

RECOVERY OF BYPRODUCT URANIUM FROM THE MANUFACTURE OF PHOSPHATIC FERTILISERS

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Introduction

Uranium is known to be associated with deposits of phosphate rocks. The concentration of uranium in the rock varies from deposit to deposit, but generally it is in the region of 30-200 $\mu\text{g g}^{-1}$ and is thus one to two orders of magnitude lower than that in uranium mineralisation (typically 1 000-5 000 $\mu\text{g g}^{-1}$). The size of phosphate deposits suggests that they could provide a significant additional medium cost uranium resource. In certain circumstances, the recovery of uranium at a low cost can be achieved if the phosphate values in the rock also are recovered. Consequently, the methods for uranium recovery are designed to obtain uranium as a byproduct from existing processes for the production of fertilisers.

This paper briefly reports the current methods of processing phosphate rock to produce fertilisers, and reviews the merits of uranium recovery processes proposed for integration with the fertiliser industry. Also discussed are factors which influence the amount of uranium which can be recovered, such as the origin of the rock and the types of fertiliser produced.

Phosphate Rock

Approximately 85% of the phosphate rock mined in the world is used to manufacture phosphatic fertilisers⁽¹⁾. It is also used in the manufacture of elemental phosphorus and various phosphate chemicals (particularly in the detergent industry), feed stocks, the smelting of iron, the production of steel alloys, and for the corrosion protection of metal surfaces.

Generic types

Phosphate rock is a commercial term for rock containing one or more phosphate minerals, usually apatite ($\text{Ca}_5(\text{F}, \text{Cl}, \text{OH})(\text{PO}_4)_3$), of sufficient grade and suitable composition to permit its use, either directly or after concentration, in manufacturing commercial products⁽²⁾.

The principal generic types of commercial interest are:

- (i) Bird guano (excrement of certain sea-fowl) on coastal islands (a relatively rare deposit).
- (ii) Apatite veins and segregations in igneous rocks.
- (iii) Primary marine phosphorites and their derivatives.

The phosphorites, which are marine sedimentary deposits composed largely of phosphate minerals, are the most abundant type of deposit and supply almost 85% of the world's phosphate requirements.

Composition

Phosphates are assessed mainly according to their phosphorus pentoxide (P_2O_5) content. Pure apatite contains about 40% P_2O_5 , but phosphate rock has a wide range of P_2O_5 contents with an average of 31% P_2O_5 (68% tricalcium phosphate)⁽³⁾. The concentrations of other constituents in the rock, in particular, iron oxide and alumina, fluorine and calcium carbonate, are also important as they dictate the type of fertiliser process in which the rock can be used and the degree of operating difficulty that will be encountered in using it.

Uranium content

The radioactivity of phosphate rock was first noted in 1908 when it was found that samples of phosphorite were many times more radioactive than the average rocks of the Earth's crust. This radioactivity is due almost exclusively to uranium and its decay products which are in equilibrium with the uranium^(4,5,6).

Most phosphate rocks contain between 30 and 200 $\mu\text{g g}^{-1}$ of uranium but contents higher than 1 000 $\mu\text{g g}^{-1}$ have been reported⁽⁵⁾. Guano and macrocrystalline apatite contain significantly less uranium than the marine phosphorites. This relationship between the uranium content and origin of the rock is a well known phenomenon^(4,7).

Production, consumption and trade

In 1973, the world production of phosphate rock was nearly 100 million tonnes⁽⁸⁾. There are six production areas (USA, USSR, North Africa, Middle East, Togo/Senegal and the Pacific Islands) currently supplying some 95% of the world's production of phosphate rock. The USA, USSR, and North Africa together account for approximately 85% of world production and dominate the export trade which amounted to 52.6 million tonnes in 1974^(1,8,9).

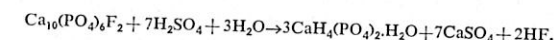
World production of phosphate rock should increase by over 60% (average annual growth rate of more than 10%) from 1973-78; during the same span, exports are expected to jump by 58%⁽⁹⁾. The increase in production capacity will be due to enlargement of existing facilities (USA, USSR, North Africa, Togo/Senegal) and the opening of new mines in the Spanish Sahara, Peru, India and Australia⁽¹⁾.

