors, and most importantly limited sample size (Wichmann et al. 2005). Additionally, some researchers suggest that small regions with elevated radon do not correlate to increases in lung cancers, such as radon up to 3,700 Bq/m³ at Ramsar, Iran (Mortazavia et al. 2005).

A major study which has recently begun to address the many issues associated with low exposure radon studies is the ‘Iowa Radon Lung Cancer Study’, described by Field et al. (2000). The Iowa study recently completed Phase 1 and is presently close to finishing Phase 2. At its heart is a significantly improved field methodology for monitoring and assessing the cumulative radon and progeny exposure over time, thereby reducing the major uncertainty associated with most studies to date. The Phase 1 results show that cumulative radon exposure is a contributor to lung cancer incidence (Field et al. 2000).

Similarly, an extensive compilation and combined analysis of 13 residential radon-lung cancer studies covering 9 European countries was recently published

(Darby et al. 2005). This study showed that the risk of lung cancer increased by 16% (95% confidence interval, 5–31%) per 100 Bq/m³ increase in radon after correction for random uncertainties in measuring radon concentrations. Importantly, the study again demonstrated that the LNT model was the best basis to understand exposure risk. Results from similar studies in North America also arrived at the same conclusions (Krewski et al. 2005).

8 Remediation approaches

There are two principal approaches to prevent, minimise or remediate radon problems—source reduction and dilution.

For most indoor issues, simple design and construction techniques are used to limit radon entry in the first place, and where still necessary, appropriate ventilation fans can be installed to extract ambient air and direct the radon to the external atmosphere (e.g. Lugg and Probert 1997; Groves-Kirkby et al. 2007). In any program or effort aimed at reducing indoor radon, it is critical to account for potential radon sources in building materials, as these can hamper the effectiveness of remediation designed to address underlying geologic radon sources (Groves-Kirkby et al. 2007). Some common techniques include (Lugg and Probert 1997):

- *Sub-floor depressurisation*: a wind or electric powered ventilation system is installed beneath the ground floor to extract ambient radon-rich air derived from underlying geology and eject it to the atmosphere (sumps may be included);
- *Floor sealing*: this involves placement of a low permeability material across the floor, especially focussing on filling in cracks and gaps. Given the difficulty in sealing 100% of open voids, sealing is often used in conjunction with other approaches;
- *Positive pressure*: by creating a slight positive pressure inside a building, it is possible to suppress the rate of radon entry. For this approach to be effective, the building needs to be air tight;
- *Increased ventilation*: this is essentially achieving a dilution of the radon through increased air flow through a building, however, the reductions are typically small and not sufficient for high radon scenarios;

- *Air cleaners*: given that the principal radiation exposure is derived from the reactive progeny, research has investigated this technique, although results showed only a minor reduction (the method is also more expensive than other more effective options).

For mining situations, outdoor radon is commonly considered to be sufficiently low due to atmospheric dispersion (especially diurnal processes, though atmospheric inversions may limit dispersion for brief periods).

The principal area where radon and progeny levels can accumulate very easily is in underground mining. As noted in the uranium mining section, the early years of mining uranium ores invariably involved significant to extreme radon and progeny exposures. As the link between this exposure and lung cancer incidence was more widely accepted, especially by the late 1960s, air quality standards were adopted which led to major ventilation systems being installed and significant reductions in radon and progeny exposures (e.g. Australian case study, Table 5). The design of underground mine ventilation systems to achieve desired radon and progeny levels is a complex and specialist field, as it involves fluid dynamics, mine design, particulates (especially their particle size distribution), radon exhalation, and attached–unattached progeny, as well as the interactions between aspects such as particulates and the attached progeny fraction; see papers in Gomez (1981).

Alternative research has investigated the efficacy of sealants on the walls of underground mines, such as polymers, though they are of arguable effectiveness compared to a well-engineered ventilation system (especially with respect to cost/benefits and other safety issues such as damage or fire) (see Franklin 1981). Other approaches include the use of bulkheads or backfilling to seal off sections of a mine from active operations and management, options commonly incorporated into existing or proposed underground mines. In addition, time workers spend in given areas can also be controlled to minimise exposures.

A major legacy of mining is the solid wastes remaining after mine closure. The tailings, low-grade ore stockpiles and waste rock are often locally significant radon sources, especially the tailings,

and they must all therefore be addressed during mine rehabilitation (Mudd 2008b). The traditional engineering approach is to design and construct a single or multi-layered soil cover over such solid wastes to minimise the radon flux emanating at the surface (e.g. Rum Jungle, Australia, Allen and Verhoeven 1986; Schlema-Alberoda, Germany, Schmidt and Regner 2005). There has been extensive research into the preferred soils and designs for soil covers, especially taking into account different climatic conditions (eg. sub-arctic of northern Saskatchewan in Canada versus the Rössing uranium mine in the Namib Desert of Namibia). The primary design approaches and models include the RAECOM code, developed by Rogers et al. (1984) for the US Nuclear Regulatory Commission, as well as a mixture of applied and theoretical models (e.g. Hart et al. 1986; IAEA 1992b; Ferry et al. 2001, 2002) (including the online implementation of the RAECOM code by Diehl (2008b)).

The most recent model to address radon in engineered soil covers is the coupled unsaturated flow-climate model Vadose/W (Krahn 2004), developed specifically to address the complex interactions of climatic forcing conditions (rainfall, evaporation, transpiration, temperature, wind speed, humidity, etc.), moisture flow in unsaturated (or saturated) soils as well as gaseous transport through such soils to the atmosphere. Vadose/W is intended to be used as an engineering design tool for soil covers in acid mine drainage or radon situations, and is arguably the most theoretically rigorous model yet developed for soil covers.

A common challenge is that although models are used to predict the effectiveness of different engineering designs, there appears to be very little monitoring of long-term cover performance. In Australia, the soil covers applied at the former Nabarlek, Rum Jungle, Radium Hill and Port Pirie sites have all faced ongoing problems of weeds, erosion, maintenance or poor construction (Mudd 2008b). At the Rum Jungle site, extensive acid mine drainage pollution continues despite the soil covers, though there appears to have been no studies of the performance of the covers with respect to radon fluxes (the rehabilitation target was set somewhat arbitrarily at 0.14 Bq/m²/s; Mudd 2008b). The extensive remediation efforts undertaken in United States and eastern Germany, where soil covers were

constructed over substantive quantities of waste rock and tailings facilities, await a more comprehensive review and analysis—as well as the test of time.

A sound rehabilitation objective for uranium mining projects should be to return radon fluxes to pre-mining levels (Mudd 2008b). At the former Nabarlek uranium project in the Northern Territory, Australia, rehabilitation works have actually achieved an overall reduction in radon flux compared to pre-mining (Bollhöffer et al. 2006). Prior to mining, very high-grade ore ($>1\%$ U_3O_8) outcropped at the surfaced, leading to radon fluxes from 3.7 to $44.0 \text{ Bq/m}^2/\text{s}$ (Clark et al. 1981). Mining saw the tailings buried up to half the depth in the mined out pit ($\sim 100 \text{ m}$ deep), with low grade and waste rock filling the remainder of the pit—thus leading to a weaker radon source in the near surface materials and soil covers averaging about $1.0 \text{ Bq/m}^2/\text{s}$ over the same region (Bollhöffer et al. 2006). Recently, however, a small area (0.44 ha) has shown significant erosion leading to exposure of the underlying radioactive wastes, giving a radon flux of $6.51 \text{ Bq/m}^2/\text{s}$ (Bollhöffer et al. 2006). At many projects in Australia, it appears very difficult to achieve a pre-mining radon flux after rehabilitation, especially in the long term (Mudd 2008b).

9 Conclusions

Radon is a ubiquitous natural carcinogen derived from the three primordial radionuclides of the uranium series (^{238}U and ^{235}U) and thorium series (^{232}Th). In general, it is present at very low concentrations in the outdoor or indoor environment, but a number of scenarios can give rise to significant radiological exposures. Historically, these scenarios were not recognised, and took many centuries to understand the links between the complex behaviour of radon and progeny decay and health risks such as lung cancer. However, in concert with the rapid evolution in the related sciences of nuclear physics and radiological health in the first half of the twentieth century, a more comprehensive understanding of the links between radon, its progeny and health impacts such as lung cancer has evolved. It is clear from uranium miner studies that acute occupational exposures lead to significant increases in cancer risk, but chronic or sub-chronic exposures, such as indoor residential settings,

while suggestive of health risks, still entails various uncertainties. At present, prominent groups such as the BEIR or UNSCEAR committees argue that the ‘linear no threshold’ (LNT) model is the most appropriate model for radiation exposure management, based on their detailed review and analysis of uranium miner, residential, cellular or molecular studies. The LNT model implies that any additional or excess exposure to radon and progeny increases overall risks such as lung cancer. A variety of engineering approaches are available to address radon exposure problems. Where high radon scenarios are encountered, such as uranium mining, the most cost effective approach is well-engineered ventilation systems. For residential radon problems, various options can be assessed, including building design and passive or active ventilation systems. Overall, radon will continue to be an ever-present carcinogen and requires eternal vigilance wherever it is encountered—in mining, processing industries, building materials, caves, or even residential homes.

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The future of Yellowcake: A global assessment of uranium resources and mining

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HIGHLIGHTS

- An extensive data set on global uranium resources and classified by deposit type.
- Comprehensive analysis of key trends, such as ore grades and recovery rates.
- Energy and carbon intensity of production shows an increase as ore grades decline.
- Mine rehabilitation often shows poor success or accounts of long-term effectiveness.
- Real constraints on nuclear power remain safety and costs compared to alternatives.

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ABSTRACT

Uranium (U) mining remains controversial in many parts of the world, especially in a post-Fukushima context, and often in areas with significant U resources. Although nuclear proponents point to the relatively low carbon intensity of nuclear power compared to fossil fuels, opponents argue that this will be eroded in the future as ore grades decline and energy and greenhouse gas emissions (GGEs) intensity increases as a result. Invariably both sides fail to make use of the increasingly available data reported by some U mines through sustainability reporting – allowing a comprehensive assessment of recent trends in the energy and GGE intensity of U production, as well as combining this with reported mineral resources to allow more comprehensive modelling of future energy and GGEs intensity. In this study, detailed data sets are compiled on reported U resources by deposit type, as well as mine production, energy and GGE intensity. Some important aspects included are the relationship between ore grade, deposit type and recovery, which are crucial in future projections of U mining. Overall, the paper demonstrates that there are extensive U resources known to meet potential short to medium term demand, although the future of U mining remains uncertain due to the doubt about the future of nuclear power as well as a range of complex social, environmental, economic and some site-specific technical issues.

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1. Introduction

Despite the utopian promise of electricity ‘too cheap to metre’,¹ nuclear power remains a minor source of electricity worldwide. In 2010 nuclear power accounted for 5.65% of total primary energy supply and was responsible for 12.87% of global electricity supply (IEA, 2012) – and both contributions have effectively been declining through the 2000s. Concerns about hazards and unfavourable economics have effectively slowed or stopped the growth of nuclear energy in many Western countries since the 1980s, and while there has been some renewed interest in the USA (the country with the most reactors), the nuclear

reactor fleet is ageing and 24 of the new proposed projects have been delayed, including 8 indefinitely (Schneider et al., 2013). However, there is still growth in nuclear energy in some countries, mostly in China, Russia and India (mainly via central planning by the national government). Over the next 15–20 years, many more nuclear power stations will reach retirement age than those contracted for actual construction (Schneider et al., 2013; Sovacool, 2011).

World primary energy production and electricity generation are given in Table 1, including the International Energy Agency's (IEA) projection of world energy and electricity demands in 2035 under the scenario of current energy policies continuing. It can be seen that, despite modest capacity growth (1.4x), nuclear's share of electricity declines by 2035 to just 9.68% – and yet some argue that it is the only logical choice for the world's power hungry demands (eg. Suppes and Storvick, 2007). Of particular note is the massive proportional increase in renewable energy-derived electricity by 2035 – biomass (3.7x), geothermal (3.2x), wind (6.3x), solar photovoltaics (16.4x), tidal (32x)

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¹ Although this statement was made by Lewis I. Strauss on 16 September 1954 mainly in reference to the future potential of hydrogen fusion power, it was made during the early years of the Cold War when nuclear fission power was being actively promoted through the ‘Atoms for Peace’ program and remains a useful metaphor generally.

Table 1

World primary energy demand and electricity generation by source for 2009 and projected 2035 demand under current energy policies (IEA, 2012).

Source	Primary energy demand (EJ)		Source	Electricity generation (TWh)	
	2010 (%)	2035 (%) ^a		2010 (%)	2035 (%) [#]
Coal	145.4 (27.29%)	176.6 (24.53)	Coal	8687 (40.57%)	16,814 (41.66%)
Natural gas	114.7 (21.52%)	171.9 (23.88%)	Gas	4760 (22.23%)	9342 (23.14%)
Crude oil	172.2 (32.31%)	194.9 (27.07%)	Oil	1000 (4.67%)	673 (1.67%)
Biomass & other	53.5 (10.03%)	78.8 (10.94%)	Biomass	331 (1.55%)	1212 (3.00%)
Nuclear	30.1 (5.65%)	47.6 (6.62%)	Nuclear	2756 (12.87%)	3908 (9.68%)
Hydro	12.4 (2.32%)	20.4 (2.84%)	Hydro	3431 (16.03%)	5350 (13.25%)
Other renewables	4.7 (0.88%)	29.7 (4.13%)	Wind	342 (1.60%)	2151 (5.33%)
			Geothermal	68 (0.32%)	217 (0.54%)
			Solar PV	32 (0.15%)	524 (1.30%)
			Solar thermal	2 (0.01%)	141 (0.35%)
			Tidal & wave	1 (0.00%)	32 (0.08%)

EJ – exa (10¹⁸) joules; TWh – tera (10¹²) watt-hours (1 EJ = 277.78 TWh or 1 TWh = 0.0036 EJ). PV – photovoltaics.

