



RECENT INITIATIVES TO IMPROVE TAILINGS AND WATER MANAGEMENT IN THE EXPANDING AUSTRALIAN URANIUM MILLING INDUSTRY

R.J. RING

Environment Division, Australian Nuclear Science and Technology Organisation (ANSTO),
Sydney

P.H. WOODS

Energy Resources of Australia — Ranger Mine,
Jabiru NT

H.B. MULLER

WMC Copper Uranium Division,
Adelaide

Australia

Abstract

This paper discusses the environmental and safety related changes that have recently occurred, or are about to be implemented in the Australian uranium milling industry. There are several drivers for these changes. The most important are the significant expansions to the Ranger and Olympic Dam uranium mills, the mining of a new orebody at Ranger and Government permission for the development of the Jabiluka deposit. The major changes in the operation of mines relate to the conservation and recycle of water, an important environmental issue in the arid country surrounding the Olympic Dam deposit, and tailings disposal strategies recently adopted or under consideration. These strategies include methods such as central thickened discharge, and cemented paste-fill for both underground and above ground disposal. The new ICRP 60 recommendations concerning radiation exposure have not been of major concern to the Australian industry, as dose rates have been historically less than the new limits. Current and expected dose rates are discussed in the context of these recommendations.

1. INTRODUCTION

Australia has a long history of uranium mining. The first major production was at Rum Jungle in the Northern Territory between 1954 and 1971. This was followed by Mary Kathleen, which operated over two periods between 1958 and 1982. Australia's second generation of mines commenced production in the 1980s, with the development of the Ranger and Nabarlek deposits in the Alligator Rivers Region of the Northern Territory and Olympic Dam in South Australia. The Nabarlek deposit was exhausted in 1988 and rehabilitated in 1995, leaving only two operating mines.

These mines were developed and operate under strict regulatory control, which requires the implementation of best practicable technology to minimise environmental impact. Integration of rehabilitation/decommissioning requirements into operating plans is an essential feature of the long-term management of these projects. The two operating uranium mines are in diverse environments. The Ranger mine is located in a tropical zone, with wet and dry seasons, and management of run-off water is a key issue. The other site, Olympic Dam, is semi-arid with poor ground water quality, and water utilisation/recycle is a major consideration. Both sites operate under the concept of a restricted release zone, where no liquor entering the zone is discharged outside the zone. The major potential source of release is therefore seepage from the tailings disposal area and methods of tailings deposition are of significant importance.

The strong increases in uranium spot prices in 1995/96 and the predicted shortfall in worldwide production capacity has prompted Australia's existing producers to undertake major expansions. These expansions have offered the opportunity to incorporate recent advances in technology that will lead to improved environmental performance, water conservation and better occupational standards.

This paper describes the recent initiatives of the Australian industry to improve tailings and water management, with particular emphasis on proposals to use dry or semi-dry methods for tailings disposal. Although the new ICRP 60 recommendations concerning radiation exposure have not been of major concern to the Australian industry, current and expected dose rates are discussed in the context of these regulations.

2. CURRENT STATUS OF URANIUM INDUSTRY

Australia has two operating uranium mines; the Ranger mine operated by Energy Resources of Australia (ERA) and the Olympic Dam copper-uranium-gold-silver project of Western Mining Corporation (WMC). Both operations produce uranium by a conventional acid leach/SX flowsheet.

The Ranger mill commenced production in 1980. The nominal capacity of the mill was 3,000 t U₃O₈ from an ore averaging around 0.3% U₃O₈. ERA has now expanded its mill capacity from 1.4 million tonnes ore/year to nominally 2 million tonnes at a capital cost of about \$A50 million. The planned annual production rate is about 5,000 tonnes U₃O₈, commencing in 1998.

The Olympic Dam deposit is one of the world's largest polymetallic orebodies, with known mineral reserves of 11.4 Mt of copper, 0.34 Mt of U₃O₈, 400 t of gold and 2,790 t of silver. Operation commenced in 1988 at an ore production rate of 1.5 million t per year from the underground mine. Uranium and copper production rates were 900 and 45,000 t per year, respectively. Since start-up, several optimization/expansion projects have been undertaken which have increased capacity to the current 3 million tonnes of ore per year, producing 85,000 t copper.

Following the approval of an EIS, submitted in 1997, WMC are currently undertaking a major expansion at Olympic Dam (ODO) to increase copper production to 200,000 t per year. The completion date for the expansion is the first half of 1999. As part of the EIS, a second phase of expansion to a copper production of 350,000 t per year was also approved. WMC has made no formal decision on the implementation of the possible second phase, however for the purposes of modeling scenarios presented in the EIS, an operational date of 2010 was assumed. At the projected production rates, Olympic Dam still has a life in excess of 200 years.

ERA has recently gained approval, through an EIS, for the underground mining of the Jabiluka deposit, which is nearby to Ranger, and for its preferred option to truck the ore to the Ranger mill for processing (Ranger Mill Alternative). Environmental approval has now just been obtained for the alternative of processing ore on the Jabiluka site (Jabiluka Mill Alternative), through the Public Environment Review process (of a lesser scale than an EIS).

Environmental approval for this second milling option was pursued due to the nature of the permission obtained from the aboriginal traditional owners of both Ranger and Jabiluka leases. Although existing agreements from traditional owners to mine and treat ore on each lease are valid, recently traditional owners have stated their opposition to the Jabiluka development and have declined to allow transfer of ore from Jabiluka to Ranger. Negotiations over which option should go ahead will continue with the traditional owners until ERA must make a final decision, in the near future, between the milling options.

A total of 19.5 million tonnes of ore is expected to be mined from Jabiluka at an average grade of 0.46% U_3O_8 yielding approximately 90,400 t U_3O_8 [1]. The lifetime of the operation will depend on the option chosen. For the Ranger Mill Alternative, production would range from 100,000 t per year of ore in year one rising to 900,000 t by year 14 until the final year 29. The corresponding U_3O_8 production would rise from about 670 to 4,000 t per year.

For the Jabiluka Mill Alternative, the mining rate would reach 200,000 t per year in year 2, and stay at this level for the first 10 years. For Stage 2, the mining rate would be increased to 900,000 t per year until the final year 29. The corresponding uranium yearly production rates are 2,500 t U_3O_8 for Stage 1, and 4,000 t for stage 2.

Construction of surface facilities common to both milling options commenced in June 1998. These comprise a water management pond, groundwater supply, surface facilities including a stockpiling area for ore and waste and a portal. The decline (access tunnel) to the orebody commenced in September 1998.