Processing of Phosphate Rock to Produce Fertilisers

The aim of processing phosphate rock is to convert the phosphate in the rock to a form available to plants; only 5-10% of the P_2O_5 in phosphate rock is available from unprocessed rock⁽¹⁰⁾. The primary phosphate-based fertilisers are produced by a number of relatively simple processes, but the overall industry is complex because of the varying degrees of mixing and blending that are practised.

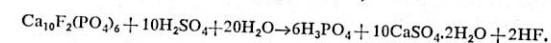
There are three basic processes for decomposing phosphate rock to obtain products suitable for use as fertilisers; these are acid treatment, thermal reduction (furnace process) and thermal treatment without reduction. Acid treatment is the most common method. The main acid treatment route involves treating pulverised phosphate rock (beneficiated) with

sufficient sulphuric acid to give a mixture of mono-calcium phosphate ($\text{Ca}(\text{H}_2\text{PO}_4)_2$) and gypsum ($\text{CaSO}_4 \cdot 2\text{H}_2\text{O}$), known as single superphosphate. Initially, a slurry is formed which is held in a container until it solidifies. The material is then removed to a storage pile where it cures for approximately three weeks until the reaction between the rock and the acid is complete. The main reaction is:

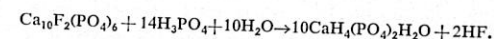


The gypsum is not removed from the mixture which seldom contains more than 20% soluble P_2O_5 .

If greater quantities of sulphuric acid are added to phosphate rock, phosphoric acid (known as wet-process phosphoric acid) is produced which is filtered from the gypsum. The reaction for the process used most extensively (the dihydrate process) is:



Triple superphosphate, which contains 44-47% soluble P_2O_5 , is obtained by acidulating phosphate rock with phosphoric acid. The basic reaction is:



Phosphoric acid is also combined with other chemicals to produce highly concentrated fertiliser salts, such as ammonium phosphates.

Nitric and hydrochloric acids are also used to treat phosphate rock, but these processes are limited in application.

In recent years, there has been a marked change in favour of the manufacture of high-analysis fertilisers in preference to the traditional single superphosphate. This trend is shown in the decline from 50% in 1960 to 29% in 1969 in the proportion of world phosphate rock consumption used in the manufacture of single superphosphate⁽²⁾.

The world consumption of phosphate fertiliser was predicted to rise from 22.4 million tonnes/a in 1971-72 to 30.2 million tonnes/a in 1975-76. The production of wet-process phosphoric acid was expected to account for more than 90% of the overall growth in phosphates, with acid consumption expected to reach 17.2 million tonnes of P_2O_5 /a in 1975-76. As a result of this development, phosphoric acid will increase its share of the total world production of phosphate rock to well over 50%⁽⁹⁾.

Processes for the Recovery of Uranium from Phosphate Rock and Solid Fertilisers

Attempts have been made to recover the uranium values during all steps of the processing of phosphate rock to produce fertilisers. Present technology indicates that significant quantities of uranium can only be recovered at a reasonable cost as a byproduct from the manufacture of wet-process phosphoric acid⁽¹¹⁾. Consequently, only the more interesting processes for the recovery of uranium from phosphate rock and solid fertilisers are reviewed in this section.

Recovery of uranium from phosphate rock by wet-processes

Initial attempts to extract uranium selectively from phosphate rock were not successful⁽¹²⁾. Alkaline leach methods were ineffective, and acid leaching resulted in the dissolution of uranium in approximately the same proportion as the dissolution of the phosphate rock because uranium is uniformly distributed throughout the structure of the rock.

The Israel Mining Industries (IMI) phosphoric acid process involves the digestion of raw phosphate rock with hydrochloric acid. They developed separate processes for the recovery of uranium from either the dissolution liquor or the undissolved residues. In the latter process, acidulation of phosphate rock was carried out under reducing conditions to give a residue containing 82% of the original uranium in the rock. The residue was roasted at 700°C and the uranium was leached from it under oxidising conditions with hydrochloric acid to yield a solution containing 90% of the uranium from the residue. A precipitate containing 1-2% uranium (95% recovery) was obtained by first reducing and then neutralising the leach liquor to pH 2 with ammonia. A concentrate of 60-70% uranium was obtained from this precipitate by carbonate leaching and re-precipitation with sodium hydroxide. Overall uranium recovery was 50%⁽¹³⁾. The estimated cost of recovery in Israel was US\$27.5/kg U₃O₈ (August 1970) for an annual uranium production of 50 tonnes with the consumption of raw materials accounting for half of this cost.

