A Review of Bigrlyi Process Development Studies

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EXECUTIVE SUMMARY

Merrill Ford was requested by the Exploration Manager of Energy Metals Australia to review the process development work done on their Bigrlyi deposit in the Northern Territory and to propose a path forward that will allow for the recovery of both uranium and vanadium.

The review of the available documents leads to the following conclusions.

- The uranium, vanadium and carbonate grades vary widely over short distances (from ppm levels to several percent). This will make selective mining difficult and dilution of the feed to the plant will be inevitable. Testwork using scrubbing with classification, gravity separation, flotation and photometric sorting has not been successful in rejecting waste material.
- The ore is relatively soft and closed-circuit ball milling can be used to achieve the fine grind required (80%<75-150 µm) for liberation of the uranium and vanadium values.
- Drill cores and their composites have been used to develop process options for the Bigrlyi deposit. A large number of laboratory acid leaches and a much smaller number of alkaline leaches have been completed. Exceptionally good uranium extractions were achieved with both approaches. However, for acid leaching the acid consumption is variable and high.
- The settling characteristics of both acid and alkaline leach residues is poor, however filtration offers a better alternative for solid-liquid separation and washing. ANSTO found that the alkaline leach residue filtered 3 to 8 time more quickly than the acid leach residue.
- No experimental work has been done on the recovery of uranium or vanadium from the leach liquors. For the acid route it has been assumed that amine solvent extraction with sodium carbonate stripping followed by sodium diuranate (SDU) precipitation, acid digestion and hydrogen peroxide precipitation of uranium tetroxide will serve the purpose. For the alkaline route the pregnant liquor, after concentration by evaporation, proceeds directly to SDU precipitation.
- Vanadium is leached with the uranium to a similar extent (30-40%) under both acid and alkali conditions and steps need to be included in the process to ensure that it does not build up in the circuit or contaminate the uranium product. However, if one prefers to recover the vanadium as a saleable product then it would be desirable to increase the vanadium dissolution in the leach. For the acid leach this requires leaching at a much lower pH and significantly increasing the acid and oxidant consumption. For the alkaline route, leach testwork will be required to confirm that at higher temperatures and pressures, high vanadium extractions are possible.

There are three pivotal decisions that need to be made before a process route can be selected.

1. The first is whether vanadium recovery as a co-product is required,
2. the second is whether acid or alkaline leaching is the most cost-effective process option, and
3. the third is whether radiometric and/or X-ray Transmission sorting is viable as a means of rejecting barren material and upgrading the plant feed.

Vanadium Recovery

Historically the vanadium price has been low when compared to the uranium price, but in recent times with the depressed uranium market and the vanadium price increasing from a long-term average of around US$6/lb to US$30/lb the potential value of these two elements in the Bigrlyi deposit are similar.

The major capital and operating costs in developing the Bigrlyi deposit lie in mining, comminution, leaching and filtration and the additional cost of recovering vanadium will be relatively small. This suggests that recovering the vanadium as a co-product will significantly improve the financial viability of the project.

It is ***recommended*** that alkaline pressure leaching tests should be undertaken in order to establish whether this approach is able to increase vanadium extraction above the 30-40% ceiling found to date. If it is, this would favour the alkaline route and further investigations will need to be undertaken to flesh out a process route for the combined recovery and sale of on-specification uranium and vanadium products.

Acid Consumption

The acid route operating cost study used an acid consumption of 65 kg/t this is well below the consumption indicated by the ANSTO leach tests (85-100 kg/t). It is also not certain whether the composites used by ANSTO for their testwork are representative of the deposit as a whole.

The main acid consuming gangue mineral is calcite. The calcite content of the samples used for leaching range from 1.5% to 13%. However, geochemical data (18) indicates that the average calcite concentration for significant parts of the Anomaly-4 deposit is over 18%. If this is correct the average acid consumption would be extremely high (over 180 kg/t) and this will adversely affect process economics for acid leaching.

It is ***recommended*** that a geometallurgical study should be undertaken to determine the likely acid consumption distribution over the entire ore body. Ideally, a mining block model with grades of the key elements, should be used as the basis for the study. Nevertheless, in conjunction with the available acid leaching data for this ore, there are numerous drill cores with downhole multi-element assays at 0.5 m intervals that could be used as a proxy.

Ore Sorting

Although radiometric sorting appears to be a technically feasible method of rejecting material below the cut-off grade with respect to uranium, it is a relatively slow sorting technology and may lose a significant fraction of the vanadium.

It is ***recommended***

a) that the radiometric data previously collected should be reanalysed to determine the vanadium deportment, and
b) X-ray Transmission sorting should be investigated as a means of upgrading the ore in terms of uranium and vanadium by a least a factor of two.

Sorting technology could be a game changer because it could either halve the size of the processing plant and reduce the capital outlay accordingly, or it could double the uranium and vanadium production rates so doubling the revenue stream (albeit for a shorter mine life) and decreasing the payback period.

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INTRODUCTION

Merrill Ford was requested by the Exploration Manager of Energy Metals Australia to review the process development work done on their Bigrlyi deposit in the Northern Territory and to propose a path forward that will allow for the recovery of both uranium and vanadium.

The Bigrlyi deposit is located approximately 350 km north-west of Alice Springs on the northern margin of the Ngalia Basin. Road access is via the Stuart Highway north from Alice Springs, then west on the Tanami Road. There are reliable road and rail connections from Alice Springs north to Darwin and south to Adelaide.

Uranium mineralisation was discovered at a number of places in the Mt Eclipse Sandstone in the early 1970's of which the most promising was that at Bigrlyi and lesser amounts at Walbiri, Malawiri and Rinkabeena. The Bigrlyi prospect consists of seven lenses which occur along a strike length of 11 km. In 2011 it was estimated to contain 28.8 Mlb U_3O_8 at a 250 ppm cut-off.