3. RADIOLOGICAL PROTECTION ASPECTS

The implications of ICRP 60 and more recent publications are discussed below, together with a summary of current doses and those projected from changes in operations.

3.1. Introduction of ICRP 60

The introduction of ICRP 60 brings with it several changes to current practice in Australia, these being:

- (a) revised annual average dose limit (down from 50 to 20 mSv);
- (b) revised tissue weighting factors;
- (c) re-emphasis on optimisation of protection (ALARA); and
- (d) the concept of "potential dose".

Because the two operating mines in Australia have managed to operate well within the 20 mSv/a dose range for many years, the introduction of a revised dose limit will not substantially affect either mine. In fact, the major expansions have provided opportunities to further reduce annual doses by improvements in equipment design and increasing ventilation rates.

Revised tissue weighting factors mean that effective dose calculations should now take into account changed dose conversion factors (DCFs). DCFs are considered in greater detail below.

Perhaps the most significant influence of ICRP 60 will be in the area of optimisation of protection. Australia has struggled with a mechanism for turning the ALARA principle into regulation, without a great deal of success in the past. ICRP 60 will mean a greater challenge to both the law makers and the mining companies.

The concept of potential dose seems to have limited application at mines.

3.2. Post-ICRP publications

3.2.1. Dusts

Various publications that followed ICRP 60 have more profound implications for uranium mining. The introduction of both a new lung model and new biokinetic models will mean a complete revision of DCFs. Table I shows how the new models affect DCFs for various mixtures of nuclides.

TABLE I. DCFs (mSv Bq^{-1})

AMAD	ICRP 61 1 μm	New lung model and biokinetics		
		1 μm	5 μm	10 μm
Insoluble U ore	0.20	0.088	0.066	0.040
Insoluble Th ore	0.37	0.081	0.060	0.036
Uranium mill tailings	0.07	0.057	0.045	0.027

Note that the default AMAD used prior to ICRP 60 was 1 μm whereas 5 μm is now the recommended default.

3.2.2. Radon decay products

The ICRP has examined the implications of the new lung model in terms of its implications for a dose conversion factor for radon decay products and has reached the conclusion that the lung model predicts about double the number of lung cancers than are actually observed in epidemiological studies. It has therefore recommended that the factor derived from epidemiology be used, not that from the lung model. The result is that the conversion factors are:

TABLE II. DCF FOR RADON DECAY PRODUCTS

Situation	Recommended value	Units
At home	1.1	$\text{mSv}/(\text{mJ h m}^{-3})$
At work	1.4	$\text{mSv}/(\text{mJ h m}^{-3})$
At home	4	mSv/WLM
At work	5	mSv/WLM

The factor derived from the new lung model is approximately double the values shown in Table II.

There is some debate in Australia as to the appropriate method of taking into account other radon decay product parameters such as the un-attached fraction and the aerosol size. This has not been resolved to date, but could have implications for underground uranium mining.

3.3. Olympic Dam

3.3.1. Mining operations

At Olympic Dam, the mean annual radiation dose in the underground mine in 1995-1996 ranged from just less than 2 mSv for a fitter, to just above 6 mSv for a production charger. Typically, about 10% of the dose arises from alpha-emitting radionuclides in dust, with the remainder of the dose equally split between gamma radiation and radon decay products [2].

The current mining method at Olympic Dam, a variant of sub level open stoping, will not change with the expansion. For this reason, individual radiation exposures are not expected to change with the increase in production rates. This is supported by plant data which shows that the annual mean radiation dose for mine workers has remained virtually constant over the last six years, even though the mining rate has increased from 1.5 to 3 million t of ore per year. As part of the expansion, there will be several changes in ore handling, including the use of driverless trains and remotely controlled crushing and loading stations. These improvements are in keeping with the company's commitment to ALARA to minimize dose.

3.3.2. Metallurgical plant

The metallurgical plant at Olympic Dam is far more complex than a conventional uranium milling operation because of the production of copper, gold and silver. The smelting operation, in particular, contributes a radiation dose not normally associated with a typical uranium processing flowsheet because of emissions of dust and volatile radionuclides. In this respect, Olympic Dam is unique as it is the only site in the world where co-production/processing of uranium and another metal is undertaken.

Like the mine, the annual mean radiation dose in the metallurgical plant has decreased since start-up because of improvements in operational and management practices, even though the plant throughput has doubled. The annual mean radiation dose in 1996 was 1.3 mSv. A breakdown of dose as a function of plant area is shown in Figure 1.

The data in Figure 1 show that the mean annual dose in the hydrometallurgy area (equivalent to a typical uranium mill) was only 1 mSv. Average *potential* doses are further compared in terms of work category in Figure 2, which presents data *calculated on the basis of persons spending the entire year performing the specified task*. This analysis shows, that apart from the smelting operations where the alpha dose is significantly increased, the potential doses in the traditional uranium processing areas would be as shown in Table III, with uranium precipitation/product packaging yielding the highest dose.

As mentioned earlier, ODO has some unique features not normally associated with a uranium milling operation. Two areas of particular importance are the copper smelting operations and precious metals refining. As polonium-210, and to a lesser extent lead-210, are volatile under high temperature

MEAN ANNUAL DOSE (mSv)

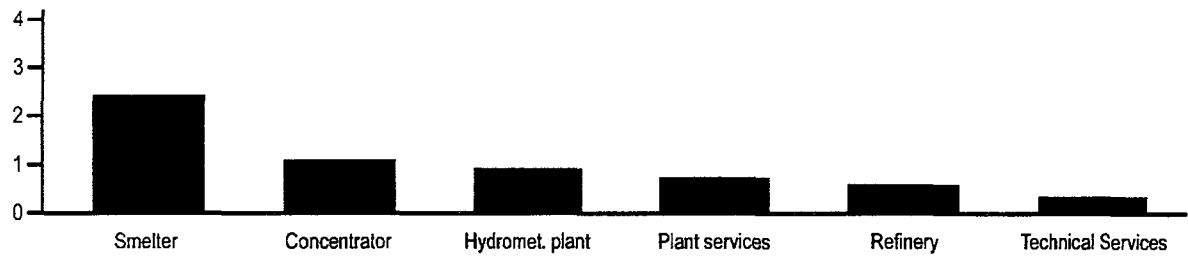


FIG. 1. Mean annual radiation doses at ODO by plant Area (1995-1996).