Recovery of uranium from single superphosphate

Recovery of uranium has been achieved by solvent extraction of the acidulated phos-

phate rock in the short period before it sets to the final single superphosphate product, and by recovery from the cured single superphosphate.

In the former process⁽¹⁴⁾, a solution of 10 vol. % octyl pyrophosphoric acid (OPPA) in kerosene extracted 50% of the uranium with a contact time of 10 minutes and a solvent to rock ratio of 0.03. Long and Valle-Riestra⁽¹⁵⁾ developed the latter type of process to recover up to 80% of the uranium from the cured superphosphate product. The process consisted of extracting uranium from the ground superphosphate by percolation with 1-2 vol. % solution of OPPA in hexane at about a 1:1 ratio of organic to ore, and stripping the uranium (after reduction by ferrous sulphate) from the loaded OPPA with hydrofluoric acid.

Although a high ratio of solvent to ore was required, a concentration of 2 vol. % OPPA in the solvent was sufficient and only brief contact times were necessary. After treatment by this procedure, superphosphate was found to have the same weight and P₂O₅ analysis. However, no agronomic tests of the material were made.

A major problem with these processes was the loss of solvent to the acidulated rock in sufficient amounts to make the cost of uranium recovery prohibitive. Another problem was the instability of the OPPA solvent which would make recycling a different proposition.

Up to 90% of the uranium was recovered from cured single superphosphate prepared from calcined rock by countercurrent leaching with water⁽¹²⁾. The addition of metallic iron to the leach solution removed the uranium as a 2-4% precipitate. The major disadvantage of this process was the dissolution of 89-98% of the phosphate during the leaching step which must be recovered by a relatively expensive evaporation of the uranium-free solution.

Recovery of uranium from triple superphosphate

Two methods were developed which involved water leaching of the superphosphate and subsequent evaporation to reconstitute the solid fertiliser and recovery of uranium from the superphosphate by solvent extraction⁽¹²⁾. While triple superphosphate was found to be more amenable than single superphosphate to solvent leaching, the disadvantages of these processes were the same as for the single superphosphate processes.

Recovery of uranium from monocalcium phosphate

The monocalcium phosphate process was developed to eliminate the evaporation step needed in the superphosphate process and to make all of the uranium in phosphate rock available for recovery⁽¹⁶⁾. In contrast, recovery from phosphoric acid during the manufacture of triple superphosphate is limited to a maximum of approximately 67% of the uranium in all the rock processed.

The process consisted essentially of digesting calcined rock with an excess of phosphoric acid and precipitating uranium from the resulting solution which was then cooled to crystallise the monocalcium phosphate. The precipitation step produced a low grade precipitate which was very difficult to wash and filter. The process also had the disadvantage of requiring modifications to existing practice in the fertiliser industry.

A second process was proposed in which 90% of the P₂O₅ and 80-90% of the uranium were leached from uncalcined rock by countercurrent contact with 25-30% sulphuric acid to produce a solution which was principally monocalcium phosphate⁽¹²⁾. The uranium was then removed as in the previous process. However, this process did not offer significant advantages over recovery from wet-process phosphoric acid.

Recovery of Uranium from Wet-Process Phosphoric Acid.

Approximately 75-80% of the uranium in the phosphate rock is in a recoverable form in phosphoric acid⁽¹⁷⁾. The remainder is in two waste streams, the gypsum from the filters and the water from the fume scrubber. The recovery of uranium from phosphoric acid solutions poses many problems. Precipitation, ion exchange, adsorption, neutralisation and solvent extraction were investigated extensively before neutralisation and solvent extraction emerged as the most successful methods.

Neutralisation methods

Neutralisation techniques have been applied only in the manufacture of ammonium phosphates and technical grade phosphate chemicals, where the neutralising step does not interfere with fertiliser production.