The deposit also contains significant amounts of vanadium and at current prices the vanadium could be a valuable co-product. The vanadium mineralisation in the Ngalia Basin is unlike that being exploited by other producers of vanadium in that it is present as a discrete mineral, montroseite [(V,Fe)O(OH)], together with vanadium bearing micas and clays rather than being distributed through magnetite.

Energy Metals is proposing to mine multiple deposits at Bigrlyi using both open pit and underground methods. Ore will be mined at a rate of 0.5 Mt/a and treated on site in a hydrometallurgical plant designed to produce about 1.5 Mlb/a of uranium tetroxide or yellow cake product for export via Darwin. The tailings will be filtered and ultimately deposited in a mined-out pit.

A number of studies have been undertaken over the years that cover wide range of metallurgical options for the extraction and recovery of uranium. However, due in part to the variability of the deposit in terms of grade and amount of reagent consuming gangue, no clear-cut conclusion has been reached on the processing route, and especially the choice between acid and alkaline processing.

OBJECTIVES AND SCOPE

The overall aim of this study is to undertake a review of the metallurgical work done to date and to provide recommendations on the way forward.

The specific objectives and scope are:

- to review the relevant reports provided by the client,
- to consider the options for the recovery of both uranium and vanadium,
- to provide recommendations for further testwork to help define and confirm the process, and
- to report the findings.

RELEVANT REPORTS PROVIDED

For the purposes of this study Energy metals provide access to a Dropbox containing the relevant reports. The ones used in this review are listed below:

1. Amdel Report 1210 (1978) “Ngalia Basin Uranium Investigation”
2. Central Pacific Minerals RNT405 (1989) “Vanadium Progress Note”
3. Amdel Report 07363 (1992) “Leach Tests, Bigrlyi Samples”
4. Metallurgical Project Consultants 6038 (2007) “Bigrlyi Project Scoping level Cost Estimation”, missing appendices.
5. ANSTO C1014 (2008) “Bigrlyi Uranium/Vanadium Project Pre-feasibility testwork Program”
6. ANSTO C1035 (2009) “Bigrlyi Heap Leaching Testwork”
7. Stirling Process Engineering (2009) “Concept Study Review”
8. RMDSTEM (2010) “Bigrlyi Heap Leaching Testwork (VAT leaching)”
9. ANSTO C1159 (2011) “Bigrlyi Assessment of Acid/Alkaline Leaching Phase 1”
10. ANSTO C1199 (2011) “Bigrlyi Batch Testwork – Hydrometallurgy Phase 2”
11. FLD Smidth (2011) “Investigation into the Thickening and Clarification of Bigrlyi Leach Feed and Leach Tailings Slurries”
12. Hydromet H10/231 (2011) “Summary Testwork Report for Bigrlyi Pre-feasibility Study”
13. Hydromet/Sedgeman (2011) “Pre-feasibility Study Report, Bigrlyi Uranium Project”, missing appendices
14. Energy Metals (2011) “Bigrlyi Joint Venture Pre-feasibility Study (Metallurgical Processing Section)”
15. Environmental Earth Sciences (2011) “Pre-feasibility Mine Water Supply, Bigrlyi Uranium Project”
16. AMMTEC MIN082 (2008) “QEMSCAN Analysis of Sample AN4 C/D”
17. AMMTEC MIN259 (2009) “Automated Mineralogical Analysis of Three Uranium Ore Samples”
18. Paladin Energy (2012) “Bigrlyi JV, Report on Geological Work Done First Quarter 2012”
19. Hydromet (2012) “Trade-off Study Acid Option – Operating Costs”
20. Hydromet (2012) “Trade-off Study Alkali Option – Operating Costs”
21. Hydromet (2010) “Lixiviant Choice for Leach”
22. Ultrasort 6057/2 (2009) “Bigrlyi Testwork Report”
23. AMMTEC A11855 (2010) “Bigrlyi Uranium Comminution Circuit Parameters”
24. Ashley (2009) “Mineralogy of the Bigryli deposit”

KEY FINDINGS BY PREVIOUS INVESTIGATIONS

Since 1978, when Amdel completed the first mineralogical study and investigated treatment methods for this ore many other studies have followed. It appears that the orebody is variable in its composition and selecting a preferred process option has been elusive.

As will be seen it has even been difficult to agree on the uranium and vanadium mineralogy and on the geological model, and perhaps most critically on whether to employ acid or alkaline leaching. A further complication has been the presence of vanadium and whether to produce vanadium as a by-product or not.

Commentary in this section reflects the views/conclusions of the authors of the documents listed in the previous section and the number in brackets (n) at the end of each bullet point refers to the report number in the previous section.

Mining

- Although mining of near surface material by open cut methods at Bigrlyi has many attractions and appears to be applicable in a general sense to at least half the resource, some consideration could also be given to underground mining (due to the steeply dipping nature of the mineralization and its narrow width). Underground mining may make selective mining a means of controlling dilution (2).
- Site water contained 1.3 g/L Cl and very little other dissolved solids (Na 530 mg/L, S 224 mg/L, Ca 185 mg/L, K 77 mg/L, Si 54 mg/L) and should be suitable for IX or SX operations (9).
- Environmental Earth Sciences (2011) found that there was a potential water supply in the fractured zone of Mount Eclipse Sandstone and nearby historical bores about 22 km from the mine site. Water quality was higher than surrounding areas and airlift tests indicated 6-12 L/s in these bores, although sustainability was unknown (15).