AVERAGE POTENTIAL DOSE (mSv/a)*

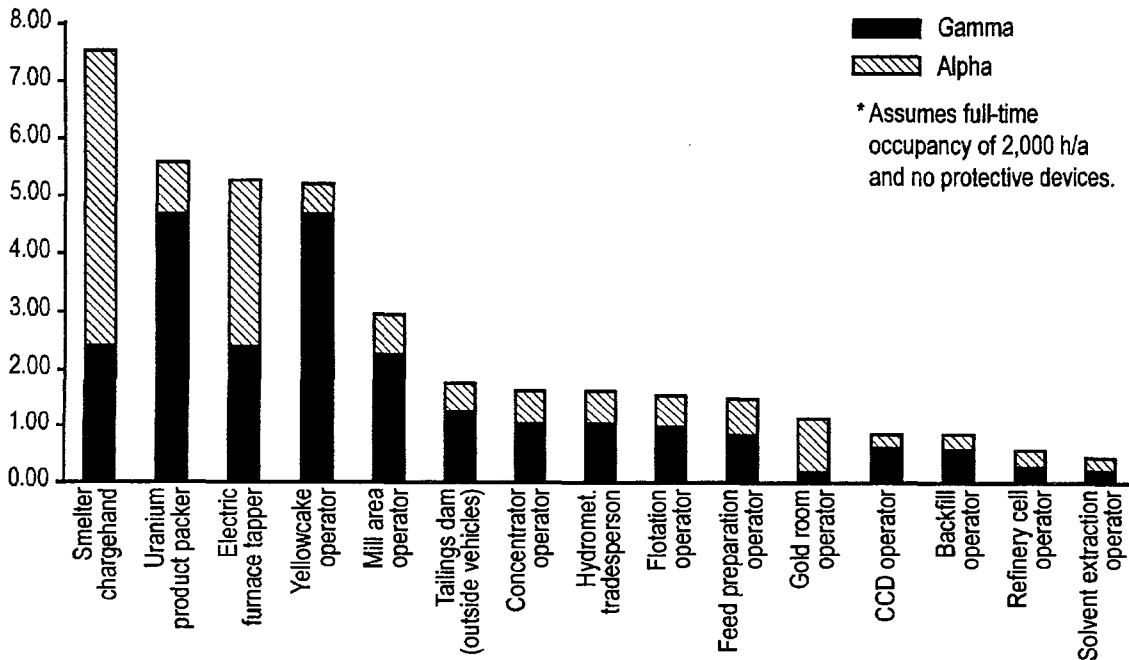


FIG. 2. Combined Alpha and Gamma potential doses in the ODO processing plant (1991-95).

TABLE III. AVERAGE POTENTIAL DOSE* (mSv/a)**

Plant Area	Potential Dose
Uranium product packer	5.5
Yellowcake operator	5.1
Mill area operator	2.9
Tailings dam	1.8
Hyromet tradesperson	1.5
CCD operator	0.7
Solvent extraction operator	0.4

* Source [2], ** For full-time occupancy of 2,000 h per year.

conditions, the inhalation of these radionuclides in airborne dusts is an exposure route that requires close attention. For this reason, control of fume and fugitive dust is an essential component of radiological protection. In this respect, protective measures for conventional

hazards, such as half face respirators to prevent SO₂ inhalation also reduce occupational radiation exposure.

The importance of particulates has also meant that recent international research, such as lung models and biokinetic models, have influenced the assessment of doses. Particle sizing and solubility are both considered when calculating the occupational exposure of the workforce.

In the expanded plant, gamma and alpha exposure rates are predicted to be similar to the existing plant as ore grade and individual exposure times would not change. As part of the expansion, a new calciner and smelter are being constructed and the potential for exposure in these areas will be reduced further by careful design and planning.

3.3.3. Member of the public

Monitoring at ODO has shown that the radiation doses to members of the public from the operation are well below the public limit. The only two local communities are Olympic Dam Village and Roxby Downs, 12.5 and 5 km, respectively from the site. For this location, the atmospheric pathway is the only one of relevance.

In order to predict the possible impact of the expansion on dose rates, it has been assumed by WMC/ODO that dose is correlated with production rates to provide an upper bound estimate. These predictions are compared in Table IV. The predicted radiation doses are between 1 and 3% of the annual average dose limit for members of the public of 1 mSv, over and above background.

3.4. Ranger operation

The expansion of the Ranger mill introduced no changes to equipment of operational procedures that would impact on radiation doses. The annual average dose for the last two years for designated mill workers has varied from 3.8 to 5.5 mSv.

The deposition of tailings in the No. 1 pit and the subsequent removal of the tailings beaches in the tailings dam has reduced the potential γ -exposure from the dam.

TABLE IV. UPPER BOUND ESTIMATES OF DOSE TO DUE TO EXPANSION** (mSv/a)

Production Status	Olympic Dam Village	Roxby Downs
Current	0.022*	0.017*
After expansion	0.053	0.042

* Average of 1991–1996 data, ** Source [2].

3.5. Jabiluka development

3.5.1. Mining operations

The proposed underground mining method at Jabiluka is a variant on long hole open stopping. The drill and development galleries will be in barren, rather than ore zones. The radiation doses predicted for the mine are presented in Table V. The data presented are for the early years of operation when the ore grade will be at its highest level. As the maximum predicted

dose is 14 mSv/a, a management system to control radiation doses will be an integral part of the operation of the underground mine [1].

TABLE V. PREDICTED RADIATION DOSE TO UNDERGROUND PERSONNEL*

Category	mSv/a
Development miners	14
Production miners	9
Backfill and service workers	8
Supervisors	13
Foremen and technical services	10

* Source [3].

A sensitivity analysis of the model used to predict the exposure has indicated that, because of the mining method, it will be necessary to closely monitor gamma radiation in the mine, and to examine all possibilities for dose reduction. It will be a priority at Jabiluka to refine predictions using actual measurements as an increase in the gamma dose by 50% would increase the maximum radiation dose to 25–30 mSv/a [1], which is greater than the ICRP 60 recommended limit.

3.5.2. Metallurgical plant

The estimated annual radiation exposures of personnel working at Jabiluka are shown in Table VI. The estimates are based on data for Ranger. The doses are only 2–3 times those measured at Olympic Dam, even though the uranium grade is almost an order of magnitude greater, because of the higher dust activity at Olympic Dam associated with the smelting operation.

TABLE VI. RADIATION DOSE ESTIMATES FOR JABILUKA PLANT PERSONNEL (mSv/a)*

	Dust	Gamma	Radon Daughters	Total
Plant operator	1.7	0.4–1.4	0.4	2.5–3.5
Maintenance	1.8	0.13–0.8	0.4	2.3–3.0
Workshops	0.8	0.01–0.3	0.4	1.2–1.5

* Source: Jabiluka Mill Alternative PER [3]

3.5.3. Member of the public

Radon from mine vents and emanating from surface stockpiles, together with uranium discharged from the calciner stack was modelled in a dispersion model to estimate the dose to a member of the public. The nearest settlement was taken as Mudginberri, 10 km from the plant. The predicted radiation dose is less than 1% of the annual average dose limit for members of the public [3].