^a IEA current policies scenario.

and solar thermal (70.5×). There are also scenarios for achieving 100% renewable energy on a global scale (Jacobson and Delucchi, 2009; Sørensen and Meibom, 2000), 100% renewable energy for the European Union (Zervos et al., 2010) and 100% renewable electricity for New Zealand (Mason et al., 2010) and Australia (Elliston et al., 2012; Wright and Hearps, 2010), among others.

There are numerous critical issues facing the energy and electricity sectors globally, with arguably the most important being (in no particular order):

- greenhouse gas emissions released by fossil fuel sources contributing to anthropogenic climate change;
- peak oil and the end of the era of cheap conventional oil;
- social and environmental impacts of unconventional oil and gas extraction, such as tar sands, shale gas (and liquids) and coal seam gas;
- concerns about energy security, especially ongoing supply issues and maintaining resources; and
- economic competitiveness of energy sources.

It is argued by some (e.g. Suppes and Storvick, 2007; UMPNERT, 2006) that nuclear power can effectively address most of these problems on the basis that it has lower carbon intensity than fossil fuels, is largely independent of oil or coal supplies and, if plutonium breeder reactors can be made commercial (while ignoring or, at best, downplaying severe proliferation risks), could provide a substantial energy resource which could last a considerable period of time.

In this context, what is the basis upon which nuclear power could be argued to increase beyond the IEA's current projections? In other words, can the nuclear fuel chain be considered a sustainable option for future electricity generation? These questions are more than of minor consequence to address. The contribution of nuclear power to nuclear weapons proliferation remains paramount and urgent (eg. North Korea, India, Pakistan, Iran, Israel, etc.) and nuclear reactor safety and long-term stewardship of nuclear waste (ie. for greater than 100,000 years) still represent fundamental concerns, especially when comparing these risks to those of energy efficiency and various renewable energy technologies such as baseload solar thermal, solar photovoltaics, wind and so on.

This paper presents a detailed study of the ability for uranium (U) resources and mining to meet possible future scenarios for expanded nuclear power. An extensive array of data is compiled and analysed, focussing on known economic resources by deposit type, mine production and U recovery efficiency, including case studies of several key U projects such as Ranger and Olympic Dam in Australia, Rössing in Namibia and McArthur River and the Elliot Lake field in Canada, with these aspects then briefly discussed in light of other sustainability issues such as energy and greenhouse gas emissions (GGEs) intensity. The paper thus provides a comprehensive assessment of the most important factors which will affect the future of the U mining sector which, albeit a somewhat small player in value terms, attracts significant political and

corporate support while remaining deeply controversial in the public energy debate.

2. Background to Uranium

2.1. Uranium and Its Uses

There are two primary uses for uranium – as fuel in a nuclear power plant to produce electricity, or as the ingredient for nuclear weapons. Natural uranium consists primarily of two isotopes, uranium-238 (²³⁸U) and uranium-235 (²³⁵U), comprising 99.28% and 0.715%, respectively (Barré, 2010). The fissile ²³⁵U can be forced to undergo a chain reaction and split (i.e. fission) and release energy in the process. In a weapon, the release of energy is uncontrolled and leads to a severe explosion. In a nuclear power plant, the ²³⁵U fission rate is controlled to limit energy release, which is used to produce high temperature steam to drive a turbine to generate electricity. The ²³⁸U is fertile and tends to capture neutrons and undergoes radioactive decay to form plutonium-239 (²³⁹Pu, which is highly fissile and can be used in nuclear weapons or as reactor fuel).

Under the Nuclear Non-Proliferation Treaty (NPT), which all but five countries worldwide have signed (namely India, Pakistan, Israel, Iran and North Korea), the traditional nuclear powers of the United States of America (USA), Russia, United Kingdom (UK), France and China are all obliged to work towards disarmament (although progress is arguably going backwards on this front internationally; see Acheson et al., 2013). Based on the NPT and a host of related nuclear safeguards, treaties and conventions, it is effectively illegal to sell U for nuclear weapons (though if a country mines U, there is virtually nothing to stop it using this U for its own internal purposes such as weapons; which has been the case for all nuclear weapons powers to date). As such, this paper will exclude consideration of U in weapons, although in reality they remain a pivotal stumbling block in any consideration of U mining and nuclear energy's contribution to sustainable energy scenarios for the world. It should also be remembered that there is a small but ongoing U demand as a fuel for naval submarines and ships in the USA, Russia, UK, China and France, although such use will not be considered in this study (Russia also has a very small U demand for commercial nuclear-powered icebreaker ships for the Arctic).

Uranium is pre-dominantly found in mineral deposits on its own, although it can be found in conjunction with certain types of vanadium deposits, or, more rarely, in certain gold fields and a few copper deposits. There are a wide variety of deposit types (see later), and mining extracts and chemically refines the U to relatively pure oxide, mainly as tri-uranium octoxide concentrate (ie. >95% U₃O₈; though some projects may produce uranyl peroxide, UO₄).

Most nuclear power plants require the ²³⁵U fraction to be increased to ~3–5% ²³⁵U to improve the fission process in a reactor. Some reactors, such as the CANDU type, can use natural uranium, but these are only a

small proportion of the global nuclear reactor fleet. The process of increasing the fraction of ^{235}U is called enrichment, and is technically very challenging and extremely energy intensive. To facilitate enrichment, the uranium is commonly converted to a hexafluoride gas (ie. UF_6), and the enriched uranium is then converted back to an oxide (ie. UO_2) in order to make the fuel rods for the reactors.

In a reactor, the fission process leads to the formation of a wide range of highly radioactive elements and isotopes (fission products), as well as some of the ^{238}U being converted to ^{239}Pu . The radiation intensity of spent nuclear fuel is several orders of magnitude higher than the original U ore. Due to the extreme radioactivity, the spent fuel is generally called high level nuclear waste.

Overall, the use of U to produce nuclear-derived electricity leads to a chain of processes from mining to conversion, enrichment, re-conversion, fuel manufacture, reactor use, and finally high level waste management – often called the nuclear fuel chain. This paper focusses only on the mining stage, as this is a crucial aspect in underpinning possible long-term projections of nuclear power this century (if it is to have a future).

2.2. A very brief history of uranium mining

Since the discovery of radioactivity by French physicist Antoine-Henri Becquerel in 1896 and the quick follow-up discovery of the radioactive U decay product of radium by his assistant Marya Skłodowska (soon to be famous as Marie Curie), there have been arguably four major phases of U mining: (i) the ‘radium’ phase from ~1900 to 1940; (ii) the Cold War (or military phase) from 1941 to the 1960s; (iii) the civilian phase from the 1970s to the mid-1990s; and (iv) followed recently (since about 2003) by a resurgence in interest in U exploration and mining. At the start of each major phase, substantial concerns were raised about the extent of U resources – that is, the ability to meet rapidly growing demands. Very quickly, however, especially for each of the first three phases, new deposits and major fields were discovered that quickly led to large new supplies.

The radium mining phase saw the primary interest in the radioactive decay product of U – and radium reached some £300,000 per ounce (gold at that time was a mere £4.25/oz), mainly for medical uses in cancer treatment or scientific research (Mudd, 2005). Initially, radium was procured from known deposits in the Erzgebirge (‘Ore Mountains’) in eastern Germany and Joachimstal in Bohemia (today’s Czech Republic), the region from where Martin Klaproth had first isolated uranium in 1789, as well as the central Colorado Plateau in the USA. In 1922, the Belgians announced a rich discovery at Katanga in the Belgian Congo, and went on to effectively monopolise the world radium market for a decade. Another rich discovery was made at Great Bear Lake in northern Canada in 1930, giving Canada a strong position in radium, although this was also the time when serious concerns began to arise with respect to cancer rates in exposed workers (especially the famous radium dial painters in the USA who suffered horrendous bone cancers and other impacts from excessive occupational radium exposure; see Clark, 1997).

The modern nuclear era was powerfully ushered in with the bombings of Hiroshima and Nagasaki in Japan by the USA, courtesy of the secret military Manhattan Project which delivered the technology and weapons – half of the U was secured from Katanga, one quarter from Great Bear Lake, and the remainder from the Colorado Plateau (all previously radium–vanadium mines). Following the end of World War 2 and the rapid escalation of the Cold War arms race between the USA and the Soviets, U exploration and mining become a priority all over the world. Wide-reaching exploration, often by an armada of hopeful prospectors with Geiger Counters, discovered a vast array of new fields and numerous new U mines were quickly developed to supply the respective military programmes. Although the USA and UK governments had incentive programmes to find and mine uranium, within a decade these had to be scaled back due to the over-whelming success of finding new U. The major regions which saw a boom in U mining include the

Elliott Lake district of northern Ontario in Canada, the Four Corners (mid-west) region of the USA, the Erzgebirge of the then German Democratic Republic (GDR, part of the Soviet Union), Kazakhstan, the Witwatersrand gold fields in South Africa (where U was a by-product of gold), as well as minor production in other countries (eg. Australia, France, Eastern European countries). Typically, most ores were between 0.05–0.5% U_3O_8 in grade and deposits were typically from 1 to 20 kt U_3O_8 . With the scaling back of military purchasing in the early-to-mid 1960s, most of these regions began to wind down or collapsed altogether, although the low grade Elliott Lake district (~0.1% U_3O_8) continued with major subsidies from the Canadian Government.

By the late 1960s, nuclear power for civilian electricity was becoming an economic prospect, and the optimistic growth scenarios again raised the spectre of possible U shortages – this time the major mining companies were quickly encouraged to go exploring again but with success well beyond that experienced during the height of the Cold War. In northern Saskatchewan, several high grade deposits (0.5–5% U_3O_8) were discovered in the late 1960s to mid-1970s (eg. Key Lake, Rabbit Lake, Cluff Lake), with super-rich deposits discovered at McArthur River and Cigar Lake in the 1980s (both averaging ~15% U_3O_8). These new deposits were also very large, containing from 20 to 200 kt U_3O_8 each. In Australia, similar success was enjoyed in finding major new deposits, albeit at more typical grades of 0.05–0.5% U_3O_8 (eg. Ranger, Jabiluka, Olympic Dam, Yeelirrie). Conversely, across Africa, the giant but very low grade Rössing uranium deposit (~0.04% U_3O_8) in Namibia was recognised as a major project, along with major discoveries in Niger by the French. Globally, U resources surged and production shifted from older Cold War centres to new regions such as northern Saskatchewan, northern and central Australia, Namibia and Niger.

There were also a range of other key developments for the U industry in the 1970s. First of all, the price surged from a mere US\$13.2/kg U_3O_8 (ie. US\$6/lb U_3O_8) in February 1973 to a record high of US\$88.2/kg U_3O_8 (US\$43.4/lb U_3O_8) by May 1978. While the causes of this price hike can be debated, such as effects of the oil price spikes or concerns over future U supplies (despite mine production being in significant excess over civilian demand during this time), another factor was the emergence of a global cartel of U companies trying to manipulate the market for their own ends (see Venturini, 1982). In addition, major public concern began to develop regarding reactor safety, with the famous Hollywood film ‘The China Syndrome’ showing a potential reactor core meltdown scenario released on 16 March 1979 – just 12 days before the major accident at the Three Mile Island nuclear facility in Pennsylvania. Combined with the ongoing controversy over nuclear weapons and associated radioactive fallout, as well as improving knowledge of the long-term health effects from radiation exposure (including excess cancer rates in nuclear veterans and U miners), the optimism about nuclear power was left without any credibility.

On 26 April 1986 a full meltdown and reactor explosion at the Chernobyl nuclear complex in the Ukraine, leading to widespread radioactive fallout across the Soviet Union and Europe. By the late 1980s, nuclear power had not expanded as rapidly as the optimists had expected and this led to the U market being strongly over-supplied by mines – crashing the U price and numerous mines and fields in the process. A considerable portion of demand was then supplied by large inventories, held either by governments or power companies, leading to new mine production being lower than demand from the late 1980s to the present. In the mid-1990s, some of the nuclear weapons stockpile of highly enriched uranium (ie. >20% ^{235}U), was downblended to produce normal enrichment reactor fuel (ie. 3–5% ^{235}U). The programme, known as ‘Megatons to Megawatts’, has supplied about half of the nuclear reactors in the USA since this time, effectively providing about 20% of global U supply. Another aspect of the 1980s to 1990s was the considerable effort directed at remediating abandoned mines from the Cold War era, with programmes in the USA and a re-unified Germany each costing billions.

At the start of the 2000s, U prices were at historic lows (especially in real terms) of US\$16.42/kg U_3O_8 (US\$7.45/lb U_3O_8) but in early 2003

began to rise in concert with hopes by some that nuclear power might provide a solution to replacing coal-fired electricity and address climate change risks from greenhouse gas emissions associated with burning coal. Remarkably, by mid-2007 U spot prices had reached the staggering level of about US\$300/kg U₃O₈ (US\$136/lb U₃O₈) but quickly crashed to around US\$97/kg U₃O₈ (US\$44/lb U₃O₈ by early 2009 – and have stayed in this vicinity ever since.

The multiple reactor accident at the Fukushima Daichi complex in Japan following the severe earthquake and tsunami in March 2011 appears to have not only dampened U prices, but given further impetus for some countries to formalise their exit from nuclear power (eg. Germany), slowing nuclear growth in other countries (eg. China) and discouraging potential newcomer countries from pursuing nuclear power (eg. Bangladesh, Belarus, Jordan, Lithuania, Poland, Saudi Arabia and Vietnam). Although the IEA still projects very modest global growth in nuclear power capacity by 2035 (Table 1), this is mostly from planned economies and certainly not western-style democracies. The future demand for U remains very difficult to predict indeed.