Process Scoping and Feasibility

- The 2007 scoping level cost estimation study set the project scale as 0.5 million t/a on the basis of preliminary mining studies. The target was to produce 1239 t/a U_3O_8 and 989 t/a V_2O_5 (assuming 95% and 70% recovery of uranium and vanadium respectively) (4).
- The assumed process included crushing to 12 mm followed by a ball mill in closed circuit with a cyclone, acid leaching, filtration, clarification, solvent extraction, iron precipitation (with lime) and precipitation of yellow cake (with hydrogen peroxide). Raffinate from the uranium extraction stage is oxidized to V^{5+} (with H_2O_2) then passed through a second solvent extraction plant to recover the vanadium. The vanadium in the strip liquor is precipitated as ammonium metavanadate (AMV) and then advances to a de-ammoniating kiln to produce vanadium pentoxide (4).
- The capital and operating costs in this study were estimated to be A$68.4 million and A$94.41/t_{feed} and A$16.75/lb U_3O_8 (including V_2O_5 production as U_3O_8 equivalent). Capital costs included allowances for general site establishment, administration and accommodation facilities, tailings storage facilities and surface mining infrastructure, but excluded mine development capital. Operating costs covered all processing and administration activities including product freight to the port of Darwin, and excluded mining and geology costs (4).
- Hydomet/Sedgeman (2011) were engaged to prepare a pre-feasibility study for the hydrometallurgical plant and infrastructure. Essentially the same flowsheet as costed by MPC (4) was used, but more detailed testwork was done to firm up on the design criteria for the comminution, leach and filtration sections of the plant. Vanadium recovery was not included. The capital costs were estimated to be

A$165.5 million and the operating costs to be US$25.2/lb U_3O_8 and US$79.9/t_{feed} (13).

- In 2012 Paladin took direct interest in the development of the project and it was decided to do a cost comparison of the acid and alkali flowsheets. Paladin provided an alkaline flowsheet and preliminary mass balance for the alkaline route and Metallurgical Systems prepared detailed mass balances for both options using SysCAD. Hydromet (2012) then prepared a capital and operating cost comparison for both flowsheets (19, 20):
    - Capex: Acid A$168 million v's Alkaline A$147 million;
    - Opex: Acid US$40.5 million pa v's Alkaline USUS49.5 million pa.

Mineralogy and Grade

Over the past 40 years several studies have been done on the Bigrlyi deposit. On each occasion, somewhat different assay values were determined depending on the purpose of the study. These values are summarized in the table below.

Source		**Element, %**							**Ref**
	U	**V**	**Al**	**Ca**	**Mg**	**S**	**Fe**	**CO_3**	
A2	0.170	0.095							1
A15	0.435	0.385							1
Flotation	0.415	0.53							1
Leach	0.510	0.41							1
Overall	0.29	0.42							2
Composite	1.27	0.43							2
Composite	0.746	0.56							3
Overall	0.805	0.70							4
Head	0.204	0.146							4
A4 CD	0.139	0.24	3.40	2.01	0.56	0.16	1.33	3.45	5
A4	0.152	0.120	3.1	7.3	0.44	0.27	0.98	11.5	6
A15	0.162	0.066	4.8	1.4	1.13	0.25	1.87	2.2	6
VAT	0.173	0.075	4.4	2.1	1.14	-	1.76	-	8
A4 BC	0.122	0.034	3.31	2.53	0.76	-	0.94	0.87	9
A4 CD	0.300	0.384	3.01	4.42	0.47	-	0.88	1.44	9
A15 CD	0.087	0.080	3.42	1.74	0.57	-	0.90	0.55	9
Bulk comp 1	0.197	0.225	3.17	3.24	0.55	0.06	0.81	0.95	9
Core 1	0.071	0.590	2.58	5.07	0.43	0.067	0.86	-	10
Core 2	0.162	0.35	3.19	0.89	0.40	0.070	0.75	-	10
Core 3	0.154	0.40	3.18	0.62	0.41	0.100	0.74	-	10
Core 4	0.483	0.17	5.19	1.01	1.17	0.040	1.81	-	10
Core 5	0.399	0.089	3.04	0.89	0.38	0.048	0.61	-	10
Bulk comp 2	0.269	0.31	3.61	1.54	0.62	0.069	1.00	-	10
Bulk leach	0.131	0.18	4.79	1.20	1.29	0.300	2.07	-	10

- Carnotite $[K_2(UO_2)_2(VO_4)_2.3H_2O]$ and uraninite $[UO_2]$ are the main uranium minerals and montroseite [(V,Fe)O(OH)] is the major vanadium mineral. Vanadium is associated with clays, but not with iron oxide, ilmenite or chromite (1, 2).
- Gangue minerals are dominated by quartz with phyllosilicates, orthoclase, muscovite and kaolin/chlorite. The ore contains about 5% calcite and traces of pyrite, marcasite, goethite, zircon, chromite and ilmenite (1).

- There is a strong correlation between uranium and vanadium for intersections >0.5 m. The mean mass ratio of U/V is 0.83. Many holes had intersections with grades above 1% V (2).
- The major uranium bearing minerals in Anomaly 4 C/D were uraninite (0.04%) and coffinite $[U(SiO_4)_{1-x}(OH)_{4x})]$ (0.11%) (16, 24), The major vanadium bearing minerals found by Ashley (24) are montroseite and vanadium bearing illite and clays. AMMTEC (16) identified nolanite, $[(V^{3+}, Fe^{2+}, Fe^{3+}, Ti)_{10}O_{14}(OH)_2]$, but Ashley (24) found that it was minor and not a significant source of vanadium in the deposit.
- The grain size of all mineral species was <13 µm except for quartz, orthoclase, calcite and dolomite. The vanadium is mainly associated these coarser minerals and chlorite (16). At 75 µm there is very little well liberated uranium, however it is exposed at the surface of the grain making it accessible for leaching. The vanadium minerals are less liberated than the uraninite and coffinite (17).
- Multi-element geochemical data shows that the average calcite content of the A4 orebody is approximately 9% and the average calcite content of A15 orebody is approximately 5% but calcite content is highly variable and about 34% of A4 samples are calcite-rich, averaging approximately18%. This indicates that high acid consumption may be a bigger problem than previously realized. A15 and A2 do not seem to be as bad, but there are very few samples from these anomalies. Carbonate is common at Bigrlyi and there are numerous calcite-enriched layers (18)