4. TAILINGS MANAGEMENT

The deposition/storage of tailings has always been a major environmental issue facing the uranium, and other mining industries. For this reason, there has been a continual evolution of tailings management practice in the Australian uranium industry. At present, managed sub-

aerial deposition strategy is used at both Ranger and Olympic Dam, although some recent tailings deposition at Ranger has been sub-aqueous.

Although there are many factors to consider, in-place density, impounded solution volume and long-term consolidation are the main factors affecting the overall performance of a tailings storage facility. These potential problem areas stem from the presence of excess water in tailings stored in impoundments and the amount and fate of seepage water and its contained solutes. This is recognized by the Australian industry and there are several proposals under investigation to use dry or semi-dry disposal alternatives. Local conditions also play an important role, particularly in regard to liquor management. For this reason, the disposal alternatives are discussed on a site-by site basis.

4.1. Ranger uranium mine

The No. 1 orebody was mined out between 1980 and 1994. The final open cut is approximately 750 m x 750 m x 175 m deep. In 1996, Ranger commenced development of the No. 3 orebody, which is located about 1.2 km from the No. 1 pit. Production from the No. 3 orebody commenced in 1996 with full-scale operation in 1997. Considerable stockpiles from Pit No. 1 were available to feed the mill whilst Pit No. 3 was brought into full production. A blend of ore is currently fed to the mill.

Until August 1996 tailings from the processing of the No. 1 orebody were pumped to a 107 ha above-ground tailings dam, with a nominal capacity of 15 Mm³. The tailings were originally deposited by the sub-aqueous technique, with a water cover of 2 m. With this method, consolidation of the tailings was poor and the settled density of tailings was less than 1.0 t m⁻³, significantly less than estimated for sizing of the tailings dam. When subsequent research showed that maintaining the tailings in a moist state was sufficient to achieve the desired radon release rate, tailings were deposited subaerially (from 1987) and maintained in a saturated condition. Following this change, settled densities were increased but the average density in the tailings dam remains at a little over 1.0 t m⁻³. Beached tailings in the tailings dam average greater than 1.2 t m⁻³ on an annual basis, although the density of recently deposited beaches is initially lower until draining is complete.

Following environmental approvals, the neutralised tailings slurry from the mill is now being deposited in the mined-out No. 1 pit, which has a capacity of 21 Mm³. Tailings are deposited from several deposition points to enhance beach development and control the location of the decant pond. Sub-aerial deposition is carried out when possible, but following a series of wetter-than-average years the majority of deposition in Pit No. 1 has been sub-aqueous. Various measures have been undertaken to remove excess tailings water. The most significant of these has been dredging of the beaches in the tailings dam to re-establish the dam as a full, year-round evaporation surface. Irrigation of tailings water on the walls of Pit No. 1 will be implemented. This is similar to a measure used at the Nabarlek uranium mine in the mid 1990s [4].

To minimise the permeability of the tailings, and thus minimise the volume of water contained within the tailings that can drain out, ERA is required to achieve a minimum density of 1.2 t m⁻³, to be demonstrated at each 20 m thickness of tailings. ERA has committed to a minimum average density for the filled pit of 1.3 t m⁻³. This is being achieved by the use of an underdrainage system connected to an adit and borehole to accelerate compaction [1]. This approach has proved successful with densities currently running at

1.26 to 1.32 t m⁻³. Higher density will also simplify rehabilitation, as the tailings will eventually be capped with rock and revegetated either as a low hill or as a centrally-draining depression. Provision has been made for possible additional drainage layers or wicks to be used to further increase this figure as the thickness of tailings increases, if required. In July 1998 tailings were equivalent to 68 m depth in the deepest part of the pit.

The in-pit disposal system adopted at Ranger has parallels to those adopted for tailings management at the Rabbit Lake mine [5] and elsewhere in Canada [6]. In-pit deposition of tailings, then capping and revegetation, at the much smaller Nabarlek uranium mine not far from Ranger, was successful without the use of underdrainage [4].

When approvals were given for the initial development of the Ranger No. 1 orebody, it was stipulated that all tailings at Ranger must be deposited in or transferred to the mine pits unless another proposal was made and accepted by the government. The alternative of in-situ rehabilitation of tailings within the existing dam has been under consideration for some time. The criteria to be satisfied was that the any alternative must result in the environment being no less well protected than by depositing or transferring the tailings to the mine pits. The issues involved in the assessment of alternatives are set out by Waggitt [7] and Woods *et al.* [8].

After considering all factors, and in particular the requirements of Best Practicable Technology as set out in the agreements and government conditions imposed on Ranger [8, 9], the recommended best practice for disposal of tailings was judged to be below grade whenever practicable [9]. This decision was endorsed by the ERA board and announced in December 1997. The major reason for preferring this over a tailings dam is there is no risk of failure of the walls of the containment structure over the extended periods required for uranium tailings containment, and that the mildly radioactive material is returned to the geological context from which it came. Some tailings (less than 300,000 m³ to August 1998) have already been transferred from the tailings dam to Pit No.1, largely for process water management reasons. A decision is yet to be made as to whether to transfer the tailings during the remaining mining operations or as part of final rehabilitation when the site is decommissioned.

Transfer of tailings is by conventional dredging with a balancing return of tailings water from Pit No. 1 to the tailings dam. Some remediation of a small permeable section of the upper walls of Pit #1 may also be required [1, 10]. Depending on the location of the mill for Jabiluka ore, the active Ranger pit No.3 may or may not also be required as a tailings repository. There is enough space in the two Ranger pits to store all tailings from both Ranger and Jabiluka projects, as ore reserves currently stand [1, 9, 10]. Depending on the amount of tailings held and resolution of water quality issues, Ranger No. 3 pit may be converted to a lake during rehabilitation [11]. Current indications are that some remediation of more permeable sections of the upper wall of Ranger No. 3 pit may be required if it is used as a tailings repository [1, 10].

4.2. Jabiluka

4.2.1. Paste-fill method

For management of tailings at Jabiluka, should milling proceed there, ERA is proposing the use of a new generation best practice method that is currently being taken up by the mining industry [3]. This method, cemented paste-fill, has evolved from research into the use of

tailings as backfill in underground mines and results in greatly improved deposition characteristics.