Overall, uranium has had a roller-coaster history over the past century, arguably being one of the mining industries' most widely varying sectors – from major boom times to severe downturn several times. It is important to keep this history in mind when examining U resources and mining, as it can inform trends and suggest useful assumptions for the future of the sector.

3. Economic uranium resources

3.1. Uranium deposit types

Uranium can be found in a wide variety of mineral deposit types, mainly related to its geochemical versatility (Dahlkamp, 2009; Dahlkamp, 2010). According to the International Atomic Energy Agency (IAEA), there are eleven primary types of U deposits (IAEA, 2009b):

- | | |
|--|---------------------------------|
| 1. unconformity-related | 7. volcanic and caldera-related |
| 2. sandstone | 8. metasomatite |
| 3. hematite breccia complex [#] | 9. surficial |
| 4. quartz-pebble conglomerate | 10. collapse breccia pipe |
| 5. vein | 11. phosphorite |
| 6. intrusive | |

[#]commonly known as an iron oxide copper–gold (or IOCG) breccia.

In addition, minor deposit types include metamorphic, limestone-paleokarst and U-enriched coal deposits (IAEA, 2009b). Historically, most U production has been derived from sandstone, unconformity and quartz-pebble conglomerate deposits. It is common for a country to be dominated by a single deposit type (eg. South Africa by quartz-pebble conglomerates; Kazakhstan by sandstones), but not always (eg. numerous types in Canada, Australia and the USA).

3.2. Economic uranium resources: methodology

Given the political prominence of U as a perceived strategic resource, especially during the Cold War years, there is a reasonable amount of data on U resources since 1950, including:

- OECD-IAEA's 'Uranium: Resources, Production and Demand' (aka the 'Red Book', bi-annually 1965 to present; OECD-NEA and IAEA, var.);
- Natural Resources Canada's 'Canadian Minerals Yearbook' (1944 to present; NRC, var.);
- US Bureau of Mines 'Minerals Yearbook' (1933 to 1993; USBOM, var.);
- US Department of Energy's 'Uranium Industry Annual' (1992 to 2005; EIA, var.);
- South African Chamber of Mines 'Facts and Figures' (2004 to 2010; CMSA, var.); and
- Geoscience Australia's 'Australia's Identified Mineral Resources' (1992 to 2011; GA, var.).

In addition, other reports also provide data (eg. Australia, see Mudd, 2012). Numerous U companies also publish their ore reserves and mineral resources in annual corporate reports, as required for publicly-listed companies in developed countries (eg. Cameco, BHP Billiton, Rio Tinto, Uranium One, Paladin Energy, Denison Mines and others). This paper compiles an extensive and global data set on reported U mineral resources for 2011 by project (521 in total), and although the list is extensive, it is by no means complete since many countries do not allow full reporting.

This paper adopts the use of total mineral resources based on statutory reporting codes (eg. JORC, NI43-101, SAMREC), since these represent a reasonable basis for the long-term assessment of resource availability, as shown by Mudd et al. (2013a) for copper and Mudd et al. (2013b) for cobalt. The OECD/IAEA Red Book uses identified resources, which consists of reasonably assured resources and inferred resources and deposits categorised into predicted cost ranges (eg. <US \$40–80–130–260/kg U). Although this is different to normal mineral resource codes, they are broadly similar and provide a realistic basis for comparison (Lambert et al., 2009). Where a code-based resource is not reported, other sources of information are used if possible (eg. Red Book). It must also be stated that there are legitimate technical concerns regarding the reliability and accuracy of Red Book U resource data, since the issues of equivalence between countries in terms of price assumptions, costs, geological modelling, independent auditing and so on (which are some of the minimum requirements for almost all statutory codes), are poorly addressed and effectively remain at a reporting countries discretion. For the U deposit database maintained by the IAEA, or at least the version which is publicly available online, only ranges are reported for grades and tonnages (eg. IAEA, 2009b). However, where there is no data available for a given deposit, we will assume Red Book or IAEA data is sufficiently accurate for the purposes of this study. In addition, several deposits have been excluded from this compilation since they are located in national parks where mining is banned (eg. the Koongarra, Ranger 68, Austatom and Coronation Hill deposits in Kakadu National Park, Australia, which collectively are estimated to contain some 27 kt U₃O₈).

3.3. Economic uranium resources: results

The trends in remaining economic resources and country average ore grades are shown in Fig. 1, with Red Book and national resource data for 2011 in Table 2, and Tables 3–6 showing compiled projects by country, deposit types, largest size and highest and lowest ore grades. Graphs of ore grade versus ore tonnage and contained U₃O₈ versus ore grade are shown in Fig. 2, with cumulative frequency curves for ore grade and contained U₃O₈ in Fig. 3. In addition, there is a further 756 kt U₃O₈ in deposits reported in the Red Book but with no ore tonnage and grade data – suggesting global U resources of at least 11,798 kt U₃O₈ (or ~41% higher than the Red Book 2011 estimate of 8367 kt U₃O₈).

In comparing Red Book, national data and the compiled data from this study, many countries have much higher U resources in this study than both former sources. For example, Australia's Red Book and national estimates for 2011 were 2050 and 2143 kt U₃O₈, respectively, compared to this study showing some 3375.9 kt U₃O₈. For Sweden, the Red Book has 15.9 kt U₃O₈ but this study some 836.5 kt U₃O₈. This highlights the crucial role of statutory resource reporting codes but also that the Red Book data is clearly inconsistent with industry expectations of economic U mineral resources. To assess whether this was possibly related to differing reporting methodologies (eg. price ranges used by the Red Book), ore reserves were compiled for Australia, Canada and Namibia. Ore reserves are, based on reporting codes, economic at present and should reflect the current U price environment. For Australia, Canada and Namibia, ore reserves amount to 400, 256 and 263kt U₃O₈, respectively – all considerably lower than either the Red Book or national estimates. Thus the differences cannot be due to the strict

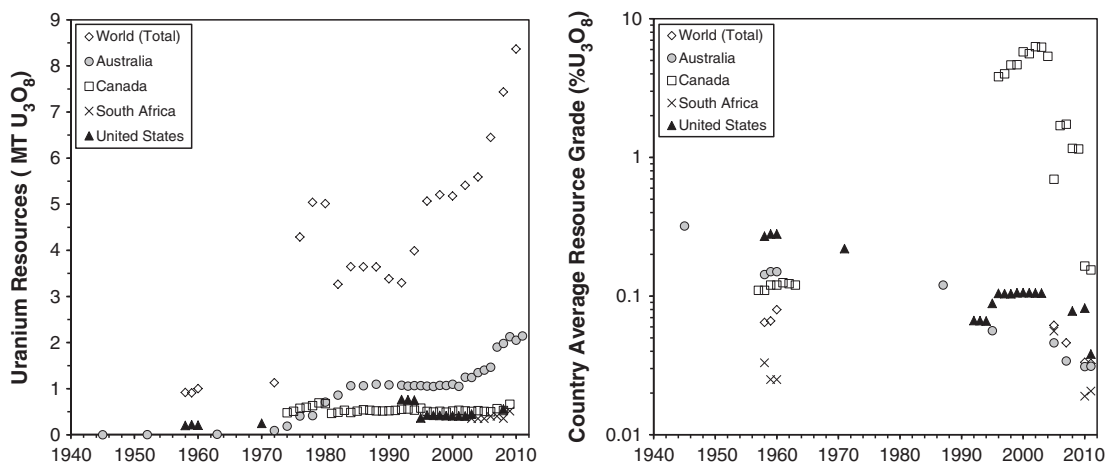


Fig. 1. Reported economic uranium resources by select countries or world over time (from Red Book or national data): contained uranium (left); average country ore grade (right).

use of ore reserves versus mineral resources and it should highlight the value of basing national and global assessments of U resources on methodologies which are statutory, public and include independent auditing — which many countries reporting to the IAEA through the Red Book do not perform (nor does the IAEA allow external auditing of its U deposit database).

The resources data shown in Tables 3–6 and Figs. 2 to 3 demonstrate a classic relationship between grade and tonnage. That is, as ore grades decline a larger deposit is typically found, until such time as the grade reduces faster than the increase in ore tonnage. This distribution closely fits the bimodal approach first proposed by Skinner (1976), suggesting that his concept of a ‘mineralogical barrier’ for mining is also realistic for U: ie. a grade below which the extreme energy costs make mining prohibitively challenging. Based on current technology and economics, the collective U resource data would suggest this is about 0.001% U₃O₈, although it should be remembered that almost all U deposits lower than ~0.01% U₃O₈ are polymetallic with U being a by/co-product. The deposits which stand out as exceptional outliers are either small tonnage but very high grade (eg. McArthur River, Cigar Lake) or the opposite in large tonnage but very low grade (eg. Olympic Dam, Viken).

The rise in the average ore grade in Canada in the 1990s is mainly due to the large, high-grade deposits discovered at McArthur River and Cigar Lake, although recent years has seen a declining trend emerge as lower grade projects such as the Kiggavik–Sissons Schultz, Michelin and others are now being re-considered as potential economic prospects. In reality, however, almost all of these lower grade deposits were discovered in the 1970s–80s, and so if they were included in the resource data set over time, the extent of Canada’s increase in country average ore grade would not be as pronounced. The 1950s–60s U

resources in Canada were dominated by the Elliot Lake district of northern Ontario, which produced ~165 kt U₃O₈ from ore averaging ~0.11% U₃O₈ by field closure in 1996 (data from Mudd and Diesendorf, 2008). Substantial mineralised ore remains at Elliot Lake, given that only some half of the identified ore has been mined (a 1957 estimate of ore resources was 342 Mt while ore production totalled ~157 Mt). Whether the remaining Elliot Lake ore could be classified as economic is speculative at best (Cochrane and Hwozdyk, 2007).

The extent of U resources in Australia is increasingly dominated by the giant Olympic Dam deposit in South Australia. The Olympic Dam copper–U–gold–silver (Cu–U–Au–Ag) project is based on an underground mine, concentrator, U–Cu hydrometallurgical complex, Cu smelter and Cu refinery. The most recent (2012) reported mineral resource is 9576 Mt ore grading 0.82% Cu, 0.026% U₃O₈, 0.31 g/t Au and 1.39 g/t Ag — a contained U of some 2499 kt U₃O₈. In October 2011, Olympic Dam was given approvals to convert to a large open cut project, whereby some U would be produced locally but a significant portion would be contained in U-rich Cu concentrates proposed to be exported to a new, specially constructed Cu smelter in China (BHPB, 2009)—for total production of 19 kt/year of U₃O₈ (though it will take close to a decade for the expansion to reach this scale). This large and complex expansion project, however, was indefinitely postponed in August 2012 by owner BHP Billiton.

In general, based on deposit data in Tables 3 to 6, there remain extensive U resources identified in existing producer countries, but most of these resources are low grade (0.02–0.05% U₃O₈). In the longer term, continuing U demand will have to mean mining these lower grade U ore resources with the resulting large increases in CO₂ emissions (see below). Furthermore, while the total of 11,043 kt U₃O₈

Table 2
Uranium resources by country: Red Book and selected national data (kt U₃O₈; 2011 data).

Country	Red book	National	Country	Red book	Country	Red book
Australia	2050.0	2,143 ^a	Brazil	326.2	Mongolia	65.7
Kazakhstan	966.4		Ukraine	264.8	Tanzania	53.9
Russia	766.7		China	195.8	Jordan	39.9
Canada	724.4	663.8 ^b	Greenland	158.8	Argentina	23.1
Namibia	610.8		India	123.7	Algeria	23.0
United States	556.6	555.9 ^c	Uzbekistan	113.4	Malawi	20.0
Niger	525.2		Botswana	96.9	Rest of the World	222.7
South Africa	438.7	435 ^d			Total	8367

^a As of Dec. 2011 (2012 Edition, GA, var.).

^b As of 2009 (NRC, var.).

^c As of 2008 (EIA, 2010) but excludes possible copper and phosphate by-product U resources.

^d As of 2009 (CMSA, var.).

Table 3

Country totals by ore, grade, contained uranium and number of deposits (2011 data).

Country	Mt ore	%U ₃ O ₈	kt U ₃ O ₈	No.	Country	Mt ore	%U ₃ O ₈	kt U ₃ O ₈	No.
Australia	10,907.2	0.031	3375.9	95	Kazakhstan	1085.7	0.050	541.1	11
Canada	608.4	0.15	938.9	50	Ukraine	211.0	0.132	279.1	4
Sweden	4810.7	0.017	836.5	12	Greenland	956	0.027	261.2	3
South Africa	3930.7	0.021	810.7	36	China	229.7	0.069	159.2	12
USA	1851.2	0.038	707.2	123	Brazil	146.4	0.106	155.2	2
Namibia	2755.7	0.025	678.1	18	Botswana	780.1	0.015	118.6	1
Russia	485.1	0.132	638.9	12	India	260.4	0.041	105.8	9
Niger	633.5	0.089	566.4	10	Rest of the World	4145.1	0.021	869.9	123
					Total	33,792	0.033	11,043	521

significantly exceeds the 2011 Red Book estimate of 8367 kt U₃O₈, major deposits and/or countries are still missing, as well as numerous small deposits <5 kt U₃O₈. According to [Xun \(2006\)](#), China's U resources as of about 1998 consisted of more than 200 U deposits containing at least 70 kt U₃O₈, with hope for between 1 and 2 Mt U₃O₈ in the long-term. The deposit types are 37.05% granite-related, 23.53% sandstone, 18.97% volcanic, 15.94% mudstone-related and 4.51% other ([Xun, 2006](#)). For Brazil, the Itataia region alone is reported to contain some 3.5 Gt of phosphate ore containing 634 kt U₃O₈ (ie. ore grade of ~0.02% U₃O₈; [Saad, 2002](#)) – compared to the current resource of 129 kt U₃O₈ ([INB, 2013](#)) – demonstrating that Brazil has considerably more U resources than the data in this paper (and even the Red Book) would indicate.