Pre-concentration of Ore

- After crushing (both A2 and A15) to about 6 mm there is no preferential distribution of uranium or vanadium to any particular size fraction (1). There are significant differences in mineralogy between the anomalies and there is no natural upgrading into any particular size fraction (7).
- After a finer grind to about P_{80}=530 µm there was only a mild upgrading (50%) of the uranium in the >850 um and <38 µm size fractions. With a fine grind to P_{80}=75 µm the upgrading in the <38 µm fraction was more marked (100%), but still insufficient for separation on the basis of size. Vanadium upgrading followed a similar pattern, but with greater upgrading (9).
- Scrubbing tests by AMMTEC showed that uranium and carbonate followed the mass distribution and concluded that there was nothing to gain in terms of improving the specific acid consumption by using scrubbing and classification (12).
- Gravity concentration involving jigging, heavy medium separation, or tabling are not feasible due to significant uranium losses to tailings. For samples from A2 and A15, 77% and 58% of the uranium was recovered in 3% and 5% of the mass respectively (1).
- Flotation achieved recoveries of 80% U and 70% V with rejection of more than 50% of the mass (1).
- Ultrasort (2009) were supplied with 102 rock samples, 48 high-carbonate and 54 low-carbonate, with the aim of removing the carbonate. The photometric analysis showed that there was no clear relationship between the photometric response and the carbonate content, and consequently the ore was not thought amenable to this form of sorting. However, the Ultrasort data indicated that radiometric sorting

shows good potential, recovering of 99% of the contained uranium in 54% of the mass, as well as increasing the ratio of uranium to carbonate. None of the sorting techniques showed any potential for upgrading vanadium, except perhaps where it is associated with the uranium (22).

Comminution

- Effective mineral liberation requires grinding to a particle size significantly finer than 75 µm (1).
- Stirling (2009) suggested that alternative comminution flowsheets, such as a single stage SAG mill or a more compact 2-stage crush and ball milling circuit, should be examined to reduce capital costs (7).
- AMMTEC conducted comminution tests and the following parameters were established for design purposes. CWi 10 kWh/t, RWi 10.9 kWh/t, BWi@375 µm 9.55 kWh/t, BWi@175 µm 12.0 kWh/t, Abrasion index 0.392 (23). These parameters indicate that the ore is soft and is unlikely to be sufficiently competent to be used as grinding media and a high ball charge would be required in a SAG mill. This ore can be classified in the high abrasive category and high consumption of liners and grinding media is expected (12).

Acid Leaching

- Acid leaching of A15 extracted 98-99% of the uranium and 25-27% of the vanadium (24h, ambient temperature and pressure, pH1.5, 50% solids, 51-53 kg/t acid, 10 kg/t pyrolusite, 3 g/L Fe^{3+}) (1).
- Dilute acid leach tests at pH1.0 and pH1.5 extracted 99.1% and 98.6% uranium and 78% and 59% vanadium respectively (5).
- Leaching at lower pH did not significantly increase the rate of uranium extraction, but did increase the rate of gangue reactions (5).
- A coarser grind decreased both the acid consumption and the uranium extraction, but appeared to have no impact on the vanadium extraction at pH1.8 (5).
- Pyrolusite, added as an oxidant, was responsible for about 10- 20 kg/t of the acid consumption (5).
- ANSTO (2008) concluded that the optimum leaching conditions were pH1.8, P_{80}=285 µm, 50^{o}C, 450 mV, for 12 hours with an expected acid consumption of 88 kg/t and pyrolusite requirement of 4.7 kg/t (5).
- ANSTO (2009) concluded that heap leaching was an unlikely option for Bigrlyi because of the high acid consumption and the formation of gypsum that would plug channels in the heap. The uranium extraction was poor in comparison to agitation leaches (6).
- Stirling (2009) commented that uranium dissolution is high by industry standards and further work should be aimed at ambient temperature leaching and consideration of oxidants other than pyrolusite. He also noted that prolonged leaching does not significantly improve extraction, but the acid consumption by gangue continues to increase (7).