In paste deposition, the tailings stream is physically dewatered to produce a high density paste (typically 65–70 wt%), similar to wet concrete, which is then pumped to the tailings containment area. The resulting deposited material has a minimal free water content and a very low hydraulic conductivity. The low water content means that it becomes feasible to add a binding agent such as Portland cement to increase the strength of the tailings and further reduce the conductivity. Paste fill systems also have the advantage that the slimes fraction does not separate from the tailings following deposition. In addition, because most solution is recovered in the paste production process, the costs of recycling solutions from ponds and underdrainage collection networks are virtually eliminated [12].

4.2.2. Underground disposal

Paste fill has been used for underground disposal of tailings in mines in South Africa [13, 14], Germany, Canada and USA.

The current proposal for Jabiluka is to dispose about 75% of the tailings produced as underground backfill. Due to the swell associated with the ore processing operation, only this proportion of the tailings can be accommodated underground in planned voids. Cemented paste-fill is planned for both above ground and underground applications. For underground disposal, approximately 80% of the tailings would receive a 4% cement addition before being pumped to primary stopes. The remainder of the tailings sent underground would have a 1% or more cement addition and would be preferentially pumped to secondary stopes as the underground void became suitable for such disposal [3]. Supernatant water expressed from the tailings would be pumped to a no-release process water management circuit.

Underground paste disposal contrasts with an earlier, more conventional, strategy proposed for a Jabiluka mill alternative [1], where the tailings were to be separated into coarse and slimes (< 20 µm) fractions by hydrocycloning, with the coarse fraction to be mixed with cement for use as underground backfill. The slimes were to be pumped to surface tailings dams and deposited sub-aerially. A particular advantage of paste fill for Jabiluka is that it would remove the need to consolidate and dewater the slimes, which would also require more effort and expense to cap securely. The potential for seepage and interaction of tailings with ground waters would also be reduced because of the inherent characteristics of the paste.

Government approval (August 1998) of the Jabiluka Mill Alternative PER [3] provides for a base case of 100% return of tailings to the deep mine workings, until further studies on the hydrogeology and other aspects of tailings disposal in specific pits are completed to the satisfaction of the authorities. These studies are currently under way and may modify the proposal from the PER described below. Should 100% of tailings be returned to the deep underground, specific voids will need to be created, with waste rock stored on the surface. Detailed mining plans allowing for this potential variation are being prepared.

4.2.3. In-pit disposal

For disposal of tailings that cannot be easily accommodated in the deep mine workings, ERA have proposed that the residual tailings are mixed with a 1% cement addition and pumped to purpose-built open pits. This would involve excavation of pits, specifically to contain the tailings, in benign, non-mineralised sandstone. Another advantage of the paste-fill technique

is the equivalent neutralising capacity of the cement, which could account for any incipient generation of acid. Only a small proportion of Jabiluka tailings are predicted to be acid producing, and some of the acid producing potential will be destroyed during the milling process which is carried out under strongly oxidising conditions.

If pits are used for disposal, it is proposed that they would be an integral part of the water management system. In this case, the pits would be used for storing surplus wet season run-off from the process plant and ore stockpile areas. When the pit contains a sufficient depth of water, the paste would be deposited sub-aqueously from a floating pontoon, with the discharge within 1 m of the surface of the tailings to ensure that the physical integrity of the paste was maintained. At other times, the paste would be deposited across the surface of the exposed tailings using a boom spreader.

4.2.4. Paste-fill production

Current plans for paste fill production are as follows. These may be modified during actual construction, should it proceed, should more appropriate methods be available. The underflow slurry from the CCD washing circuit would be treated with lime in a dedicated circuit to increase the pH to about 5. The neutralised slurry is stored in a large agitated tank to provide surge capacity for the paste fill plant. The neutralised tailings slurry would be filtered on a single belt filter and the resultant filter cake would be repulped with a cement additive prior to being pumped to the disposal areas.

4.3. Olympic Dam

The tailings at Olympic Dam are discharged by sub-aerial deposition into an above ground tailings storage facility (TSF). This method is ideally suited to the Olympic Dam climate where evaporation rates are 14–18 times rainfall.

Supernatant liquor decanted from the tailings is pumped to a series of evaporation ponds. Before discharge, a portion of the acidic tailings slurry is directed to a desliming plant, where the coarse fraction is separated by hydrocycloning, neutralized and used in cemented aggregate fill (CAF) for underground mine fill. At present approximately 4% of the tailings is disposed of underground. The slimes from hydrocycloning are thickened and combined with the bulk of the tailings slurry, which is discharged to the TSF at a solids concentration of 40–50 wt%.

A portion of the liquor from the evaporation pond is combined with tailings prior to deposition in the storage cells. Some of the tailings liquor and contained salts remain within the pores of the tailings solids. This process enables the concentration of dissolved salts in the ponds to be controlled, as excessive levels would result in reduced evaporation rates and precipitation of salts in the evaporation ponds.

4.3.1. Above ground disposal of tailings

At present about 2.7 Mt are discharged annually to the TSF. This will increase to about 7.0 Mt when the expansion is completed. The existing tailings system consists of:

- A paddock method tailings storage facility, comprising three storage cells of about 190 ha total area, with each cell having its own decant facilities for supernatant tailings liquor; and

Two clay and HDPE-lined evaporation ponds, each divided into four cells, with a combined evaporative area of 68 ha. The ponds are used to dispose of tailings liquor and excess acidic process liquor, such as a raffinate bleed and liquor recovered from thickening of the slimes.

The tailings are deposited from spigots along a distribution pipe running along the perimeter walls. A thin layer of tailings about 100 mm thick is deposited during each deposition cycle and allowed to dry for a period of about three to four weeks. The layer reduces to about 60 mm during the drying process. Deposition takes place over a length of approximately 200 m. The tailings beach has an average slope of approximately 2% over the first 200 m and about 1% thereafter. The current height of the perimeter wall is 12 m and the maximum planned height is 30 m.

The densities of the deposited dry tailings are of the order of 1.6–2.05 t m⁻³, with an average of approximately 1.7–1.8 t m⁻³ (particle density is 3.2–3.6 t m⁻³). The moisture content of the tailings is approximately 20–25 wt%, corresponding to a pore saturation of 75–100%.

Two tailings storage systems were considered for the expansion of the facility. These are:

- (a) continuation of the existing paddock tailings storage method, with the construction of additional cells similar to the existing; and
- (b) adoption of a new method, central thickened discharge (CTD) involving further thickening of the tailings slurry and discharge through central risers to form a final tailings profile resembling a series of intersecting flat cones.