In summary, history is indeed repeating itself yet again – with a new rush based on higher prices and perceived hope for nuclear power, global U exploration has delivered outstanding results within the past five years. The major success continues to be expansion at known projects (eg. Ranger and Olympic Dam in Australia, McArthur River in Canada), and major new discoveries across the world. These range from super low grade ore (e.g. Etango-Goanikontes, Namibia, at 0.0193% U₃O₈) to super high grade ore (e.g. Wheeler River-Phoenix, Canada, at 15.74% U₃O₈), with the latter being the highest grade deposit currently known in the world (2011 data). Although there can be a small time lag with reporting through the Red Book, it is clear that the trajectory for known global U resources is still upwards, although the global average ore grade will continue to decline based on the dominance of lower grade ores and deposit types. It is clear that there is ongoing success in finding or proving new U resources over time (including across a wide range of deposit types and regions of the world) – and even based on various scenarios for the future of nuclear power (optimistic or otherwise), there is sufficient U to meet these demands (see later discussion).

Table 4

Top and bottom 10 deposits by highest and lowest ore grade, respectively (2011 data).

Mine/project	Status	Mt ore	%U ₃ O ₈	kt U ₃ O ₈	Deposit Type
Wheeler River-Phoenix	Deposit	0.114	15.74	17.89	Unconformity
Cigar Lake	Development	1.030	15.13	155.72	Unconformity
McArthur River	Operating	1.464	14.98	219.30	Unconformity
Roughrider East/West	Deposit	0.556	4.73	26.30	Unconformity
Millenium	Deposit	0.806	3.81	30.67	Unconformity
Tamarack	Deposit	0.229	3.74	8.59	Unconformity
Paul Bay	Deposit	0.213	3.53	7.52	Unconformity
Midwest-Midwest A	Deposit	0.852	2.85	24.27	Unconformity
Dawn Lake	Deposit	0.347	1.69	5.86	Unconformity
McLean Lake Group	Care & Maint.	1.132	1.22	13.76	Unconformity
Laguna Salada-Guanaco/Buried Lake	Deposit	64.0	0.0063	4.01	Surficial/calcrete
Round Top	Deposit	1033.8	0.0054	56.07	Intrusive
Kloof-Driefontein Tailings	Deposit	371.6	0.0047	17.47	Quartz-Pebble Cong. Tailings
Kloof-Driefontein	Deposit	40.9	0.0042	1.72	Quartz-Pebble Conglomerate
San Juan de la Costa	Deposit	80	0.0040	3.20	Phosphorite
East Rand-Ergo Tailings	Deposit	173.48	0.0039	6.75	Quartz-Pebble Cong. Tailings
Mumbwa	Deposit	87	0.0037	3.22	IOCG Breccia complex
Ammaroo	Deposit	88	0.0023	2.02	Phosphorite
Talvivaara-Kolmisoppi	Operating	660	0.0018	11.88	Shale-hosted
Talvivaara-Kuusilampi	Operating	890	0.0017	15.13	Shale-hosted

4. Uranium mining

4.1. Overview

The uranium mining sector is a relatively minor area of the global mining industry, with a global value of about US\$7.28 billion (i.e. 68,848 t U₃O₈ in 2012 valued at ~US\$105.8/kg U₃O₈, or ~US\$56.3/lb U₃O₈; data from [BREE, 2013](#); [WNA, 2013](#)) compared to coal and iron ore which are of the order of US\$100s of billions globally. The U sector, however, is widely regarded as a key strategic asset and is intimately linked to considerations by certain countries of national security and energy independence – despite the reality of the ongoing menace of nuclear weapons and the broader energy debate (especially in the context of greenhouse gas emissions from fossil fuels and climate change risks).

Overall, there are only a small number of U mines, perhaps of the order of 50 or so globally, shared mostly between Kazakhstan, Canada, Australia, Russia, Namibia, Niger, USA, South Africa and smaller countries. Most of the largest mines are operated by publicly-listed mining companies, such as Cameco, Rio Tinto, Areva, BHP Billiton, Uranium One and ARMZ. In Kazakhstan, every project has a significant shareholding of the government through its state-owned enterprise KazAtomProm, though most U mines are now operated by western companies in joint venture with KazAtomProm.

4.2. Brief review of uranium ore mining and milling

The U industry uses conventional mining and milling methods, and has also been a pioneer in developing new technologies in the mining industry, such as solvent extraction and in situ leaching. Mining is typically through open cut or underground methods, depending on depth, size and other factors (e.g. rivers or lakes). Given it is common for U

Table 5
Top 10 deposits by contained uranium (2011 data).

Mine/Project	Status	Mt ore	%U ₃ O ₈	kt U ₃ O ₈	Deposit Type
Olympic Dam ^a	Operating	9129	0.027	2486.4	IOCG Breccia complex
Viken ^b	Deposit	2854.4	0.017	485.7	Shale-hosted
Elkon	Deposit	217.1	0.173	376.3	Metasomatite
Imouraren	Development	419.21	0.078	328.2	Sandstone
Häggån ^c	Deposit	1791	0.016	286.6	Shale-hosted
Husab-Rössing South	Deposit	529.9	0.042	222.4	Alaskite Intrusive
McArthur River	Operating	1.464	14.98	219.3	Unconformity
Inkai	Operating	350.77	0.056	197.8	Sandstone
Kvaneffjeld ^d	Deposit	619	0.0257	159.1	Vein
Cigar Lake	Development	1.030	15.13	155.7	Unconformity

^a Olympic Dam also contains 0.86% Cu, 0.32 g/t Au and 1.48 g/t Ag plus approximately 0.5% rare earth oxides.

^b Viken also contains 0.268% V₂O₅, 0.028% MoO₃ and 0.032% Ni.

^c Häggån also contains 0.0214% Mo, 0.155% V, 0.0324% Ni and 0.0454% Zn.

^d Kvaneffjeld also contains 1.06% rare earth oxides and 0.219% Zn.

deposits to occur in clusters, as in the Athabasca Basin of Canada or Colorado Plateau of the USA, a central mill can often process ores from a variety of mines. Milling begins with fine grinding, followed by either acid or alkaline leaching, solvent extraction, chemical precipitation and finally calcination to produce tri-uranium octoxide (i.e. U₃O₈). Acid-based leaching is most common, as it is faster and often achieves a more complete extraction, as well as being a relatively cheaper reagent. Alkaline leaching is suited for particular ore types which contain significant calcite (or limestone), the most common of which is surficial (or carnotite/calcrete) type deposits. Further discussion of U ore mining and milling is given by IAEA (1993); IAEA (2009a); OECD-NEA and IAEA (1999) and Özberk and Oliver (2000).

In situ leaching (ISL) is a very specialised form of U production, and is commonly only suitable for sandstone type deposits. The process of ISL involves drilling hundreds of groundwater bores into the sandstone ore, using some as injection bores and most as extraction bores. The reagents are added to the re-circulating solutions, including an acid (eg. sulfuric acid) or alkali (sodium bicarbonate) plus a strong oxidant (eg. oxygen, hydrogen peroxide, or hypochlorate), thereby dissolving the U 'in situ' in the ore formation and bringing it to the surface in the extracted solutions. Although ISL used to be a relatively minor U source, restricted to a handful of mines throughout the world, the rapid growth of ISL mines in Kazakhstan in the past several years has seen ISL now dominate conventional sources — reaching ~45% of global U production in 2012 (WNA, 2013). A major issue with ISL is the challenge of remediating impacted groundwater resources after mining, with the extent of groundwater contamination often severe at Cold War-era acid ISL sites (Mudd, 2001a; Mudd, 2001b) — although civilian-era alkaline

sites have also proved much more difficult to remediate than anticipated (Hall, 2009; Otton and Hall, 2009).

Historical U production by country is given in Fig. 4, showing the major dominance of a select handful of countries such as Canada, the United States, Germany (dominantly the former Eastern Germany), the former Soviet Union and its now component states since 1992 (e.g. Kazakhstan, Uzbekistan etc.), Niger and Australia. By the collapse of the Soviet Union in 1991, it is interesting to note that the USSR, USA, Canada and Eastern Germany had each produced 445.2, 397.5, 292.2 and 256.5 kt U₃O₈, respectively. Cumulative world production by the end of 2012 has been approximately 3101.8 kt U₃O₈ (which compares to the 8367 kt U₃O₈ of resources reported by the 2011 Red Book), given by country in Table 7 with 2012 production.

The nominal U price over time since the start of the civilian era is given in Fig. 5, showing the late 1970s peak of ~US\$43/lb U₃O₈, the low of ~US\$7.10/lb U₃O₈ in late 2000 and the more recent peak of ~US \$136/lb U₃O₈ in June 2007. The price peaks and troughs mirror the rise and fall in exploration and mining.

4.3. Statistics of current uranium mines

A compilation of production statistics for hard rock U projects around the world is given in Table 8, and for ISL mines in Table 9. By comparing the tables, the dominance of a small number of mines (the Top 5 mines, McArthur River, Ranger, Rössing, Priargunsky and Olympic Dam, produced 37.5% of 2011 global mine production) as well as the popularity of ISL for sandstone ores is clear. The more than two orders of magnitude difference in ore grades, from McArthur River at ~15% U₃O₈ (before blending at Key Lake with very low grade material to facilitate milling) to ~0.04% U₃O₈ at Rössing, is extremely unusual in global mining (compared to say maximum and minimum Cu or Au ore grades). This highlights the decidedly variable nature of individual U projects (scale, technical challenges, economics, ore processing, related environmental issues, etc.).

Another important aspect of the data in Tables 8 and 9 is that not all companies report complete production statistics. For example, some mines do not report waste rock (even open cut mines) — despite this often being a major portion of mine waste, and some do not report complete mill data (e.g. ore milled, ore grades, U extraction, or solution volumes and concentrations for ISL).

The available data for average country ore grades over time is compiled in Fig. 6, showing the relative magnitude of different countries such as Namibia versus Australia, as well as the increasing ore grades in Canada as the dominance of Saskatchewan's rich deposits grows (especially the start of the high grade McArthur River project in 2000). Given the increasing proportion of low grade projects under development in Canada, Australia and Namibia, and especially ISL production in Kazakhstan, the overall global average ore grade will continue to decline over time.

Table 6
Deposit type totals by grade, contained uranium and number of deposits (2011 data).

Deposit type	No.	Mt ore	%U ₃ O ₈	kt U ₃ O ₈
Alaskite intrusive	12	1973.8	0.026	513.7
Breccia pipes	2	0.53	0.612	3.2
Coal	3	197.7	0.049	97.8
Intrusive	11	1772.0	0.009	159.3
IOCG Breccia complex	7	9714.8	0.027	2602.6
Metamorphic	4	14.1	0.074	10.5
Metasomatite	24	700.0	0.129	902.0
Phosphorite	12	732.2	0.024	174.2
Quartz–Pebble Cong. tailings	8	2099.8	0.009	179.0
Quartz–Pebble conglomerate	28	1817.4	0.034	622.0
Sandstone	195	3597.3	0.065	2336.1
Shale-hosted	20	7147.4	0.014	1010.9
Surficial/calcrete	61	1466.3	0.028	415.2
Unconformity	38	441.2	0.246	1083.9
Vein	36	1218.7	0.034	417.2
Volcanic	54	753.2	0.058	436.6
Unclassified	6	145.5	0.054	78.4

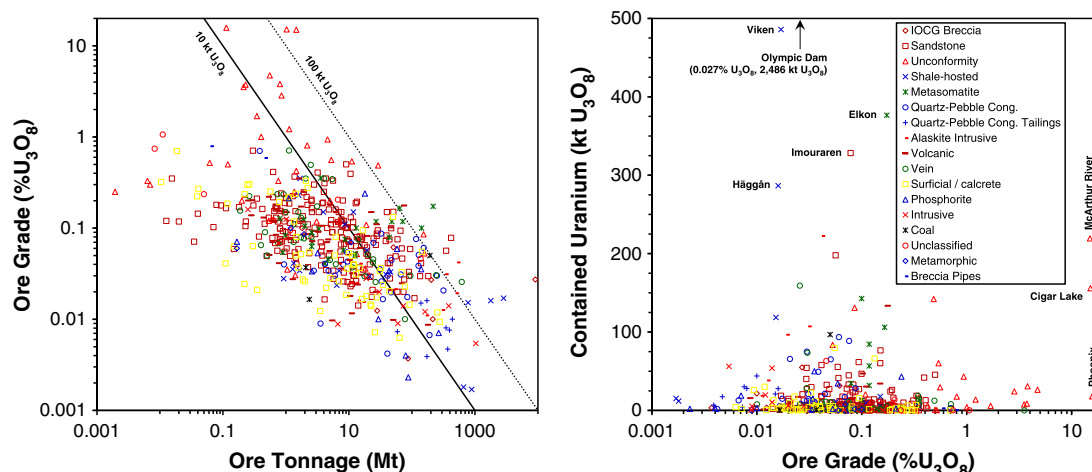


Fig. 2. Uranium grade-tonnage relationships: ore grade versus ore tonnage (left); contained U_3O_8 versus ore grade (right) (2011 data; note most are logarithmic axes).