Ref	Head, ppm		Residue, ppm		Time	Solids	Grind	Temp	pH	Fe^{3+}	Extraction		Oxid	H$_2$SO$_4$
	U	V	U	V	h	%	µm	ºC		g/L	U%	V%	kg/t	kg/t
1	5100	4100	75	3000	24	50	21%<75	Amb.	1.5	3	98	27	10*	51
1	5400	4000	75	3100	24	50	44%<75	Amb.	1.5	3	99	23	10*	53
2	12700	4300	180	3000	24	50	40%<75	Amb.	1.0	3	99	30	10*	134
2	12700	4300	640	-	24	50	40%<75	Amb.	1.5	3	96	-	10*	75
2	12700	4300	540	-	48	50	40%<75	Amb.	1.5	3	96	-	10*	86
3	7462	5601	131	1904	24	50	80%<150	Amb.	-	3	95	62	10*	150
3	7462	5601	93	1020	24	50	80%<75	Amb.	-	3	95	72	10*	150
4	2040	1460	-	-	12	40	80%<75	Amb.	-	0	95	70	0.64**	150
5	1390	2400	20	520	24	2	80%<75	40	1.0	2	99.1	78.2	0	667
5	1390	2400	20	990	24	2	80%<75	40	1.5	2	98.6	58.7	0	245
5	1390	2400	25	840	24	50	80%<75	50	1.2	5	98.1	64.9	9.5$^{\#}$	146
5	1390	2400	34	1130	24	50	80%<75	50	1.5	5	97.8	52.9	8.2$^{\#}$	120
5	1390	2400	42	1350	24	50	80%<75	50	1.8	5	97.0	43.5	9.1$^{\#}$	101
5	1390	2400	59	1430	24	50	80%<75	50	2.0	5	95.7	40.5	5.5$^{\#}$	89
5	1390	2400	42	1270	24	50	80%<75	50	1.8	5	97.1	47.0	4.7$^{\#}$	99
5	1390	2400	59	1250	24	50	80%<173	50	1.8	5	95.9	47.7	5.1$^{\#}$	96
5	1390	2400	59	1330	24	50	80%<285	50	1.8	5	95.8	44.4	5.0$^{\#}$	88
5	1390	2400	127	1320	24	50	80%<285	50	2.0	5	90.6	44.9	2.0$^{\#}$	84
6	1520	1200	661	790	1008	33	6.25 mm	Amb.	1.8	0	56.7	33.8	1.56**	155
6	1620	660	322	600	744	33	6.25 mm	Amb.	1.8	0	80.2	8.3	1.22**	55
8	1729	752	250	435	72	30	6.5 mm	Amb.	1.8	0	85.5	42.1	8.0*	54
9	1970	2250	19	1180	24	2	Pulverize	50	1.8	2	99.0	47.6	**	141
9	1970	2250	12	450	24	2	Pulverize	60	1.2	2	99.4	80.0	**	609
9	1970	2250	120	1310	24	50	80%<280	40	1.8	0	93.9	41.8	1.1$^{\#\#}$	104
9	1970	2250	249	1420	24	50	80%<530	40	1.8	0	87.4	36.9	0.8$^{\#\#}$	101
9	1970	2250	258	1490	24	70	80%<530	40	1.8	2	86.9	33.8	1.4$^{\#\#}$	88
9	1970	2250	136	1360	24	50	80%<280	40	1.8	0	93.1	39.6	2.4$^{\#}$	105
9	1970	2250	210	1410	24	70	80%<530	40	1.8	2	89.9	37.3	1.9$^{\#}$	92
9	1970	2250	114	1270	24	50	80%<280	40	1.8	0	94.2	43.6	2.7$^{\#\#}$	104
9	1970	2250	284	1370	24	70	80%<530	40	1.8	1	85.6	39.1	1.1$^{\#\#}$	89
9	1970	2250	60	1310	24	50	80%<150	40	1.8	0	96.9	41.8	1.45$^{\#\#}$	103
10	2690	3100	105	2430	24	50	80%<150	35	**2 g/L**	0	96.1	21.6	3.5$^{\#}$	59.3
10	2690	3100	62	1890	24	50	80%<150	35	**5 g/L**	0	97.7	39.0	3.4$^{\#}$	70.0
10	2690	3100	57	1950	24	50	80%<150	35	**10 g/L**	0	97.9	37.1	3.7$^{\#}$	81.2
10	2690	3100	174	2200	24	50	80%<150	35	2 g/L	0	93.5	29.0	**2.7$^{\#}$**	58.3
10	2690	3100	108	2220	24	50	80%<150	35	2 g/L	0	96.0	28.4	**3.4$^{\#}$**	58.3
10	2690	3100	78	2210	24	50	80%<150	35	2 g/L	0	97.1	28.7	**5.2#**	59.8
10	2690	3100	335	2230	24	50	**80%<450**	35	2 g/L	0	87.5	28.1	3.2$^{\#}$	57.6
10	2690	3100	280	2390	24	50	**80%<300**	35	2 g/L	0	89.5	22.9	3.2$^{\#}$	58.3
10	2690	3100	92	2280	24	50	**80%<106**	35	2 g/L	0	96.6	26.5	3.4$^{\#}$	59.8
10	2690	3100	210	2530	24	50	80%<150	**20**	2 g/L	0	92.1	18.4	2.6$^{\#}$	53.4
10	2690	3100	118	2370	24	50	80%<150	**30**	2 g/L	0	95.6	23.5	3.1$^{\#}$	56.3
10	2690	3100	66	2170	24	50	80%<150	**45**	2 g/L	0	97.5	30.0	3.8$^{\#}$	62.8
10	2690	3100	115	2380	24	**40**	80%<150	35	2 g/L	0	95.7	23.2	3.5$^{\#}$	58.6
10	2690	3100	89	2350	24	**60**	80%<150	35	2 g/L	0	96.7	24.2	3.6$^{\#}$	56.6
10	2690	3100	40	2570	24	50	80%<150	35	2 g/L	2.8	98.5	17.1	3.4$^{\#}$	52.8
10	1310	1800	40	1440	12	**60**	80%<150	35	2 g/L	2.8	97.0	20.0	12.5$^{\#}$	63.6
10	1310	1800	38	1470	12	50	80%<150	35	2 g/L	2.8	97.1	18.3	12.6$^{\#}$	62.9
10	1310	1800	34	1460	12	**60**	80%<150	35	2 g/L	2.8	97.4	18.9	11.8$^{\#}$	60.5
10	1310	1800	22	1410	12	50	**80%<106**	35	2 g/L	2.8	98.3	21.7	9.4$^{\#}$	57.2
10	710	5900	41	3620	24	50	80%<150	35	2 g/L	0	93.9	38.2	5.1$^{\#}$	145.8
10	1620	3500	148	2940	24	50	80%<150	35	2 g/L	0	90.5	16.7	3.3$^{\#}$	34.3
10	1540	4000	150	3030	24	50	80%<150	35	2 g/L	0	89.9	24.3	3.0$^{\#}$	28.2
10	4830	1700	94	1480	24	50	80%<150	35	2 g/L	0	98.0	14.5	4.3$^{\#}$	50.5
10	3990	890	68	750	24	50	80%<150	35	2 g/L	0	98.3	15.7	2.5$^{\#}$	35.9