In 1952, the first commercial plant to recover uranium as a byproduct from the processing of phosphate rock (1 500 tonnes of phosphate

daily) was built in Illinois by the Blockson Chemical Company^(18, 19). This company produced technical grades of sodium phosphate; consequently, the phosphate rock was calcined as a pretreatment to eliminate organic constituents which could not be tolerated in the product. The process began with the partial neutralisation of wet-process phosphoric acid to form monosodium phosphate at a pH of 4-5. The bulk of the impurities, largely calcium, iron and aluminium phosphates and silicates were precipitated at this stage and filtered off. Addition of a strong oxidising agent, such as chlorine, to the acid before the neutralisation step resulted in retention of about 80-90% of the uranium in solution. After filtration, the addition of sodium hydrosulphite to the filtrate precipitated 90-95% of the uranium in solution as a gelatinous uranous phosphate. This product was filtered off and upgraded by a simple procedure to a product containing 40-60% U₃O₈. Evaluation of the process indicated that 60-70% of the uranium in the phosphate rock could be recovered at a very reasonable cost.

Although this method produced a lower uranium recovery than solvent extraction, it had considerable merit in such areas as simplicity of control, cost of installation and reagent requirements and consumption⁽²⁰⁾.

Recovery of uranium by solvent extraction

The organic solvent is the most critical feature of the solvent extraction process. Octyl pyrophosphoric acid was the solvent initially selected for pilot plant investigation. Unfortunately OPPA is chemically unstable and tends to hydrolyse readily to octyl orthophosphoric acid (OPA) which is a less effective extractant by a factor of at least ten. The hydrolysis is accelerated in the presence of strong acids and by increased temperatures^(4, 21). A process using a 1-2 vol. % of a solution of OPPA in kerosene was investigated extensively in the laboratory^(21, 22), followed by pilot-plant testing, and construction in 1955 of an industrial-size plant in Florida by the International Minerals and Chemical Corporation (IMC)⁽²³⁾.

The IMC plant and process

In spite of previous pilot-plant experience, considerable difficulty was encountered with the operation of the commercial-scale plant of the IMC process^(20, 23). The initial acidulation released about 70-90% of the uranium in the concentrated rock, and yielded a uranium concentration in the phosphoric acid which aver-

aged about 0.16 g l⁻¹. After cooling and clarification, the acid was percolated through reduction pits containing scrap iron to reduce the uranium to the tetravalent state (solution e.m.f. of 0.0 V) before solvent extraction. The pits were replaced by a rotary stainless steel drum operated in the same manner as a ball mill because the stationary bed of scrap iron passivated rapidly owing to the formation of an adherent coating of an iron phosphate and gypsum⁽²⁰⁾.

Several different types of equipment for solvent extraction were tried. Only centrifuges proved satisfactory for separating the organic and aqueous phase. After the extraction step, the pregnant liquor was contacted with 25% sulphuric acid to remove calcium as this and other metal ions are also complexed by the pyrophosphate ester resulting in a reduced extraction coefficient. In addition, emulsions tended to be stabilised by increased amounts of suspended solids, much of it gypsum, which collected at the organic-aqueous interface.

The solvent inventory in the plant was recycled about every 24 hours. Under these conditions, the loss of the pyrophosphate ester through degradation and other causes was about one-third of the inventory/cycle. Uranium was removed from the loaded solvent by a single stage, stripping operation using a 12% solution of hydrofluoric acid. The precipitated uranium was recovered by settling and then centrifuging.

The final product composition varied widely because of the many variables in the processing operation. Typically, the product contained 40-60% U₃O₈, 12-15% fluorine and up to 5% calcium. The recovery of uranium from the phosphoric acid feed to the plant varied widely. After 18 months operation, the plant was expected to have a realistic maximum recovery of 75% of the uranium in the feed. The IMC solvent extraction process for recovery of uranium from wet-process phosphoric acid proved difficult to control because of wide fluctuations (up to 50%) in the uranium concentration in the daily phosphoric acid feed to the plant⁽²⁰⁾.