4.4. Uranium recovery versus ore grades

A key aspect of the future for U mining is the recovery rate — and whether as ore grades decline the recovery rate is also likely to decline. Surprisingly, this aspect has never been analysed before in the literature. Based on the extensive production data sets updated from Mudd and Diesendorf (2008), conventional U mines from Canada and Australia from the 1950s to 2011, including some mines from Malawi, Namibia and Mongolia, are shown in Table 10. The available solution data for selected in-situ leach U mines is shown in Table 11, from mines mostly in Kazakhstan. Recovery versus ore grades are plotted for cumulative production and quarterly/annual data in Fig. 7, with long-term trends in recovery rates over time shown in Fig. 8.

For in-situ leach U mines, the data in Table 11 is U recovery from process solutions and not the extent of leaching and recovery from the ore zones, which is never published. As shown, process solution recovery rates are typically very high, ranging from 93 to 99%, although curiously the process recovery for the Honeymoon project is unusually low at 47.2%. For ore zone recovery, the approach needed is to examine U concentrations before and after leaching, or alternately to compare U reserves (and/or mineral resources) prior to ISL mining with cumulative production, although there are no recent ISL projects which have reached this stage. The Red Book states that mining recovery at ISL U mines in Kazakhstan is typically 80 to 90% (OECD-NEA and IAEA, var.).

For the Beverley ISL U mine in Australia, however, initial resources reported in 1998 were 21 kt U_3O_8 although after a decade of production, totalling some 7.44 kt U_3O_8 by the end of 2010, the Beverley deposits were effectively depleted and the project had sought and received approval to develop other U deposits nearby and from late 2010 most production was derived from these new extensions (WNA, 2013). The significant discrepancy between initial resources and production to date remains poorly explained by Heathgate Resources (a subsidiary of General Atomics), although it does suggest difficulties in estimating extractable U resources by acid ISL methods.

Overall, based on the data presented and aspects analysed, it is difficult to ascertain that recovery is simply a function of average ore grade. Although some common deposit types, such as unconformity deposits, have a typically higher average recovery rate than others (e.g. calcretes), a higher recovery rate could be achieved if it was economic (e.g. longer leach times, stronger oxidants). In reality, the recovery rate will depend on mining method (as this can affect ore dilution rates and thereby ore grades), the processing method and technology used (especially acid or alkaline chemistry), U mineralogy (especially if difficult minerals like brannerite or davidite are present), market prices, input costs (chemicals, diesel, electricity, labour, finance, etc.), accidents causing site interruptions, as well as environmental regulation and constraints. All of these factors can vary significantly over time and lead to different outcomes with respect to ore grades and recovery. Finally, as a project

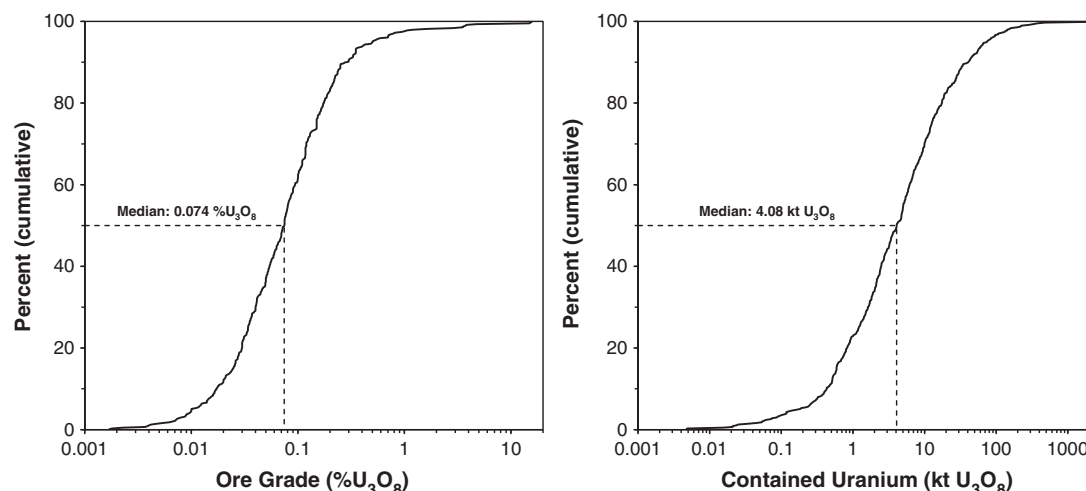


Fig. 3. Cumulative frequency curves for ore grade (left) and contained U_3O_8 (right) (2011 data; note some logarithmic axes).

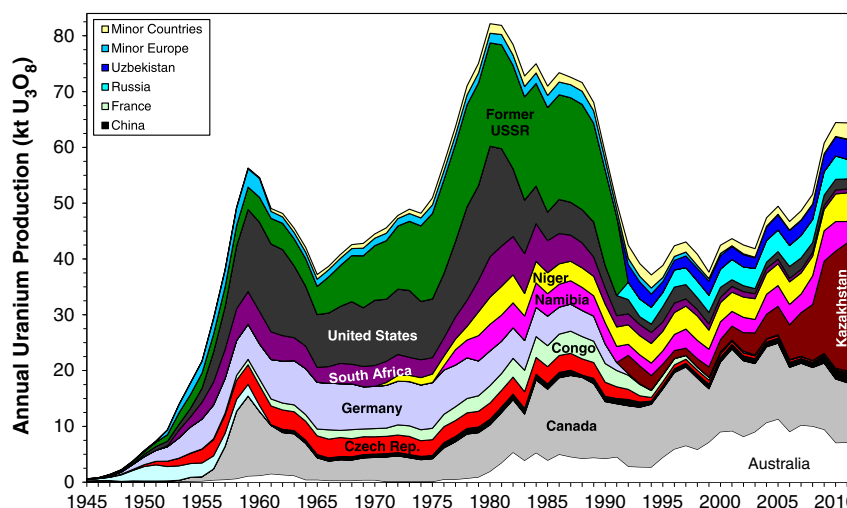


Fig. 4. Historical world U production by country (data compiled from OECD-NEA and IAEA, 2006; OECD-NEA and IAEA, var.; WNA, 2013).

develops more experience with a given deposit and its characteristics, recovery can often increase over time, as shown for many projects in Fig. 8.

Any realistic model of recovery versus ore grade would therefore need to not only include deposit type and U mineralogy but also processing configuration and economic considerations.

4.5. Uranium mining case studies

To further examine the trends in reserves/resources and cumulative production, a selected range of projects are compiled in Table 12, namely Olympic Dam and Ranger in Australia, Key Lake, Rabbit Lake, McArthur River and the Elliot Lake field in Canada and Rössing in Namibia. For Key Lake and Ranger, cumulative production has already exceeded initial reserves/resources, while for most others the sum of cumulative production and remaining resources substantially exceeds the initial reserves/resources. The key factors which have underpinned this common increase in total U resources, at least at the projects in Table 12, are successful exploration (ie. new deposits or extensions), lowering of cutoff grades, buoyant market conditions (allowing re-investment) or sometimes changes in project configuration (eg. new mining or processing methods). While there are some projects whereby U recovered was less than initial reserves, this is less common now due to the conservative nature of mineral resource codes – although ISL mines may be an exception (see comment on Beverley above).

Overall, it is clear that modern mining practices for reserve-resource reporting as well as the inherent conservatism in feasibility studies are

commonly leading to outcomes whereby total U resources for a project can readily exceed initial estimates, sometimes substantially.

5. Key environmental aspects of uranium mining

5.1. Energy, water and greenhouse gas emissions intensity

A welcome trend across all sectors of global industry is the strong emergence of sustainability reporting over the past decade. Numerous mining companies have certainly been at the forefront of this change in corporate accountability by publishing annual sustainability reports alongside statutory financial reports (Mudd, 2009b). The most popular protocol is the Global Reporting Initiative (GRI, 2006) – a coalition of the United Nations, industry, government and civil society groups. The use of the GRI for reporting is (still) voluntary, and it includes core and voluntary indicators covering economic, social, environmental, human rights and labour aspects of an organisation's activities, with some being qualitative while others are quantitative. A specific sector supplement was recently finalised to facilitate improved and more relevant sustainability reporting for mining (GRI, 2010). Some mining companies continue to rely on internally developed systems for sustainability reporting, with variable comparison to the GRI. The extent to which a report meets GRI requirements can also be assessed, giving a company's report an 'application level', essentially a measure of thoroughness or quality assurance. The issue of external auditing is emerging as a key test regarding the credibility of reports (Fonseca, 2010) – that is, the old 'spin versus substance' debate.

Table 7
Cumulative (1945 to 2012) and 2012 uranium production by country (t U₃O₈).

Country	Cumulative	2012	Country	Cumulative	2012
Canada	547,297	10,610	France	89,617	4
USSR + Russia ^b	518,100 ^b	3386	Uzbekistan ^a	54,014 ^a	2830
United States	436,230	1882	China	43,563	1769
Germany ^c	258,974 ^c	59	Dem. Rep. Congo	30,182	0
Australia	216,054	8242	Gabon	29,950	0
South Africa	192,024	548	Hungary	24,872	0
Niger	145,939	5502	Romania	22,102	106
Namibia	132,205	5300	Ukraine ^a	20,965 ^a	1132
Czechoslovakia	131,336	269	Rest of the World	60,135	2077
Kazakhstan ^a	148,225 ^a	25,133	World	3,101,783	68,848

^a Production from 1992 to 2012 only (production previously part of the USSR).

^b Much of USSR production (445,206 t U₃O₈ up to the end of 1991) was from Kazakhstan, Uzbekistan and related countries.

^c Production mostly from the former German Democratic Republic (GDR or Eastern Germany; 251,610 t U₃O₈).

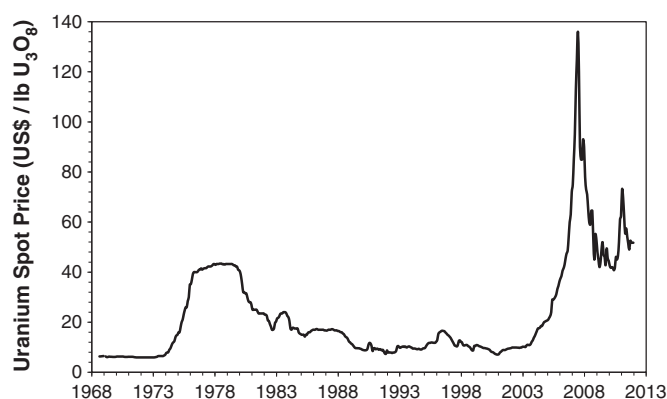


Fig. 5. U spot price over time (data from ABARE, var.; BREE, 2012b; BREE, 2013).

Table 8

Available production statistics for conventional uranium mines (2012 data).

Mine/project	Deposit type	Mine type	Process	Mt ore	%U ₃ O ₈	t U ₃ O ₈
McArthur River ^a	Unconformity	Underground	Acid-SX	0.194	4.61	8825
Olympic Dam ^b	IOCG Breccia	Underground	Acid-SX	9.576	0.054	4,032 ^b
Ranger	Unconformity	Open cut	Acid-SX	2.577 ^c	0.17	3710
Arlit-Somaïr	Sandstone	Open cut	Acid-SX/HL	~2.30 ^d	~0.168 ^d	3613
Rössing	Alaskite Intrusive	Open cut	Acid-IX/SX	12.127	~0.026 ^e	2699
Kraznokamensk	Volcanic	Underground	Acid-IX	~1.38 ^d	~0.18 ^d	2359
Langer Heinrich	Surficial calcrete	Open cut	Alkaline-IX	3.331	0.0809	2306
Rabbit Lake	Unconformity	Underground	Acid-SX	0.260	0.71	1782
Akouta-Cominak	Sandstone	Underground	Acid-SX	~0.47 ^d	~0.405 ^d	1772
Kayelekera	Sandstone	Open cut	Acid-RIP-SX	1.388	0.1182	1298
Vaal River Group ^f	Quartz-Pebble Cong.	Underground	Acid-IX/SX	4.318	0.013 ^g	549
White Mesa ^h	Surficial calcrete	Various	Acid-SX	~0.10 ^d	0.38 ^g	379
Trekkopje	Surficial calcrete	Open cut	Alkaline-HL	~1.68 ^d	~0.018 ^d	296
			Total	~39.7	~0.085 ^d	33,620

Notes: Acid – sulfuric acid leaching; SX – solvent extraction; IX – ion exchange; RIP – resin-in-pulp; HL – heap leach.

^a McArthur River ore is processed at Key Lake and diluted with waste rock to facilitate milling (hence a much lower ore grade than its resources).

^b Olympic Dam also produces Cu, Au and Ag, although the uranium is reported as uranium oxide concentrate which is about 97–98% U₃O₈.

^c Mineral resources grade only.

^d Approximate data only from OECD-NEA and IAEA (var.) or respective company sources. All production data from company annual or quarterly reporting, with some mine and process information from OECD-NEA and IAEA (var.).

^e At Rössing, milled ore grade is not reported, value assumes 85% recovery (Johnson, 1990), consistent with reserve-resource grade reporting over time.

^f Gold also produced.

^g Yield only, not true assayed ore grade.

^h Vanadium also produced, data for conventional ore only and not alternate feed materials.

Given the growing extent of reporting, it is possible to use the data to analyse U mining and derive ‘sustainability metrics’ such as energy, water and greenhouse gas emission (GGE) intensity. Although some U mines and companies do publish sustainability reports and data, many (still) do not – often despite their rhetoric about the safety credentials of U mining and nuclear power or its low carbon intensity. This paper summarises the available data only; for a more detailed discussion on this area see Mudd (2009b); Mudd and Diesendorf (2008) and Mudd and Diesendorf (2010). The energy, water and GGEs intensity of some U mines is given in Table 13, and the relationships between energy and GGEs intensity with respect to ore grade are shown in Fig. 9. The relationship between GGEs and energy intensity of U production is shown in Fig. 10.