*MnO$_2$, ** H$_2$O$_2$, $^{\#}$Pyrolusite, $^{\#\#}$NaMnO$_4$,

- RMDStem (2010) conducted preliminary Continuous Vat Leach (CVL) tests by splitting the crushed (to <6.5 mm) material into four different size fractions, curing for 24 h at 70°C with 8 kg/t MnO_2 and 53.6 kg/t H_2SO_4 and then bottle rolling at 30% solids for 48 h. Over 80% uranium dissolution was achieved across all size fractions, but vanadium dissolution was lower, rising from 26% in the coarsest fraction to 66% in the finest fraction (8).
- Another series of acid and alkaline leach tests were conducted by ANSTO (2011). Diagnostic tests at 2% solids showed that 99% of the uranium is readily soluble and that 48% of the vanadium dissolves at pH1.8 and 50°C. Under stronger acid conditions (pH1.2, 60°C) the vanadium extraction increased to 80%. Alkaline leaching (at atmospheric pressure and 90°C) achieved 96% uranium extraction, but only 25% of the vanadium dissolved without oxidant, and this increased to 38% with oxidant (9).
- For the conventional acid leaches, acid addition was consistently around 90 kg/t and 105 kg/t, for 50% and 70% solids respectively. Uranium extraction increased from 86% to 94% and to 97% as the grind size was decreased from P_{80} of 530 µm to 280 µm and finally to 150 µm. ORP and ferric addition appeared to only have a minor impact on uranium extraction (9).
- Optimisation acid leaches were undertaken by ANSTO (2011) to investigate the impact of grind, leaching temperature, slurry density, ORP and make-up water source (10). They concluded:-
    - that a low free acid (2-3 g/L) was adequate for good uranium extraction and slower acid consumption;
    - that high temperatures were not required to achieve satisfactory uranium extractions and elevated temperatures raised acid consumption;
    - that grind size had a significant detrimental effect on uranium extraction, a P_{80} of 150 µm was found to be optimum;
    - that leaching in raffinate, which contained about 3 g/L Fe, improved uranium extraction by 2% due to the ferric concentration being increased from 1 g/L to 3 g/L, and reduced acid consumption by 6 kg/t due to the higher TDS repressing gangue dissolution; and
    - that the rate of leaching was improved by increasing the slurry density from 50% to 60% due to increased ferric concentrations.
- A “bulk leach” composite was made up with core diluted by barren material to better match the expected plant feed material. This material was leached to confirm the optimised conditions (12 h, 2 g/L acid, P_{80}=106 and 150 µm, raffinate, >3 g/L Fe^{3+}, 35°C, 50% and 60% solids). Uranium extractions over 97% were achieved with acid additions in the range 57-64 kg/t. The pyrolusite consumptions were very high and ranged from 9.4 kg/t to 12.6 kg/t. Vanadium extractions were low and ranged from 17% to 21% (10).
- Variability leach tests were done on five different core composites in bore hole water (no extra iron). Uranium extractions ranged from 89.9% to 98.3% and vanadium extractions ranged from 15.7% to 38.2% after 24 hours. Acid and pyrolusite additions ranged from 28-146 kg/t and 2.5-5.1 kg/t respectively (10).

Alkaline Leaching

- Alkaline leaching under the mild conditions tested was less effective than acid leaching for both uranium and vanadium extraction (5).
- For the ANSTO (2011) tests the maximum uranium extraction of 97.5% was obtained for the test with high oxidant addition. Under the range of conditions applied during this series of tests the vanadium dissolution was inconsistent and appeared to be unaffected by change of oxidant addition or grind (9).

Ref	Head, ppm		Residue, %		Time	Solids	Grind	Temp	Na$_2$CO$_3$	NaHCO$_3$	Extraction		Oxid
	U	V	U	V	h	%	µm	ºC	g/L	g/L	U%	V%	kg/t
2	12700	4300	2750	3800	24	50	40%<75	Amb.	50	20	79	12	10*
5	1390	2400	76	1850	48	30	80%<75	95	20	10	94.7	22.9	?**
5	1390	2400	119	1710	48	30	80%<75	95	15	5	91.6	28.7	?**
9	1970	2250	83	1690	48	2	Pulverize	90	40	10	95.8	24.9	-
9	1970	2250	68	1400	48	2	Pulverize	90	40	10	96.5	37.8	#
9	1970	2250	48	1490	48	50	80%<75	90	40	10	97.5	33.7	6.1$^{\#}$
9	1970	2250	179	1560	48	50	80%<75	90	40	10	90.9	30.7	2.4$^{\#}$
9	1970	2250	199	1460	48	50	80%<150	90	40	10	89.9	35.1	2.4$^{\#}$
9	1970	2250	108	1470	48	50	80%<75	90	40	10	94.5	34.7	2.2$^{\#}$
9	1970	2250	120	1450	48	50	80%<150	90	40	10	93.9	35.6	2.2$^{\#}$

*KMnO$_4$, **Air, $^{\#}$H$_2$O$_2$,

Solid/Liquid Separation

- Settling tests done at ANSTO (2008) indicated that the mass flux rates for all leach conditions were relatively low and would be about 0.1 m^2/tpd for neutral feeds, 1.1 m^2/tpd for acid and 3.0 m^2/tpd for carbonate slurries (5).
- Stirling (2009) commented that the ANSTO settling test results are very poor when compared to other laterite and calcrete deposits. He thought that this data should be reviewed because it would have a major impact on capital costs. He suggested that the poor settling could be a consequence of silica polymerization and gelling and that the silica could negatively impact phase disengagement in SX (7).
- Settling tests done at ANSTO (2011) produced rather different results to those done in 2008. On this occasion the alkaline slurries performed similarly or perhaps better than the acid slurries despite the much finer particle size (75 µm v's 280 µm). The best results for the acid and alkali settling tests were 0.19 m^2/tpd and 0.14 m^2/tpd respectively (9).
- Filtration tests on the alkaline leach residue filtered and washed about 3 to 8 times more quickly than the acid leach residue despite the finer particle size. The slowness of the filtration of the acid residue was attributed to gypsum formation causing cloth blinding. The filter cake contained about 15% moisture and filtration rates (form plus wash) varied from 0.23-0.36 (kg/h)/m^2 and 0.79-1.47 (kg/h)/m^2 for the acid and alkaline residues respectively (9).
- Rheology tests conducted by ANSTO indicated that yield stresses were low up to 60% solids and would not pose significant issues for pumping (10).
- Settling tests conducted by FLD Smidth (2011) indicated that the feed slurry could be settled to 60-62% solids and that the acid leach residue could be settled to 58-60% solids at a flux rate of 0.53 tph/m^2. For both slurries, they estimated that 14 m diameter high rate thickeners would be able to settle 81 t dry solids per hour. A

similar size clarifier with a rise rate of 1 m/h was expected to produce an overflow containing 100-150 mg/L suspended solids (11).