Oak Ridge solvent extraction processes

An improved extraction process was developed⁽²⁴⁾ at the Oak Ridge National Laboratory (ORNL) to overcome the deficiencies of the IMC process. The synergistic extractant combination of di (2-ethylhexyl) phosphoric acid (D2EHPA) plus trioctylphosphine oxide

(TOPO) dissolved in an aliphatic diluent was found to be a superior solvent. For this extractant to be effective, the uranium must be in the hexavalent form.

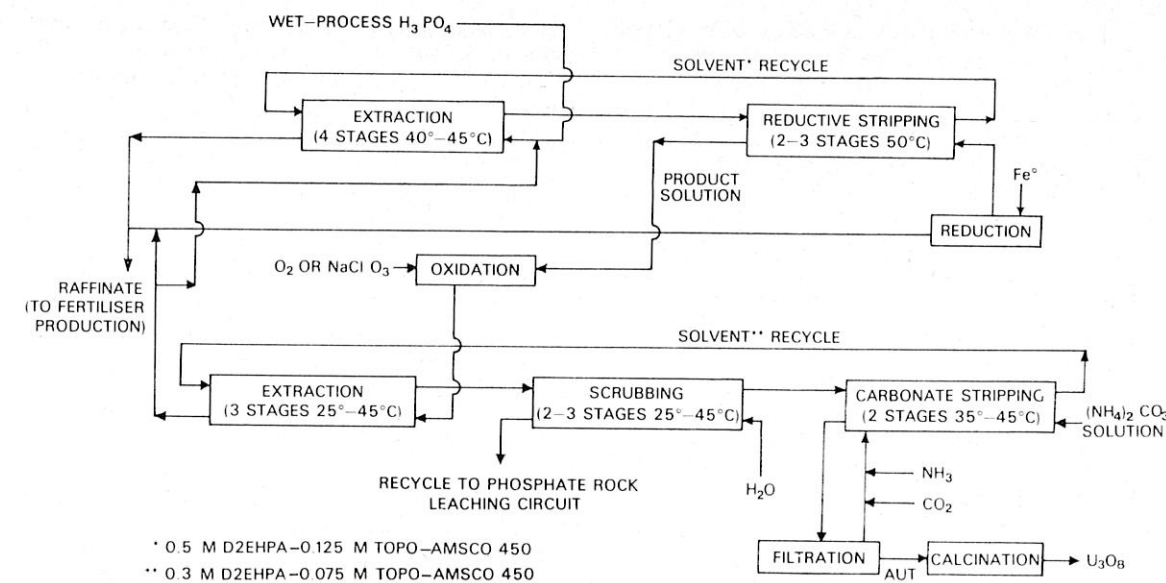
A two-cycle process flowsheet (process I) using this extractant was tested successfully on a bench scale⁽²⁵⁾ (see Figure 1). In the first cycle, the uranium was recovered from the solvent (0.5 M D2EHPA—0.125 M TOPO in Amsco 450) by contacting it with a phosphoric acid solution containing ferrous iron which reduced the uranium to the less extractable tetravalent state, effecting its transfer to the aqueous phase. Under the proper conditions, uranium strip solutions containing about 12 g l⁻¹ of uranium were readily obtained.

In the second cycle, the uranium in the first-cycle strip solution was oxidised to the hexavalent state and then extracted with 0.3 M D2EHPA-0.075 M TOPO in Amsco 450. The use of the reductive strip in the first cycle to produce a solution concentrated in uranium allowed re-extraction and stripping of the uranium under more favourable conditions. In addition, the low concentration of uranium in the acid feed to the first cycle limited the amount of uranium that could be loaded into the organic phase. However, in the second cycle, the solvent was efficiently loaded because of the higher uranium concentration. The two-cycle operation resulted in a 95% recovery of uranium from the acid compared with an average recovery of 82% in the absence of a reductive strip section⁽²⁶⁾.

The two-cycle process was run continuously with wet-process acid samples, using mixer-settlers for periods equivalent to about 40 cycles of the solvent through the stripping section, without any significant change in performance or problems with phase separation.

The organic extract was scrubbed with water to remove extracted phosphoric acid and then stripped with an ammonium carbonate solution under conditions that resulted in direct precipitation of the uranium as ammonium uranyl tricarbonate (AUT). The AUT precipitate analysed as 12.7% NH₃, 44.9% U, and 34% CO₃ after being washed and then air-dried. Calcination of the air-dried precipitate for two hours at 600°C yielded a product that analysed as 97.5% U₃O₈.