4.3.2. Expansion of sub-aerial discharge system

Expansion of the existing system would involve staged construction of another two tailings storage cells with a combined area of 340 ha, increasing the overall area to 530 ha. The maximum height of the embankments would be 30 m. The floor of the storage cells would be covered by a 0.3 m clayey soil liner. Additional evaporation ponds of about 40 ha would also be provided.

A key design criteria of the expanded facility would be minimization of seepage, including minimizing the amount of supernatant liquor on the tailings and maximizing evaporation from the tailings surface. To achieve these goals it is necessary to restrict the rate of rise to 1–2 m per year (current practice), which should ensure removal of all free liquor from the tailings.

Supernatant liquor would continue to be removed from the cells using central decants and then transferred to the evaporation ponds. Seepage from the supernatant pond would be minimized by providing an underdrainage system below the expected maximum footprint of the supernatant pond.

The focus on seepage minimization follows previous detection of seepage from under the TSF and other ponds. The seepage was in part attributed to the operating strategy adopted in the initial years of operation of the TSF. Unlike some sub-aerial systems, water was not withdrawn from a central sump, but was allowed to pond in the middle and evaporate in-situ. As the groundwater contained highly saline water, that was unfit for human or animal consumption, there were no harmful effects on the environment [15].

The tailings system was modified to the present arrangement in 1994 and 1995 to allow for:

- (a) removal of tailings liquor from the top of the cells; and
- (b) construction of the lined evaporation ponds.

4.3.3. Central thickened discharge

The CTD method was evaluated in pilot trials and feasibility studies, and has potential to offer economic and operational advantages at Olympic Dam over the current system. This method, which was originally suggested by Robinski [16], is being used in Australia at four sites; Mount Keith, Gove, Elura and Peak.

The optimum CTD method is tailor made for a particular site to suit the tailings rheology, disposal rate, project life, topography, tailings liquor composition, soil profile and hydrogeology.

At Elura and Peak a single deposition location is used. Elura uses a storage area with three radial partitions. Mount Keith uses a number of central risers and Gove uses a longitudinal stack with a series of risers.

The optimum CTD arrangement developed for Olympic Dam involves the discharge of thickened tailings initially along a series of ridges arranged over existing sand dunes. A number of ridges are used to limit the rate of rise of tailings during the initial beach development. Deposition would occur from a number of spigots arranged along the length of each ridge. Deposition would eventually occur only from the central ridge to form a single longitudinal. The height of the tailings ridge would be 30–35 m above the natural ground level. A pipeline around the perimeter of the outer embankment would convey the collected supernatant liquor back to the process plant or to the evaporation ponds.

Advantages of CTD at Olympic Dam are considered to be:

- (a) greater operational flexibility in terms of rate of production and deposition of tailings;
- (b) the system holds a greater volume of tailings for any given embankment height;
- (c) operating costs are less;
- (d) there are increased opportunities for water recycle as more water is recovered before deposition and stormwater runoff would be greater due to increased surface area. Evaporation potential from the deposited tailings will be greater because of the larger surface area.

Disadvantages are as follows:

- (a) a greater land area is required to store a given volume of tailings;
- (b) higher initial capital costs;
- (c) drains and structures are required over the area not covered by tailings to control and collect supernatant tailings liquor.

Extensive field trials have been carried out which indicate that the maximum solids concentration that can be achieved in a thickener prior to discharge would be 55–60 wt%, resulting in a beach slope of about 2.5% for vertical discharge and 2% for side discharge. For

20 years of production after the expansion, the storage area would about 660 ha. Features of the overall design include:

- (a) tailings delivery pipes would be on causeways on the surface of the tailings;
- (b) supernatant liquor and stormwater are collected in lined reclaim ponds, which would overflow into stormwater ponds following significant rainfall; and
- (c) floor preparation of the area will not need to be as extensive as for the paddock system currently in use as less supernatant liquor would result because of the higher initial solids concentration in the tailings slurry.

Field trials were undertaken to provide data to design a filling strategy that ensures that the tailings mound would be stable under all operating conditions, including extreme meteorological events. The radon release rate from the CTD system would be greater than for the paddock system because of the greater surface area, but this has no significant implications for radiation doses.

4.3.4. *Underground disposal of tailings*

After removal of ore from primary stopes the voids are backfilled with cemented aggregate fill (CAF). CAF consists of crushed waste rock, or rock obtained from a surface dolomite quarry and neutralized deslimed mill tailings or dune sand with the addition of Portland cement and pulverized fly ash. The addition of cement is about 2% wt of the total mixture and flyash is about 4%. The proportion of deslimed tailings (sand) in CAF is about 25 to 30% wt which will represent about 17 to 20% of the total tailings.

Historically the amount of tailings used in CAF has been limited to about 5% of the total tailings due to technical difficulties associated with pumping deslimed tailings and the availability of surplus dune sand which can be used in lieu of the deslimed tailings sand.

The amount of deslimed mill tailings which can be returned to the mine is limited by the requirement to provide an economical CAF with adequate strength.

Paste technology has been considered but is not economically viable at present due to the increased requirement for cement and flyash to achieve the required strength.

5. WATER MANAGEMENT – OLYMPIC DAM CASE STUDY

5.1. Background

The nominal average production rates at ODO for the current and expanded plant are shown in Table VII. The average uranium and copper grades for the period 1998 to 2010 are predicted to be 0.072 and 2.4%, respectively, which are slightly less than in the earlier years of operation.

TABLE VII. NOMINAL YEARLY PRODUCTION AT OLYMPIC DAM

	Ore (Mt)	U ₃ O ₈ (t)	Cu (t)	Au (kg)	Ag (kg)
Existing mill	3.0	1500	85,000	850	13,000
Expanded mill	8.7 – 9.2	4,630	200,000	2050	23,000

5.2. Process flowsheet

Uranium is recovered by a conventional acid leach, solvent extraction, ADU precipitation and calcination process. However, the overall flowsheet is very complex because of the co-production of copper, silver and gold. However, in terms of water management and tailings disposal, the uranium processes tend to dominate. A simplified flowsheet is shown in Figure 3.