- Bernard Guyot Filtration conducted belt filtration tests on the "bulk" acid leach residue slurry at ANSTO. A polyester filter cloth (PES214-2) and guar flocculant (RMG20) was selected for the trials. The flocculant dose was 180-210 g/t. The leach discharge slurry was readily flocculated, filtered and washed to displace soluble uranium and an overall flux rate of 1100 (kg/h)/m^2 was determined (12).

Solvent Extraction

- Stirling (2009) endorsed the 2-stage sequential amine SX process for uranium and vanadium recovery and favoured ammonium sulphate/ammonia stripping. However, he felt that the target uranium loading on the solvent (5% Alamine 336) should be 4.0 g/L U_3O_8 as opposed to 2.1 g/L U_3O_8 used for the scoping study. He also suggested the following operating phase ratios and design criteria (7).

Section	Operating O/A	Mixer time minutes	Specific Throughput Aqueous (m^3/h)/m^2
Extraction	1:1	0.5	2.3
Scrub	3:1	4.0	1.0
Strip	2:1	4.0	1.4
Regeneration	2:1	4.0	1.0

Vanadium Recovery

- The major cost of producing vanadium from magnetite lies in the high temperature processing required to liberate the vanadium and the subsequent chemical processing. This type of technology was abandoned by the Colorado Plateau producers in favour of low temperature acid dissolution and solvent extraction of the vanadium (2). Salt roasting is probably only economic at vanadium grades greater than 1% V (7).
- Central Pacific Minerals (1989) thought that heap leaching might provide the long residence times that may facilitate vanadium leaching (2).
- CPM (1989) also considered that although the vanadium grade was deemed to be economic elsewhere it may not sustain mining in its own right. However, they thought that Bigrlyi had some natural advantages; it is not located in a national park, it is not near watercourses, the local population is (was?) in favour of mining and uranium production would cover many of the costs (2).
- Roasting the ore (P_{80}=150 µm) at 830^{o}C for 2 hours with 10% NaCl followed by a water leach for 4 hours and then and acid or alkali leaching for 20 hours only leached 61% or 39% of the vanadium respectively. The uranium extraction was even worse (36% and 4% respectively) (3).
- Stirling (2009) noted that the results of the salt roasting trials were surprisingly poor and further tests should be done to completely rule out this option. They also suggested that an economic trade-off between increased vanadium extraction in the leach and acid consumption should be analyzed (7).
- Chemical precipitation has been used in the past to recover uranium and vanadium directly from leach liquors. Acid liquors were neutralized to pH 6.0 with caustic or ammonia and the precipitate redissolved in sulphuric acid. The pH was again

adjusted to pH 2.5 to precipitate iron vanadate and the uranium recovered from the liquor. Alkaline liquors were neutralized with sulphuric acid to pH 6.0 and the solution boiled to drive off CO_2 and precipitate uranyl-vanadate and red cake for further processing (7).

- Stirling summarized the uranium/vanadium flowsheets applied in the USA (presented below).

Owner	Plant	Overall Process	Vanadium Product
Union Carbide	Uravan	Early process used salt roast and recovered vanadium as ferric vanadate. Plant later converted to acid leach. IX was used to separate uranium from vanadium.	Crude precipitate sent to Rifle plant
Union Carbide	Rifle	Complex circuit. High vandium ore treated by salt roast followed by water leaching. Vanadium from this process extracted by tertiary amine after pH adjustmeent to 3.0. Stripped with soda ash and recovered as AMV. AMV then treated for oxide recovery. Low vanadium ore acid leached together with residue from HG water leach. Uranium recovered using D2EHPA SX. Vanadium recovered from raffinate by ammonia precipitation. Precipitate recycled to salt roast.	V_2O_5
Vanadium Corporation of America	Naturita	Salt roast followed by sodium carbonate quench and leach. The uranium/vanadium carbonate leachant precipitated as synthetic carnotite that was reduced by fusion to produce oxide. Excess (to carnotite) solution precipitated as red cake.	V_2O_5 and red cake
Vanadium Corporation of America	Durango	Salt roast followed by sodium carbonate quench and leach. The uranium/vanadium carbonate leachant precipitated as synthetic carnotite that was reduced by fusion. The product was then water leached and the final precipitate was red cake. The residue from nthe initial soda ash leach was leached in acid and then both vanadium abd uranium extracte dby SX using a mixed D2EHPA/amine solvent. A crude mixed U/V precipitate was then generated.	Red cake and a crude U/V precipitate
Climax Uranium	Grand Junction	Salt roast followed by water leach. Vanadium from the water leach precipitated as red cake. This was then fused and the final product was vanadium pentoxide (88% V_2O_5). Residue from water leach acid leached. Leachate reduced with Fe to -440mV and pH 1.0. Liquor then treated via D2EHPA SX to recover uranium. Raffinate containing vanadium neutralised and returned to the salt roast.	V_2O_5
Foote Minerals	Navajo	Ore acid leached to extract both vanadium and uranium. Feed reduced to -200mV and pH <1.8. Uranium extracted using D2EHPA in 4 stages. Vanadium bearing raffijnate pH adjusted to 1.8 to 2.2 with ammonia and then extracted with D2EHPA. Loaded solvent stripped with sulphuric acid . Strip liquor precipitated with NH_3 /$NaClO_3$ to produce red cake. Final product vanadium pentoxide.	V_2O_5
Mines Development	Edgemont	Ore acid leached. Uranium extracted using RIP at pH 1.8 and -380mV. RIP tails separated using thickeners to recover vanadium solution. Solution reduced with Fe (-200 to -250mV), pH adjusted to 1.9. Vanadium extracted in 4 stages with D2EHPA. Loaded solvent stripped \with sulphuric acid. Strip liquor precipitated with NH3 /NaClO3 to produce red cake. Final product vanadium pentoxide.	V_2O_5
Atlas Corporation	Moab	Ore acid leached. Feed prepared at pH 1.0 and -150mV. Uranium extracted using Tertiary Amine. Extracted uranium sent to final recovery along with alkaline leach product from low V carboante leach.Raffinate from Amine SX adjusted to pH 2.0, -150mV and extracted with D2EHPA in 4 stages. Loaded solvent stripped with 15% sulphuric acid. Strip liquor precipitated with NH3 /NaClO3 to produce red cake. Final product vanadium pentoxide.	V_2O_5
White Mesa	White Mesa	Ore acid leached with hot sulphuric acid and sodium chlorate as oxidant for 24 h. Uranium extracted with 2.5% tertiary amine (2.5% isodecanol and kerosene). Raffinate oxidised and vanadium extracted with amine. Loaded organic stripped with soda ash. AMV precipitated by adding ammonium silphate. The AMV precipitate is filtered, dried and fused to vanadium pentoxide.	V_2O_5