The high cost and unavailability of TOPO in commercial quantities are serious disadvantages in the ORNL process, but it was shown that a mixture of octylphenylphosphoric acid



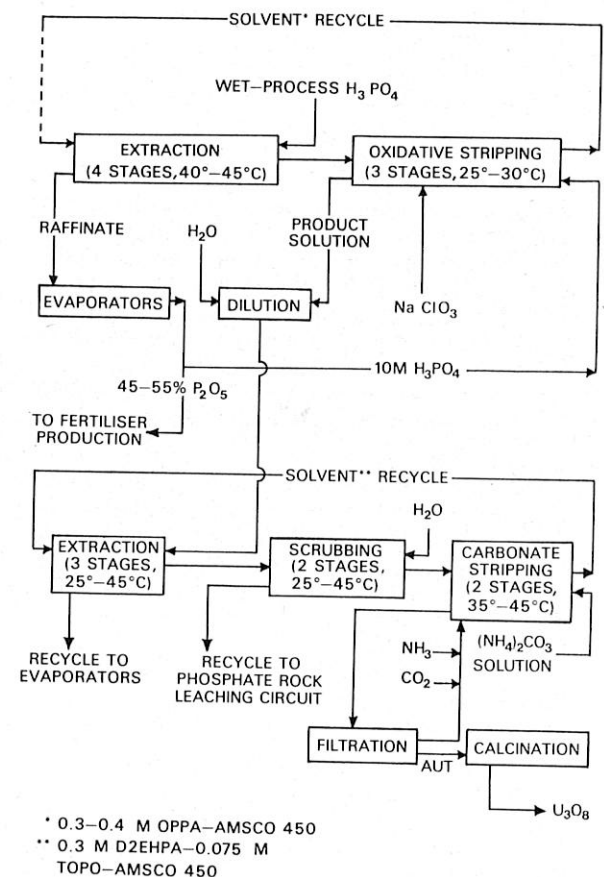
Above: Figure 1. ORNL Process (I) flowsheet for the recovery of uranium from wet-process phosphoric acid (after Hurst et al.²⁵).

Right: Figure 2. ORNL Process (II) flowsheet for the recovery of uranium from wet-process phosphoric acid (after Hurst and Crouse²⁶).

and tributyl phosphate in a kerosene diluent is a suitable and much cheaper alternative^(27, 28, 29).

This development led to investigation at ORNL⁽³⁰⁾ of an alternative first-cycle process that used a commercial mixture of mono- and dioctylphenylphosphoric acids as the extractant. This solvent which has a higher extraction coefficient for uranium than D2EHPA-TOPO, extracts uranium(IV), thus eliminating the liquor oxidation step required in process (I) (see Figure 1). The D2EHPA-TOPO solvent was still used in the second step to produce a high grade U₃O₈ concentrate.

The proposed process (process II) is shown in Figure 2. Uranium was recovered from the first-cycle solvent by contacting it with 10 M H₃PO₄ containing sodium chlorate, the strip solution was loaded with up to 15-20 g l⁻¹ of uranium. After dilution to 6 M H₃PO₄, the strip solution was highly amenable to treatment in a second cycle using the D2EHPA-TOPO solvent. Calcination of the precipitated AUT at 600°C yielded a product that contained >98% U₃O₈.



The first cycle was demonstrated continuously with the solvent subjected to about 80 complete extraction-stripping cycles. Green (made from calcined rock) and brown (typical) wet-process phosphoric acids were used during this test. Uranium recoveries were 98% and 90%, respectively, in the extraction stage and 98% or more in the stripping stage. The solvent showed no appreciable loss of extraction power with cycling and was sufficiently stable for process use. During the processing of green acid, operation was smooth and there was essentially no buildup of solids at the aqueous-organic interface. With brown acid, results were much more variable. When the feed contained dispersed solids, most of the solids accumulated at the interface and the settlers flooded rapidly. However, with filtered feed, operation was satisfactory.