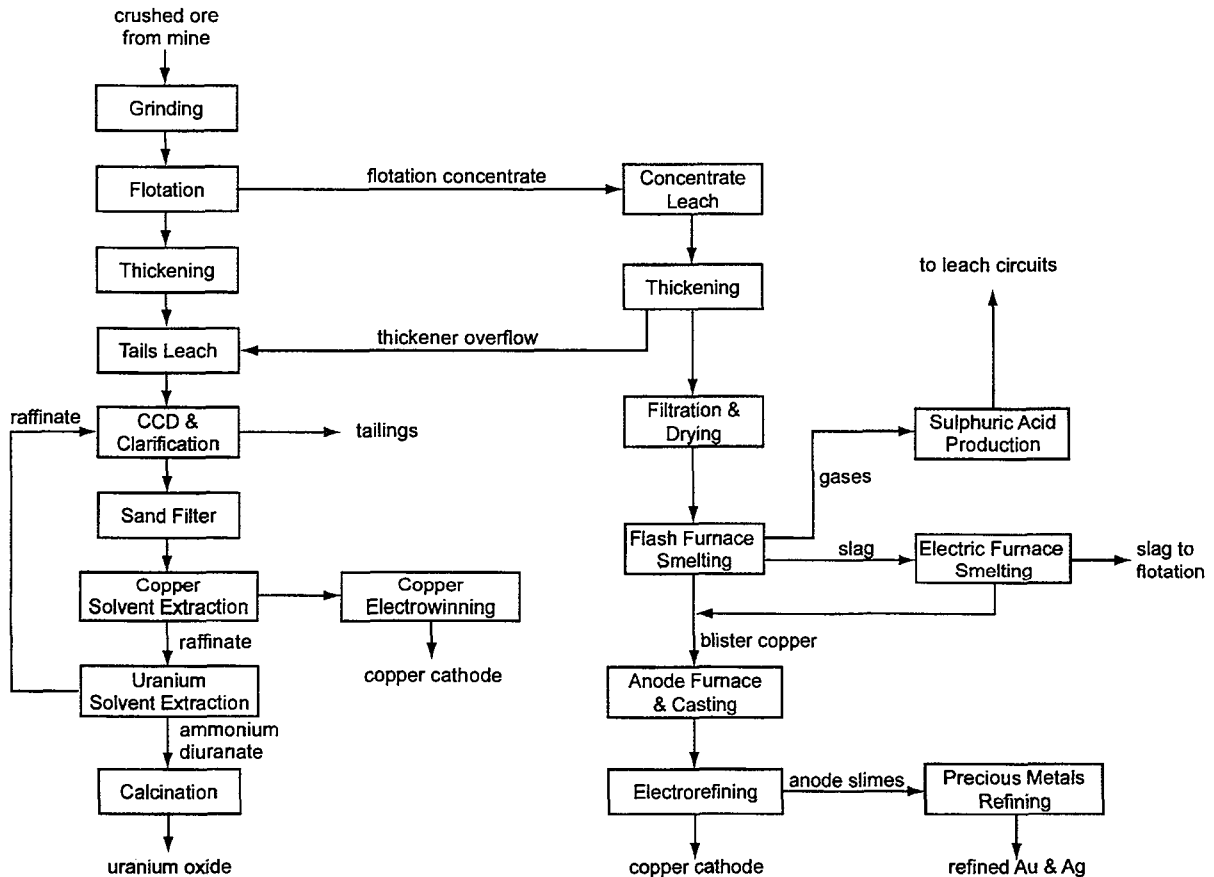


FIG 3. Simplified Flowsheet of Olympic Dam Operation.

The major ore minerals consist of copper sulphides and the uranium minerals, uraninite, coffinite and brannerite. The ore is first floated to produce a copper sulphide concentrate and a flotation tailings, which contain the bulk of the uranium in about 90% of the mass of the ore. The copper concentrate is leached in a dedicated circuit to dissolve uranium that reports to the concentrate. After solids/liquor separation, the leach liquor is added to the main uranium leaching circuit. The leached concentrate is smelted to produce blister copper, which is subsequently converted to anodes that are electrorefined to produce high purity cathode copper. A high proportion of the gold and silver in the ore report to the anode slimes generated in the electrorefining process. The slimes are treated to produce high purity gold and silver.

The flotation tailings are thickened from 28 to 62-63 wt% and then fed to the tails leach circuit. The neutral liquor from the thickener overflow is recycled to the milling circuit. In the tails leach, uranium, and a substantial proportion of the remaining copper not recovered by flotation, are leached at a temperature of 55°C and at a free acidity of about 10 g L⁻¹. The

leached solids are washed in a CCD circuit using recycled raffinate. The pregnant liquor is first treated by solvent extraction to remove copper, which is recovered from the strip liquor by electrowinning. Uranium is then recovered from the copper raffinate in conventional solvent extraction/ammonia precipitation circuits.

The tailings solids from the CCD circuit are pumped to a storage facility.

5.3. Expansion

In the expansion, the mining method and metallurgical processes remain essentially unchanged, and are thus well understood. The major areas re-assessed to take advantage of recent advances in technology and changes in environmental requirements include:

- (a) the sustainable supply of water;
- (b) the containment of tailings; and
- (c) the management of radiation exposures.

The latter two issues are covered in Sections 3.1 and 4.3. The importance of water supply is discussed in detail here.

5.4. Water supply

Groundwater at Olympic Dam is highly saline, with a total dissolved solid in the range 20–40 g L⁻¹. Owing to its salinity, the groundwater is not used as a plant water supply, but is extensively used underground for dust suppression and drilling.

Water for potable and process uses at Olympic Dam is obtained from the Great Artesian Basin (GAB). The GAB is a groundwater basin that underlies about 1.7 million km² of central Australia. The total water storage is estimated to be 8,700 million ML.

The current total raw water use from the GAB is 15 ML/d. Potable water is produced in a desalination plant and is used in the local town and plant for personnel facilities and in processing where desalinated water is required. Process water (TDS of 2,300 mg L⁻¹) is used in the metallurgical plant for processing operations. It comprises raw water and the reject water from desalination. Process water accounts for 70% of the supplied bore water.

The cost of water at Olympic Dam is quite high, as shown in Table VIII. This factor, and a desire to conserve water, has driven the implementation of water minimisation programmes.

There are essentially two process water circuits in the overall metallurgical process (see Figure 4). The first is the neutral circuit, which comprises the milling and flotation circuits.

TABLE VIII. WATER COSTS* (\$/kL)

	Cost
ODO Process water	1.61
ODO Potable water	2.40
City water**	0.88

* Source [2], ** Water pricing in major capital city.