The first key observation from Table 13 and Fig. 9 is the inverse ore grade-energy/GGE intensity relationship. By comparing the Ranger and Rössing mines, they have average ore grades of ~0.3% and ~0.04% U₃O₈, respectively, with Ranger having both a lower energy and GGE intensity for U production. The high grade Saskatchewan U mines,

however, do not necessarily have a lower intensity – such as Rabbit Lake with a similar GGEs intensity as Rössing while McLean Lake has a similar energy intensity. For McArthur River, the energy cost of mining and processing each tonne of ore is clearly considerable (~6000 MJ/t ore), much higher than most other mines (~120–600 MJ/t ore), showing that the energy-intensive ground freezing and remote mining methods used more than offset the higher grade since energy intensity is only marginally lower than Ranger (ie. 153 vs 216 GJ/t U₃O₈). Finally, the Beverley ISL mine has a similar energy intensity to conventional mines, showing that although there is no ore excavation and processing, the energy costs are offset by the lower grade ores and production, intensive solution pumping and lower overall extraction efficiencies.

If the energy-GGEs versus ore grades data sets are assessed as a whole and power regressions developed, the following equations are arrived at (where ore grade is in %U₃O₈):

$$\text{Energy intensity (GJ/t U}_3\text{O}_8) = 219.49 (\text{ore grade})^{-0.141} \quad (R^2 = 0.169) \quad (1)$$

$$\text{Carbon intensity (t CO}_2\text{/t U}_3\text{O}_8) = 15.493 (\text{ore grade})^{-0.357} \quad (R^2 = 0.555) \quad (2)$$

The GGEs-energy intensity of U production (Fig. 10) effectively shows the carbon intensity (ie. t CO₂/GJ) of total energy use for a particular mine, as well as the relative differences between mines. In comparing the various mines, McLean Lake has the lowest carbon intensity presumably due to the high fraction of gas-hydro-wind in the provincial grid,² compared to Olympic Dam with the highest gradient which is based on a dominantly gas/wind/coal-based grid³ and large underground mine whereby more electricity is used in mining than in open cut methods (e.g. Ranger).

² Electrical generation capacity in Saskatchewan is 1686 MW coal (39.8%), 1529 MW gas (36.1%), 823 MW hydro (19.4%), and 198.6 MW wind (4.7%) (www.saskpower.com; accessed 20 August 2013).

³ For Olympic Dam, the South Australia (SA) grid has traditionally been dominated by coal and gas, with recent strong growth in gas and renewables now diluting coal's role in particular. SA's electrical grid is now 1151 MW wind (23.6%), 2676 MW gas (54.9%), 780 MW coal (16.0%), 130 MW solar photovoltaics (2.7%), 113 MW oil (2.3%), 20 MW biomass (0.4%), 4 MW hydro (0.1%) (BREE, 2012a).

Table 9

Available production statistics for in-situ leach uranium mines (2012 data).

Mine/project	Process	ML solutions	mg/L U ₃ O ₈	t U ₃ O ₈
Katco (Tortkuduk/Myunkum)	Acid-IX	No data	No data	4316
Karatau	Acid-IX	12,138	210	2518
South Inkai	Acid-IX	34,837	65	2205
Inkai	Acid-IX	No data	No data	1996
Akbastau	Acid-IX	6040	238	1418
Akdala	Acid-IX	17,853	74	1291
Zarechnoye	Acid-IX	23,989	49	1111
Kharasan	Acid-IX	8784	80	687
Smith Ranch-Highland	Alkaline-IX	No data	No data	499
Dalur	Acid-IX	No data	No data	624
Beverley	Acid-IX	No data	No data	387
Crow Butte	Alkaline-IX	No data	No data	363
Khiagda	Acid-IX	No data	No data	391
Willow Creek (Irigary)	Alkaline-IX	7750	40	282
Honeymoon	Acid-SX	5541	61	154
Palangana	Alkaline-IX	No data	No data	70
	Total	–	–	18,312

Notes: ML – mega-litre; Acid – sulfuric acid leaching; SX – solvent extraction; IX – ion exchange. All ISL mines are based on sandstone deposit types. All production data from company annual or quarterly reporting, with some mine and process information from OECD-NEA and IAEA (var.).

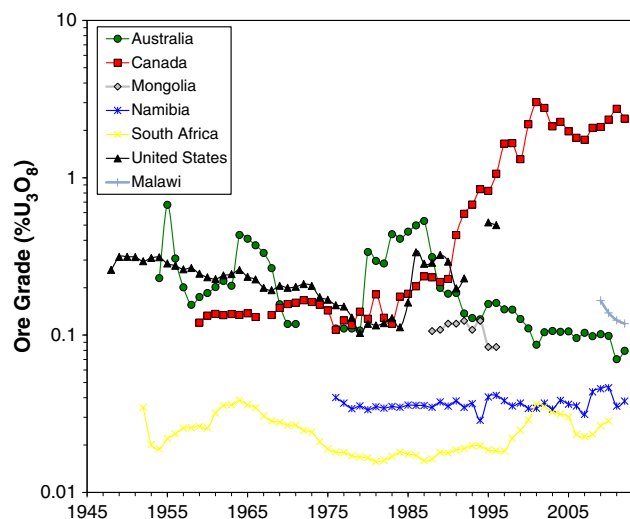


Fig. 6. Average uranium grades in ore milled by country over time.

The second key observation from Table 13 and Fig. 9 is the gradually increasing energy/GGEs intensity over time. At the Ranger mine, ore grades have declined from 0.28% to 0.23% U₃O₈ from 2005 to 2010, respectively — leading to a strong increase in energy/GGEs intensity over this time. Ore grade declines are also apparent for the McLean Lake and Olympic Dam mines. For the Rössing mine, the increasing intensity is mainly related to an expansion of open pit mining scale, whereby waste rock mined has increased from 7.5 to 42 Mt/year over 2005 to 2010 while ore milled has been stable at ~12 Mt/year.

Taken together, these two key findings are crucial in understanding the future of U mining. As shown with U resources, although some high grade mines will continue for some time, the future of U mining will be increasingly dominated by lower grade ores. In conjunction with the energy/GGEs intensity issues, this means that there will be significant upward pressure on energy/GGEs intensity as ore grades decline. The growing intensity will be very gradual, and each mine will have its own particular circumstances to address, such as old infrastructure, larger scales (deeper pits, longer haul routes for waste rock), declining ore grades, possible constraints such as carbon pricing or water resources, etc.

Another major aspect which is often over-looked in understanding energy/GGEs intensity is U mineralogy. Some U minerals, such as uraninite, carnotite or coffinite, are readily soluble in acidic or alkaline solutions (especially with an added oxidant such as pyrolusite, oxygen or hydrogen peroxide), while others such as brannerite and davidite have very low solubility without extremely aggressive conditions (ie. extremely strong acids, high temperatures and/or high pressures). Some of the world's major U deposits have a significant proportion of brannerite and hence are not readily processed using conventional milling methods. For example, Olympic Dam ore has 54% coffinite, 34% brannerite and 12% uraninite (Hayward and O'Connell, 2007) — with a historical extraction efficiency of 67.1% (data updated from Mudd, 2009a) showing that most uranium is dissolved from the coffinite and uraninite with only a very small fraction derived from the brannerite. The Elkon deposit in Russia was first discovered in 1960 and remains one of the largest U deposits in the world (third after Olympic Dam and Viken) — but has yet to be developed since the U mineralogy is dominantly brannerite (Boytssov, 2009). If the mineralogy is refractory this means low recoveries and high unit production costs — often making a project uneconomic. It is possible to treat refractory ores, using a

Table 10
Cumulative production data and recovery rates for selected uranium mines (Australia, Canada, others).

Country	Mine	Deposit Type	Processing	Data Period ^a	Ore milled (Mt)	%U ₃ O ₈	t U ₃ O ₈	Recovery (%)	Waste Rock(Mt) ^b
Australia	Rum Jungle	Unconformity	Acid-IX/SX	1954–71	0.682	0.33	1841	81.13	~16.3
Australia	Radium Hill	Intrusive	HMS-Acid-IX	1956–61	0.969	0.117	852.1	75.15	No data
Australia	Rockhole	Unconformity	Acid-SX	1959–62	0.013	1.107	139.7	94.07	No data
Australia	Upper South Alligator Valley Group	Unconformity	Acid-SX	1959–64	0.095	0.483	392.7	85.55	No data
Australia	Mary Kathleen (1958–63)	Metamorphic	Acid-IX	1958–63	2.668	0.172	4092	89.22	3.863
Australia	Mary Kathleen (1976–82)	Metamorphic	Acid-SX	1976–82	~6.3	0.10	4801	76.20	16.68
Australia	Nabarlek	Unconformity	Acid-SX	1980–88	0.669	1.65	10,875	98.31	2.33
Australia	Ranger	Unconformity	Acid-SX	1981–2012	45.672	0.288	116,005	88.86	~200
Australia	Olympic Dam	Breccia Complex	Acid-SX	1988–2012	148.379	0.063	66,850	71.47	~13.7
Canada	Bicroft-Macassa	Intrusive	Acid-IX	1956–63	2.318	0.094	2025	93.12	No data
Canada	Pronto	Conglomerate	Acid-IX	1956–58	1.327	0.121	1374	85.46	No data
Canada	Faraday	Intrusive	Acid-IX	1957–64	2.683	0.104	2629	94.36	No data
Canada	Elliot Lake – Denison	Conglomerate	Acid-IX	1957–92	62.771	0.112	65,809	93.71	No data
Canada	Elliot Lake – Quirke	Conglomerate	Acid-IX	1957–90	32.337	0.116	35,333	94.58	No data
Canada	Elliot Lake – Nordic	Conglomerate	Acid-IX	1957–90	34.971	0.112	37,355	95.28	No data
Canada	Elliot Lake – Lacnor	Conglomerate	Acid-IX	1958	2.239	0.096	1977	91.87	No data
Canada	Elliot Lake – Dyno	Intrusive	Acid-IX	1958–60	0.807	0.062	479	95.63	No data
Canada	Elliot Lake – Stanrock	Conglomerate	Acid-IX	1958–63	2.432	0.094	2122	92.74	No data
Canada	Elliot Lake – CanMet	Conglomerate	Acid-IX	1958	0.664	0.096	584	91.64	No data
Canada	Port Radium	Vein	Acid-IX	1958, 60	0.156	0.59	889	96.14	No data
Canada	Gunnar	Vein	Acid-IX	1960–61	1.256	0.185	2207	95.20	No data
Canada	Elliot Lake – Stanleigh	Conglomerate	Acid-IX	1960, 1983–96	12.509	0.090	10,326	91.28	No data
Canada	Beaverlodge	Metasomatic	Alkaline-SX	1962–63, 1977–82	2.526	0.202	4743	93.13	No data
Canada	Elliot Lake – Milliken	Conglomerate	Acid-IX	1964	0.481	0.112	531	94.50	No data
Canada	Bancroft	Intrusive	Acid-IX	1978–82	1.646	0.081	1269	94.84	No data
Canada	Rabbit Lake	Unconformity	Acid-SX	1979–2012	12.329	0.73	83,920	93.84	No data
Canada	Cluff Lake	Unconformity	Acid-SX	1982–2002	3.228	0.85	26,849	97.36	No data
Canada	Key Lake-McArthur River	Unconformity	Acid-SX	1983–2012	6.780	3.03	199,596	97.11	No data
Canada	McLean Lake	Unconformity	Acid-SX	1999–2010	1.526	1.49	22,095	99.77	No data
Malawi	Kayelekera	Sandstone	Acid-RIP	2009–12	3.379	0.127	3208	74.94	7734
Mongolia	Dornod	Sandstone	not known	1988–96	0.946	0.107	920	90.79	19,325
Namibia	Langer Heinrich	Calcrete	Alkaline-IX	2007–12	11.728	0.090	8401	79.75	27,034
Total					406.5 Mt	0.195	720,490	90.86	

Notes: Acid — acid leaching; IX — ion exchange; SX — solvent extraction; HMS — heavy media separation; RIP — resin-in-pulp extraction.

^a Data period is only the years for which data is available, and not necessarily the total period of operation.

^b Waste rock includes low grade ore.

Table 11

Available cumulative production data and recovery rates from solutions for selected for in-situ leach (ISL) uranium mines (Kazakhstan, Australia, USA).

Country	Mine	Processing	Data period ^a	Solutions (ML)	mg U ₃ O ₈ /L	t U ₃ O ₈	Recovery (%)
Kazakhstan	Akdala	Acid-IX	2007–2012	85,083	89	7446	98.80
Kazakhstan	South Inkai	Acid-IX	2007–2012	94,754	81	7555	98.49
Kazakhstan	Karatau	Acid-IX	2010–2012	35,090	204	7098	99.18
Kazakhstan	Akbastau	Acid-IX	2011–2012	10,893	253	2722	98.82
Kazakhstan	Zarechnoye	Acid-IX	2011–2012	45,197	46	1976	94.67
Kazakhstan	Kharasan	Acid-IX	2008–2012	20,853	79	1631	98.92
Australia	Honeymoon	Acid-SX	2011–2012	6479	65	200	47.22
USA	Willow Creek	Alkaline-IX	2011–2012	10,696	38	379	93.25

Notes: Acid – acid leaching; IX – ion exchange; SX – solvent extraction. All data from corporate reporting of Uranium1.

^a Data period is only the years for which data is available, and **not** necessarily the total period of operation.

combination of high strength acids or alkalis with high temperature and/or pressure (eg. the former Elliot Lake district, Canada, which was primarily U in brannerite, or the former Radium Hill mine, Australia, where the U was in davidite), but this invariably increases chemical and energy inputs and places even greater pressure on the energy/GGEs intensity of U production.