Total chemical reagent costs were estimated to be less than US\$2.2/kg U_3O_8 for 95% overall uranium recovery when processing green acid containing 0.1 g l⁻¹ of uranium. The solvent loss was assumed to be 0.03% of the total acid volume. For brown acid it was concluded that more extensive continuous tests were required to determine the degree of acid clarification needed for tolerable solvent losses.

Other extraction processes

A solvent extraction process also was developed to operate in conjunction with the IMI phosphoric acid process⁽³¹⁾. The uranium recovery process was based on extracting the uranium from a recycle stream in which an effective concentration of uranium of about 300 mg l⁻¹ was obtained compared with 40-50 mg l⁻¹ in the original acidulated solution. It was suggested that uranium could be easily extracted from the recycle stream with OPPA.

A preliminary design and feasibility study for a plant to produce 40-50 tonnes/a of uranium showed that uranium could be recovered by extraction with OPPA at a cost of US\$15.4/kg U_3O_8 (September 1971)⁽³²⁾. The application of this process is limited and some doubt must exist about the economics of using OPPA as an extractant.

Recent Developments Concerning the Recovery of Uranium

The ORNL processes were modified and improved by the Gulf Research and Development Corporation who announced, in April 1974, the successful operation of a demonstration "mobile pilot" plant for uranium recovery

from wet-process phosphoric acid. This new system was claimed to be commercially profitable with a price for yellowcake in the vicinity of US\$15.4-17.6/kg U_3O_8 . The company has estimated that there is a potential in Florida for an annual yellowcake production of 3 000 tonnes for the next 50 years, based on the ability of the Gulf process to recover about 96% of the available uranium⁽³³⁾.

The demonstration unit was mounted on two enclosed truck trailers which were to be transferred to a number of Florida phosphate plants to assess the performance of the recovery process under varying conditions. The first commercial recovery plants were to be under design and construction by 1975.

Also in 1975, Uranium Recovery Corporation (URC) commenced partial operation of a commercial plant to recover uranium from a phosphoric acid plant at Bartlow, Florida. The URC process involves establishment of a network of recovery modules at various phosphate plants in central Florida, and trucking uranium bearing material from the modules to a central processing refinery at Mulberry, Florida, where uranium oxide will be recovered from the solvent. The refinery was scheduled to start up at a capacity of 900 tonnes/a of uranium concentrates⁽³⁴⁾.

Westinghouse Electric Corporation and Freeport Chemical Co. also have developed and tested solvent extraction processes on a pilot-plant scale, but have yet to reach a decision on its exploitation⁽³⁵⁾.

In addition, the USSR is reported recently to have offered aid to Morocco for a feasibility study on extracting uranium from newly discovered Moroccan phosphate deposits⁽³⁶⁾.

Appraisal of Uranium Recovery from Phosphate Rock

Recovery of uranium from a wet-process phosphoric acid stream is claimed to be a commercial proposition even at a uranium price of US\$15.4-17.6/kg U_3O_8 , which is considerably less than the projected cost for yellowcake in the 1980s.

Approximately 85% of the uranium in phosphate rock is found in wet-process phosphoric acid, and 95% of this quantity can be recovered by solvent extraction and precipitation^(17, 25, 30). The production of high analysis fertilisers (requiring the production of phosphoric acid) in proportion to the total production of phosphatic fertilisers is greater in the

TABLE 1 — MAXIMUM QUANTITY OF URANIUM RECOVERABLE FROM PHOSPHATE ROCK

Country	Yearly production of phosphate rock ^(a) , 1972 (thousands of tonnes)	Estimated uranium content ^(b) ($\mu\text{g g}^{-1}$)	Estimated quantity of uranium in rock (tonnes)	Estimated quantity of uranium recoverable from the production of phosphoric acid ^(c) (tonnes)
Israel	937	110	103	49
Jordan	714	140	100	48
Morocco	15 105	110	1 661	797
Senegal	1 250	90	113	54
South Africa	1 966	90	176	85
Togo	1 855	90	167	80
Tunisia	3 387	60	203	97
Russia	10 000	40	400	192
United States	38 465			
Florida	28 000 (70%)	150*	4 200	2 016
Other	10 465	80	840	403
			Total	7 963
				3 821

* Typical figure quoted for average uranium content of Florida phosphates.

(a) Post-beneficiation.

(b) Of beneficiated rock.