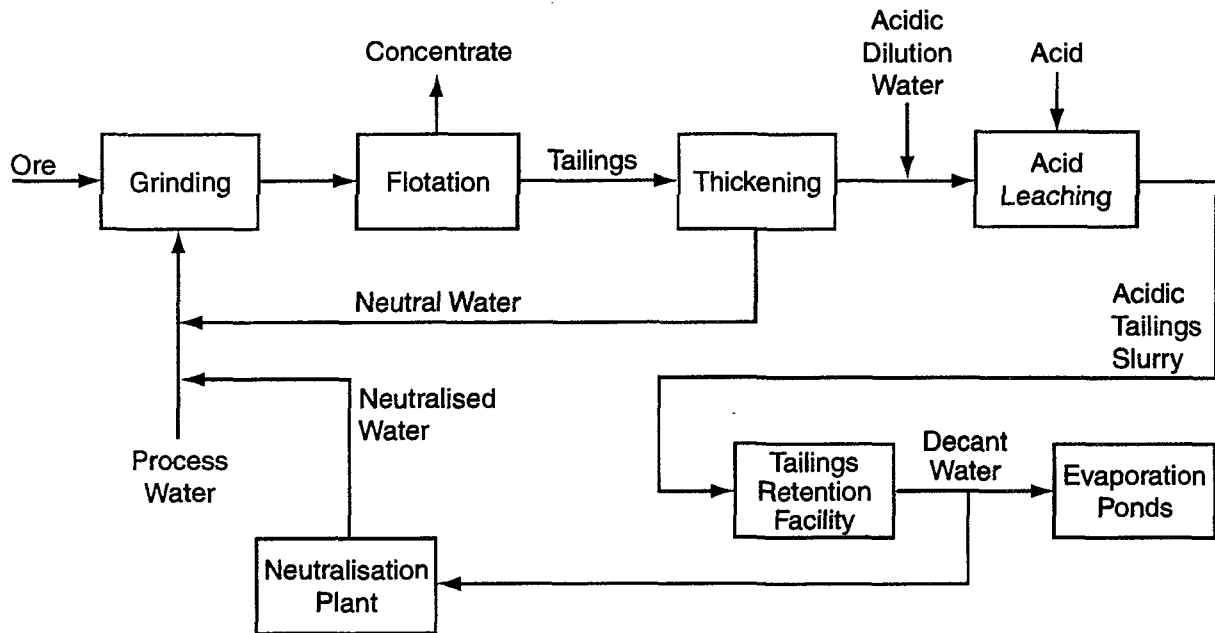


FIG 4. Process water circuits at Olympic Dam.

This circuit is a major consumer of water, with water “lost” from the circuit with the flotation tailings thickener slurry. The lost water enters the acidic liquor (leaching) circuit and exits the plant with the leached tailings.

Water usage in the plant will increase to about 30 ML/d when the expansion is completed. This equates to 1.24 kL t⁻¹ of ore milled, compared to a pre-expansion usage of 1.57 kL t⁻¹, a reduction of 21%. This saving will be achieved, despite increased water usage in some scrubber operations, by the following measures:

- (a) installation of high compression thickeners in the concentrator for increased recovery of neutral process water; and
- (b) construction of a neutralisation circuit to allow treatment and recycle of acidic tailings water to the neutral, front end of the process.

The first measure involves the use of 2 × 38 m diameter, high compression thickeners, each handling 550 t h⁻¹, to dewater the flotation tailings feed to the main uranium leaching circuit. The thickeners will have a nominal mud height of 3 m and be equipped with an educt system for feed dilution to 15 wt% solids. Compared to conventional thickeners, the underflow density will be increased on average from 62.5 to 70-72 wt%, recovering an additional 4.5 ML/day for recycle to the milling circuits. As the increased density is too high for effective agitation in the subsequent leach circuit, acidic tailings liquor recovered from the tailings storage facility will be recycled to achieve the desired density in leaching.

Recycle of acidic tailings liquor to the process is an obvious approach to further reduce the net water consumption of the process. To make a significant impact in this regard, water must be recycled to the grinding circuit where the ore is first slurried, but to avoid corrosion of the grinding mills and provide the appropriate conditions for the flotation circuit, a near neutral pH is required.

As part of the expansion, a neutralisation circuit is being constructed, partly using redundant equipment. The circuit will treat of 5 ML d⁻¹ of acidic water recovered from decanting/settling of leached tailings slurry. The circuit consists of six agitated tanks and four small high rate thickeners. Initially, lime will be used to raise the pH from about 1 to 7, but this will probably be replaced by calcined dolomite, mined from a local quarry. A feature of the process is the incorporation of the HDS approach of recycling thickener underflow to increase the settled density of the gypsum/metal hydroxide sludge produced.

For the Olympic Dam acidic liquor, which has a TDS of 45 g L⁻¹, the HDS process should allow the recovery of 94% of the liquor, compared to 87% from conventional neutralisation, where a more gelatinous sludge is produced. The volume of sludge produced by the HDS process is also about 50% less than from conventional neutralisation. The sludge will be disposed of in the tailings retention system. The decision to apply the HDS process to the treatment of liquor with a high total dissolved solids content was based on the results of continuous mini-scale trials using both lime and calcined dolomite conducted at ANSTO [17].

The scope for increasing the recycle of acidic water at Olympic Dam is limited by the potential build-up of salts and organic contaminants which could have a detrimental effect on process efficiency. Chloride ion introduced by the use of sodium chlorate as an oxidant in uranium leaching and as a constituent of the ore (from groundwater) is the major limiting factor as its concentration reduces the loading of uranium in solvent extraction.

6. CONCLUSIONS

Significant expansions in production have given Australia's two uranium producers the opportunity to adopt/assess advances in technology that will lead to improved environmental performance. The most significant changes that have taken place or are being considered involve tailings deposition. These include:

- (a) the decision to transfer tailings from the Ranger tailings dam to the mined out pits for final containment and rehabilitation;
- (b) the deposition of the current production Ranger tailings into a mined-out pit with an underdrainage system to ensure a high settled tailings density;
- (c) the plan to use the cemented paste fill method for the placement of tailings in the proposed underground Jabiluka mine;
- (d) the continued assessment of the more novel use of the paste method for disposal of tailings in purpose-built open pits; and
- (e) consideration of the central thickened discharge method as an alternative to the existing sub-aerial discharge system for the expansion of the Olympic Dam tailings storage facility.

The other major area of environmental importance to receive attention is water conservation in arid regions. By using more efficient solid/liquid separation equipment and installing a treatment circuit to neutralise acidic water, the Olympic Dam operation will reduce the water consumption per tonne of ore milled by in excess of 20%.

The new ICRP 60 recommendations will not, in general, impact on the Australian uranium mining operations. Nonetheless, a careful management system will be required to control radiation doses at the underground mine proposed for the Jabiluka development.

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