5.2. Mine rehabilitation and long-term stewardship

A major problem with the early Cold War era U mines was poor mine waste management and invariably a lack of mine site rehabilitation. Along with the rise of environmental regulation from the 1970s in most countries, U mining has also been required to undertake mine rehabilitation after operations cease. The community typically expect that this will lead to a stable site, with no ongoing pollution or radioactivity issues. The experience internationally, however, is quite varied, and this sub-section will only briefly touch on this area. Further details, including numerous papers on various U mines, are found in IAEA (2005); IAEA (2011); Merkel and Hasche-Berger (2002); Merkel and Hasche-Berger (2005); Merkel and Hasche-Berger (2008) and Merkel and Schiepek (2011), amongst a plethora of others.

As noted above, countries such as Germany and the United States have spent billions of dollars on remediation of U mines from the Cold War era.

In the old Germany Democratic Republic (GDR, or Eastern Germany), the Soviets operated large scale, low grade mines in the

Saxony–Thuringia region. At the time of reunification in 1991, West Germany had to accept the cost of all remediation of these mines – a total of €6.6 billion (or ~13 billion Deutschmarks; Hagen and Jakubick, 2005). The Wismut legacy included 5 large underground mines, one large open cut mine (84 Mm³), waste rock dumps (311 Mm³) and tailings (160 Mm³). The major problems included excessive radiation exposure risks (especially due to radon build-up), leaching of heavy metals and radionuclides to surface water ecosystems due to growing acid mine drainage, groundwater contamination, physical safety risks, and aesthetic issues, amongst others. All of these issues were made more pressing due to the high population density of the Saxony–Thuringia region. The programme was planned to last for more than 20 years, and by 2011 most of the work was completed with efforts moving to ongoing maintenance and monitoring. Many of the challenges had never before been attempted – such as remediating extremely soft tailings with a density half that of water, backfilling a large open cut, groundwater remediation from underground (or in-place) leaching, to large scale re-contouring of waste rock dumps. Although early indications are that the programme has been relatively successful, there remains a need for caution as ongoing monitoring indicates the true status of the former sites.

In the USA, many of the Cold War era U mines supplied the military programme of the day, and the US Government has therefore taken on the liability of rehabilitation of these old sites. The large effort was known as the ‘Uranium Mill Tailings Remediation Action’ (or UMTRA) programme, and included numerous sites in mid-west states (Colorado,

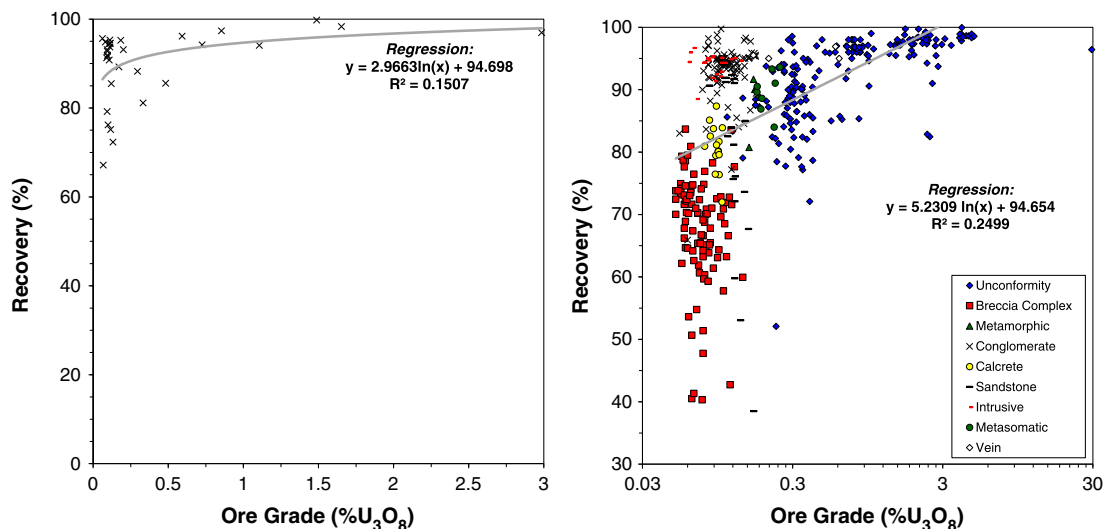


Fig. 7. Uranium recovery versus ore grades for uranium mines – cumulative production data only (left), all quarterly and annual milling data by deposit type (right; note logarithmic x-axis).

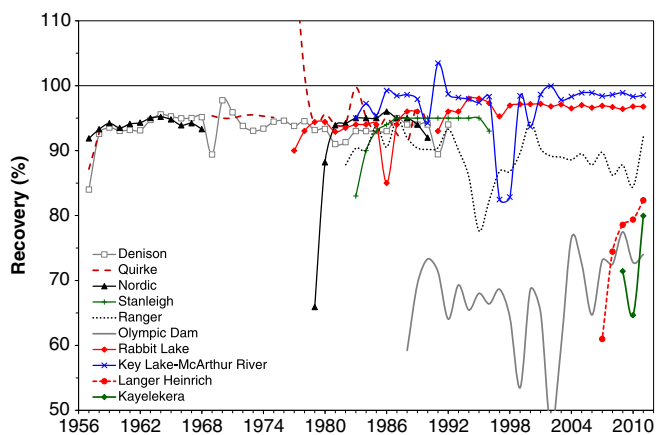


Fig. 8. Uranium recovery over time for various uranium mines.

Utah, New Mexico, Arizona) and others. At some sites, tailings dams and waste rock dumps were covered and revegetated, while at some the tailings were excavated and emplaced at a new, more suitably engineered site. The programme has cost billions of dollars, although a thorough review of the entire programme does not appear to have been published (yet).

In Australia, based on the authors' experience, there is often a widely held belief of successful rehabilitation of legacy U projects – but most of the time this view is held by those who have never visited the sites. In brief, the major Cold War-era U mines in Australia were the Mary Kathleen, Rum Jungle, Radium Hill-Port Pirie and the Upper South Alligator Valley, with the latter rehabilitated only in the 2000s (after the Coronation Hill saga) while all others were rehabilitated in the mid-1980s. Further small U projects were also developed at Pandanus Creek-Cobar 2, Fleur de Lys, George Creek, Brock's Creek and Adelaide River in the Northern Territory and Myponga in South Australia, though no substantive rehabilitation work is known for each site. The Nabarlek project, which operated from 1979 to 1988, was a 'modern U mine' and approved and operated under strict regulations and supervision, being rehabilitated in the mid-1990s. Other 'modern U mines' are still in operation at Ranger, Olympic Dam and more recently Beverley. At present, surprisingly, there is no former U project in Australia which can be claimed as a successful, long-term rehabilitation case study – all still require ongoing monitoring and maintenance, and some remain mildly to extremely polluting (such as acid mine drainage affecting surface waters and/or groundwater; airborne dispersion of tailings; erosion of

covers exposing highly radioactive materials; radionuclide and heavy metal uptake into certain plants; further details in Bollhöfer et al. (2006); Klessa (2000); Lottermoser (2011); Lottermoser and Ashley (2005); Lottermoser and Ashley (2006); Lottermoser et al. (2005); Mudd and Diesendorf (2010) and Mudd and Patterson (2010). Some photographs of Rum Jungle and Mary Kathleen are given in Fig. 11.

In South Africa, the Witwatersrand Basin gold deposits often contained low concentrations of U, which led to South Africa being an important U producer in the early Cold War years (see Fig. 4). By 2011, about 6.4 Gt of gold ore has been processed, of which about 775 Mt has been processed for U (data updated from Mudd, 2007). The extensive underground network of underground mines and tailings facilities have left a major legacy, contributing to problems such as acid mine drainage, erosion, public health risks and constrained land use challenges (eg. Barthel, 2011; Coetzee et al., 2002; Hamman, 2012; Winde, 2011a, 2011b; Winde and Sandham, 2004). Given that most of these sites closed many years ago and no operating company remains responsible, the costs have been left to a government which is struggling to find the budget to address the substantial scale of the various challenges. Some photographs are shown in Fig. 12.

For Canada, remediation of old U mines has focussed on the Elliot Lake and Bancroft fields in Ontario, the Uranium City group mines in northern Saskatchewan and Port Radium and Rayrock in the Northwest Territories (Stenson, 2005). Recent results from ongoing environmental monitoring of the rehabilitated tailings areas at Elliot Lake generally show reasonable effectiveness with water quality continuing to improve over time, although there are some issues whereby water quality is deteriorating at some monitoring locations (see ME, 2011). As with other countries, the test of time will be crucial in understanding the effectiveness of U mine rehabilitation, and the lack of data and published studies for most Canadian former U mines limits comparisons to international efforts.

In summary, successful rehabilitation of former U mines still remains challenging, with issues varying widely and dependent on a range of site-specific factors (climate, site history, scale, geology, rehabilitation approach, etc). In many parts of the world, and especially in developing countries (e.g. Niger, Gabon, Argentina, Brazil), community concern is often related to the negative legacy of past U projects – and this can lead to significant opposition. There is no room for complacency or over-confidence in U mine rehabilitation, with the need for high standards and sustainable rehabilitation outcomes clear throughout the world. Finally, there is a critical need for a thorough and comprehensive review of the success (or otherwise) of global U mine rehabilitation efforts and programmes; such a review could help synthesise best practices and highlight common problems and possible solutions.

Table 12

Case study U mines – initial reserves/resources, cumulative production and remaining resources.

Mine	Disc.	Initial reserves/resources				Cumulative production				Remaining resources (2012)		
		Year	Mt ore	%U ₃ O ₈	kt U ₃ O ₈	Years	Mt ore	%U ₃ O ₈	kt U ₃ O ₈	Mt ore	%U ₃ O ₈	kt U ₃ O ₈
Ranger	1969	1980	44.651	0.247	110.4	1981–2012 ^a	45.672	0.288	116.0	86.97	0.08	71.3
Olympic Dam ^b	1975	1982	2000	0.06	1200	1988–2012 ^a	148.38	0.063	66.8	9576	0.026	2499
McArthur River ^c	1988	1995	1.316	14.35	188.9	2000–2012 ^a	0.780	13.5	105.0	1.477	13.74	203.0
Key Lake ^d	1975	1977	1.85	2.45	45.2	1983–2012 ^{a,d}	~6.02	~1.68	~97.0	0.062	0.52	0.3
Rabbit Lake	1969	~1975	~4.5	~0.45	20.4	1975–2012 ^a	12.84	0.71	85.8	2.329	0.77	17.9
McLean Lake-Midwest	1979	~1996	0.696	2.78	22.8	1999–2010	1.526	1.49	22.1	1.826	1.99	36.3
Elliot Lake Field ^e	1953	~1957	~321	0.11	354	1957–1996	~157	0.11	162.7	201.4	0.057	114.6
Rössing	1928	~1970	~350	~0.035	~125	1976–2012 ^a	~409	~0.035	~123	467	0.028	129.1

Notes: Disc. – discovered (first deposit in the area where there are more than one).

^a Still in operation.

^b Olympic Dam also produces Cu–Au–Ag.

^c McArthur River is as mined, since this ore is then transported to Key Lake for milling.

^d Key Lake is being used to process McArthur River ore and low grade material from Key Lake is used for dilution, this has been added where possible.

^e Elliot Lake includes all major mines in the field (Denison, Quirke, Panel, etc.).

Table 13

Energy, water and greenhouse gas emissions intensity of some uranium mines (further details in and data updated from Mudd and Diesendorf, 2008), including Langer Heinrich and Kayelekera data from Paladin (2012).

Project	Ore grade	Water		Electricity		Total energy		CO ₂ emissions	
	%U ₃ O ₈	m ³ /t ore	m ³ /t U ₃ O ₈	kWh/t ore		MJ/t ore	GJ/t U ₃ O ₈	kg CO ₂ /t ore	t CO ₂ /t U ₃ O ₈
Olympic Dam ^{17.4%, a}	0.053–0.114	0.264 ± 0.072 (24)	526.8 ± 77.9 (24)	16.2 ± 1.3 (6)		118 ± 24 (21)	250 ± 53 (21)	20.7 ± 3.6 (19)	43.5 ± 10.2 (19)
Ranger	0.231–0.423	0.153 ± 0.077 (11)	59.8 ± 29.0 (11)	nd		571 ± 117 (19)	216 ± 54 (19)	42.4 ± 10.8 (19)	16.1 ± 4.9 (19)
Rössing	~0.03–0.04	0.267 ± 0.027 (11)	917.5 ± 76.4 (18)	19.5 ± 0.8 (5)		125 ± 26 (18)	436 ± 149 (18)	15.4 ± 2.2 (18)	53.4 ± 14.6 (18)
Beverley (ISL)	~0.18	–	8520 ± 1500 (7)	–		–	216 ± 63 (5)	–	11.2 ± 3.3 (5)
McArthur River	3.91	49.2 (2)	1250 (2)	1083 (2)		6027 (2)	153 (2)	782 (2)	19.9 (2)
Rabbit Lake	0.78	14.3 (2)	2027 (2)	438 (2)		2513 (2)	355 (2)	337 (2)	47.5 (2)
McLean Lake	0.53–2.29	4.85 ± 0.87 (7)	476.5 ± 312.2 (7)	249 ± 17 (7)		4078 ± 645 (7)	415 ± 293 (7)	198 ± 43 (7)	20.7 ± 16 (7)
Cluff Lake	2.71	9.78 (1)	365 (1)	nd		5187 (1)	194 (1)	325 (1)	12.1 (1)
Langer Heinrich	~0.090	0.729 (2)	987.8 (2)	38.0 (2)		449 (2)	608 (2)	59.3 (2)	80.6 (2)
Kayelekera	~0.12	1.548 (2)	1626 (2)	nd		620 (2)	653 (2)	43.7 (2)	46.0 (2)
Production weighted average		4.72 (69)	805 (76)	319 (24)		996 (75)	287 (80)	100.2 (73)	32.8 (78)

Notes: nd – no data.