(c) 60% of the rock is assumed to be processed to phosphoric acid.

US than in most other countries. At least 60% of the rock used in that country is converted to phosphoric acid⁽³⁷⁾. Based on figures of 60% conversion to phosphoric acid and 80% overall recovery from the rock, estimates have been made of the maximum amount of uranium recoverable from phosphate for current world production figures as shown in Table 1. These results indicate that approximately 3 820 tonnes of uranium/a is potentially available.

Recent changes in the pattern of world trade in phosphatic fertilisers present a number of options for the recovery of uranium from wet-process phosphoric acid. During the 1970s, an international and intercontinental trade in phosphoric acid has developed as a result of technological progress in its transport and storage⁽¹⁾. International trade in phosphoric acid increased by approximately 50% in 1970 to 0.2 million tonnes of P_2O_5 ⁽¹⁾, and could reach 6.3 million tonnes in 1980⁽⁸⁾. Consequently, the countries exporting wet-process acid and those importing phosphate rock for

conversion to acid will have the opportunity in the future for the recovery of byproduct uranium.

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Ministerial Statement

RANGER URANIUM ENVIRONMENTAL INQUIRY — FIRST REPORT

The following statement was made by The Hon. Kevin Newman, M.P., Minister for Environment, Housing and Community Development, in the Australian Parliament's House of Representatives, Canberra, A.C.T. on Thursday, 11 November 1976.

The Government has received the First Report of the Ranger Uranium Environmental Inquiry which comprised His Honour Mr Justice Fox, Professor C. B. Kerr and Mr G. G. Kelleher as the Commission of Inquiry. The Government wishes to express its deep appreciation of the work of this Commission.

The principal findings and recommendations of the Inquiry which relate to the development and export of Australia's uranium resources have been considered by the Government and their thrust is broadly acceptable and provides a basis for future decisions on the industry. In particular the Government strongly supports the Inquiry's view on the need for the fullest and most effective safeguards on uranium exports and the strict regulation and control of uranium mining and milling.

This Environmental Inquiry has been long and exhaustive and has presented opportunities for all interested individuals and groups within the community to put their views. As the Prime Minister said on 7 November, there will shortly be a debate in the National Parliament on the First Report of the Inquiry, and this will reflect the views being expressed in the community. The Government will take decisions on the further development of the Australian uranium industry in the light of public discussion and that debate. The Second Report of the Ranger Uranium Environmental Inquiry should further illuminate particular issues.

The preparation of Environmental Impact Statements for uranium mining projects outside of the Northern Territory may now proceed but the Government will not make final decisions until the Commission of Inquiry has made its Second Report. The Environmental Impact Statements will need to take account of any physical and environmental safeguards appropriate to mining sites, set down in the Inquiry's First Report.

The Government and the previous Labor Government have both repeatedly stated that

export contracts approved prior to 2 December 1972 will be honoured. This Government had decided that no exports of uranium from Australia would be made until the First Report of the Inquiry had been received. Mary Kathleen Uranium, the only mine now producing uranium in Australia, is stockpiling production and contract obligations have been met by borrowing uranium from the United Kingdom. Other Australian firms with contractual commitments, also approved prior to 2 December 1972, Peko Mines N.L./Electrolytic Zinc Company of A'asia Ltd and Queensland Mines Ltd, will be able to meet those commitments only by making appropriate arrangements for export from the Government's stockpile.

The Government is most concerned that rigid control and safeguards be applied to uranium exports, along the lines of the recommendations of the Inquiry. It will not permit the export of uranium unless it is satisfied that there are adequate and proper safeguards on the handling, transport and processing en route and in respect of the ultimate consignee. The Government is satisfied that appropriate controls would apply to the shipments under existing contracts, which will be used for electric power generation in Japan, the United States and the Federal Republic of Germany, all of which are parties to the Nuclear Non-Proliferation Treaty. The material will be processed in the United Kingdom, Canada and the United States, which are also parties to the NPT and require safeguards on the material which they process additional to those of the NPT. Also these countries are well known to have a responsible attitude to safeguards question.

The handling and disposal of radioactive waste resulting from nuclear power generation lies with the countries concerned, and we are advised, are subject to the strictest regulation and control in these countries within existing technology.