^a Olympic Dam is presented on the basis of attributing 17.4% of inputs and outputs to U production, since this is the long-term average proportional revenue from U (Cu is 76.1%, Au–Ag 6.5%; data updated from Mudd (2009a).

6. Future uranium demand and realistic constraints

The data compiled and synthesised in this paper are extensive and raise numerous issues with respect to the future of U mining, with the primary issues remaining: (i) potentially mineable U resources; (ii) the future of nuclear power; (iii) energy and greenhouse gas emissions accounting; and (iv) environmental impacts from U mining and rehabilitation.

6.1. Economic uranium resources and the future of nuclear power

The future of nuclear power clearly remains contested and contentious – and therefore difficult to forecast accurately. While some optimists remain eternally hopeful, reality appears to be relegating nuclear power to the uneconomic category of history. If we assume, however, that the optimists might be correct, such as the IEA scenarios, these are shown in Fig. 13. There are three primary IEA scenarios – current

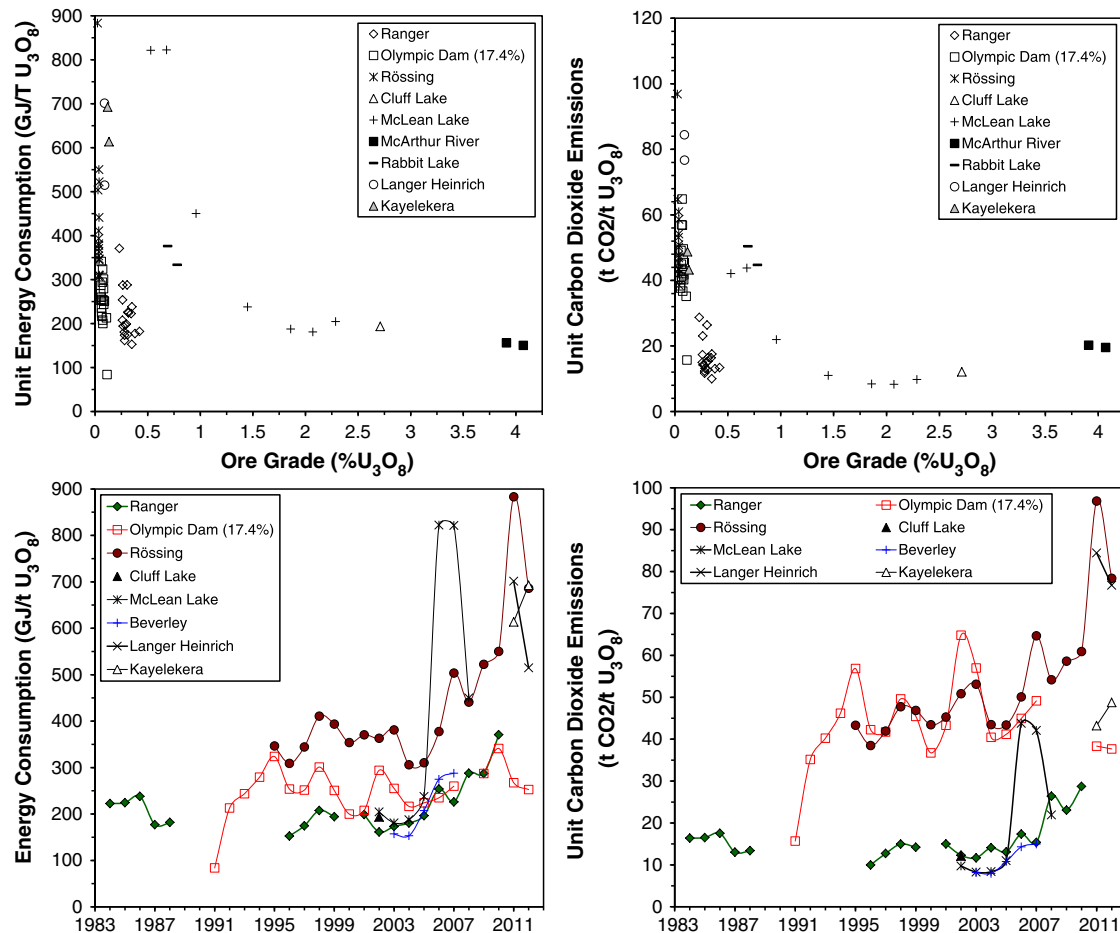


Fig. 9. Energy intensity versus ore grade (top left); greenhouse gas emissions intensity versus ore grade (top right); energy intensity over time (bottom left); greenhouse gas emissions intensity over time (bottom right).

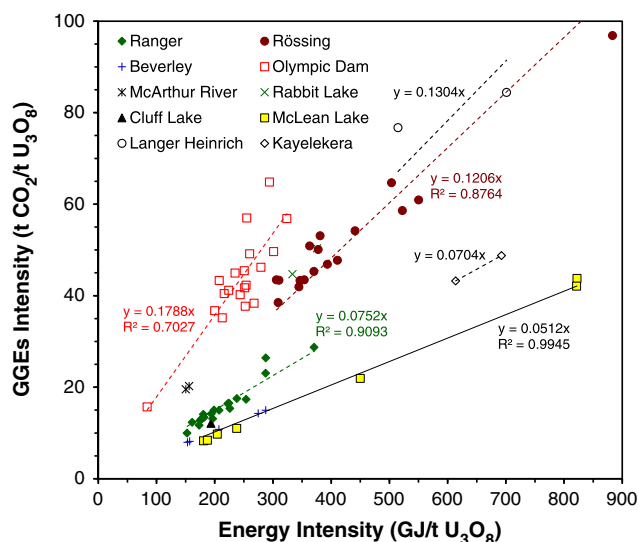


Fig. 10. Greenhouse gas emissions versus energy intensity of uranium production.

policies, new policies and a '450 ppm CO₂' policy scenario, all of which project modest growth in nuclear power although the current policies does reach a peak in 2030 and decline by 2035. Future U demand is estimated based on historical data for U consumption from nuclear generation (data from [OECD-NEA](#) and [IAEA, var.](#)), giving an average of ~205 t U₃O₈/GW_e/year. For each of the IEA scenarios, cumulative U demand by 2035 is 2.53, 2.46 and 2.93 Mt U₃O₈, respectively. If the IEA projections

of nuclear capacity are extrapolated to 2050 and 2100 using basic regressions, the current, new and 450 policy scenarios suggest 678–518–1099 GW_e by 2050, respectively, with 846–498–2104 GW_e by 2100, respectively. Cumulative U demand for these scenarios would 3.98–4.41–5.80 Mt U₃O₈ by 2050 and 9.19–12.42–22.32 Mt U₃O₈ by 2100, respectively – all of which are well within this studies' estimates of U resources (except the very optimistic 450 scenario perhaps).

In considering possible future U requirements, there are several factors which are likely to continue to increase potential supply. These include by-product resources (e.g. phosphates, gold, rare earths), re-enrichment of depleted U residues (as already practised in Russia), discovery of new deposits (very likely given continuing global success), reactors continuing to improve the efficiency of U use (ie. lower t U₃O₈/GW_e/year), continuing action to dismantle nuclear weapons material and downblend this for reactor use (as has been happening for over 20 years), use of mixed oxide fuels (ie. U–Pu fuel or 'MOX'), as well as possible new ore processing technologies to enhance U recovery (especially from by-product or unconventional deposit types).

Overall, there is a strong case for the abundance of already known U resources, whether currently reported as formal mineral resources or even more speculative U sources, to meet the foreseeable future of nuclear power. The actual U supply into the market is, effectively, more an economic and political issue than a resource constraint issue.

6.2. Energy and greenhouse gas emissions

In context, given a standard 1 GW_e reactor would use ~205 t U₃O₈/GW_e/year and this studies' estimate of 32.0 t CO₂/t U₃O₈, this would lead to ~6560 t CO₂/GW_e/year from the mining stage of the nuclear fuel chain. Based on a typical 80% utilisation rate (using IAEA's PRIS



Acid mine drainage (AMD), containing acid, salts, heavy metals and radionuclides, leaking from White's waste rock dump, Rum Jungle, Australia (July 2007, ~21 years after rehabilitation)



Severe AMD impacts on the East Finnis River, Rum Jungle, Australia (August 2011, ~25 years after rehabilitation)



Former open cut of the Mary Kathleen U mine, Australia, including inset of public sign; water quality of the open cut is continuing to deteriorate over time (April 2010)

Fig. 11. A small selection of photographs of environmental problems at the former Rum Jungle and Mary Kathleen U mines (all by the author).



Fig. 12. A tailings dump in the West Rand, Witwatersrand Basin, South Africa, showing wind-blown erosion from the top and acid mine drainage in the foreground (photo's author).

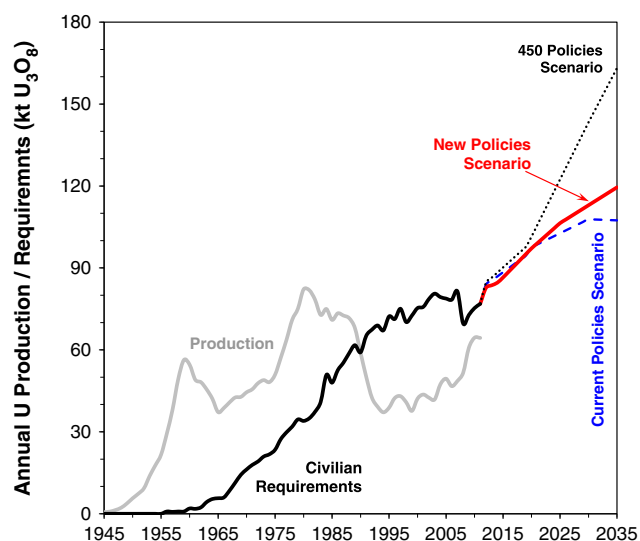


Fig. 13. Civilian U demand with historical U production versus IEA future scenarios for nuclear power.

online database⁴), a 1 GW_e reactor would generate ~7 TWh/year, meaning the ~6560 t CO₂ would contribute a minor ~1 kg CO₂/MWh — or some 0.1% of a typical coal-fired power plant with a carbon intensity of 1 t CO₂/MWh. There are, of course, numerous other steps in the nuclear chain which include greenhouse gas emissions, such as the electricity intensive enrichment stage (or accounting for >100,000 years of long-term high level nuclear waste management), but for mining to begin to materially affect the carbon intensity of nuclear electricity, the carbon intensity of mining would have to increase at least a hundred fold or more.

Based on Eq. 2, an ore grade of 0.001 %U₃O₈ would give rise to ~182 t CO₂/t U₃O₈ — or less than one order of magnitude than the average of current mines. Of course, extrapolating estimates to such very low grades is problematic since at these grades U would be a by-product, or at best a co-product and the energy-GGEs would need to be allocated across all of the various metals extracted (eg. polymetallic shale deposits such as Viken or Håggån). Although the data used to derive Eq. 2 is based on full U allocation only (ie. Olympic Dam has 17.4% of inputs-outputs allocated to U), as the nature of very low grade ores is different, it is reasonable to expect different processing technology to current mines (eg. heap leaching),

with the relative value of by/co-products being very important in determining the allocation of inputs-outputs to U. It does suggest, however, that any future increase in the carbon intensity of U production is unlikely to reach a level which could materially affect the carbon intensity of nuclear power.

7. Summary and conclusions

This paper has provided an extensive review of the mining and milling of uranium (U) ore in the context of a perceived future increase in global nuclear power. Despite its early promise, nuclear power remains a minor source of primary world energy supplies as well as electricity — and its share is projected to continue to decline, especially as renewable energy sources enter substantial growth in the coming decade. Although the detailed evaluation of known U resources shows that there is sufficient U to meet expected nuclear power demands by 2050 and beyond (assuming nuclear power continues), this will increasingly have to be from lower grade deposits. A detailed compilation and analysis of the sustainability metrics of U production, such as energy and water inputs and greenhouse gas emission outputs, shows that they are inversely related to ore grade — meaning that as global average ore grades decline the unit intensity of U production will increase, albeit leading to a minor increase in the carbon intensity of nuclear power. In reality, the primary issues affecting U mining will remain the divisive nature of the nuclear debate — principally nuclear weapons, reactor safety (especially versus alternatives such as renewables) and long-term high level nuclear waste management, as well as the continuing success in the ability of the U industry to achieve and demonstrate successful mine remediation and rehabilitation in the eyes of local, indigenous and environmental communities.

Acknowledgements

This paper is the result of research efforts over many years to continue to expand the understanding of the global situation for uranium mining based on real data. As part of this journey, particular people and organisations need thanks and recognition: Peter Diehl, Gundjeihmi Aboriginal Corporation, Mineral Policy Institute, community environment groups across Australia (especially FoE, ACF, ECNT, CCSA, CCWA), various academic colleagues, several state mining departments (especially the Supervising Scientist Group), anonymous reviewers of earlier papers, and some uranium mining companies for freely providing some data in request (in particular, Energy Resources of Australia; though it should also be noted some companies still refuse data requests). All opinions and views expressed, of course, remain my own.

⁴ The IAEA operates the 'Power Reactor Information System' or PRIS, which publishes annual data on nuclear reactors, capacity and generation; see www.iaea.org/PRIS/home.aspx.

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