COMMENTARY ON PREVIOUS INVESTIGATIONS

Mineralogy and Grade

Each of the anomalies sampled have very different compositions in terms of uranium grade and gangue composition. To further complicate matters a mining plan has not been developed and consequently the amount of mine dilution is unknown. Nevertheless, the deposit is relatively high grade and contains slightly more vanadium than uranium.

Initially the uranium mineralization was thought to be a mixture of uraninite and carnotite, but subsequent examinations have concluded that there is very little carnotite and instead around 75% of the uranium occurs as coffinite. (Carnotite is present in the weathered zone at Bigrlyi (to a depth of about 20m in A14 & A15) and also in A2 which is deeply weathered to 100m.

The vanadium mineralization has also been found to be predominantly montroseite and vanadium bearing micas and clays. Both the uranium and vanadium minerals are fine grained (<10 µm) and the majority of vanadium appears to be associated with clays.

The main acid consuming gangue mineral is calcite. The calcite content of the samples used for leaching range from1.5% to 13%. However, geochemical data indicates that the average calcite concentration in a substantial part of the deposit is over 18%. This will have a significant impact on the viability of the acid route.

Pre-concentration of Ore

Scrubbing with size classification, flotation, gravity concentration and photometric sorting have all been tested as means of upgrading the ore and/or rejecting carbonaceous material. Unfortunately, none of these techniques have shown any promise.

Preliminary radiometric sorting results are encouraging and indicate that over 40% of the ore could be rejected as waste with 99% uranium retention in addition to improving the uranium to carbonate ratio in the concentrate. Vanadium deportment is not available, but could be an important consideration.

Radiometric sorting is slow (due to the gamma counting time required to estimate the grade of each rock particle) when compared to photometric and electromagnetic measurements. X-ray Transmission analysis is a new and rapid technique for measuring the heavy metal content of rocks for sorting applications and could have the additional benefit of being able to take both the uranium and vanadium contents into account.

Leaching

The uranium in the ore leaches exceptionally well (>95% dissolution) under both mild acid and alkaline conditions. However, the vanadium is more refractory and only 20-40% dissolves under these conditions. Finer grinding, higher temperatures, higher acid and oxidant concentrations and longer leaching times all improve the leaching of both uranium and vanadium, but usually this is at the expense of higher acid and oxidant consumptions.

Parameter	**Units**	**June 2008 (5)**		**January 2011 (9)**		**May 2011 (10)**	
		U	**V**	**U**	**V**	**U**	**V**
Head grade	ppm	1390	2400	1970	2250	1310	1800
Residue grade	ppm	59	1330	114	1270	34	1460
Dissolution	%	95.8	44.4	94.2	43.6	97.4	18.9
Slurry density	% solids	50		50		60	
Grind P_{80}	µm	285		280		150	
Temperature	^{0}C	50		40		35	
pH	-	1.8		1.8		1.9	
Reagent	-	H_2SO_4		H_2SO_4		H_2SO_4	
Concentration	g/L	2.5		3.0		2.0	
Leach time	h	12		24		12	
ORP	mV	450		550		550	
Oxidant	Type	Pyrolusite		$KMnO_4$		Pyrolusite	
Fe^{3+}	g/L	5		0		2.8	
Oxidant addition	kg/t	5.0		2.7		11.8	
Acid addition	kg/t	90		104		61	

Both heap and vat acid leaching were attempted, but uranium and vanadium extractions were poor compared to agitated tank leaching.

ANSTO (5, 9,10) conducted three campaigns to optimise the acid leaching parameters. Their considered optimum conditions deduced from the campaigns are summarised above. The difference in acid consumption is largely due to the different calcite contents of the initial composite used for the test (5%, 8% and 3% respectively).

Far fewer alkaline leach tests have been conducted because the direction of thinking was that acid leaching would be the preferred option. From the data it is clear that for alkaline leaching a finer grind is required, but uranium and vanadium extractions that are similar to acid leaching can be achieved.

Parameter	**Units**	**June 2008 (5)**		**January 2011 (9)**		**January 2011 (9)**	
		U	**V**	**U**	**V**	**U**	**V**
Head grade	ppm	1390	2400	1970	2250	1970	2250
Residue grade	ppm	76	1850	48	1490	120	1450
Dissolution	%	94.7	22.9	97.5	33.7	93.9	35.6
Slurry density	% solids	30		50		50	
Grind P_{80}	µm	75		75		150	
Temperature	^0C	95		90		90	
pH	-	9.2		9.2		9.7	
Reagent	-	$Na_2CO_3/NaHCO_3$		$Na_2CO_3/NaHCO_3$		$Na_2CO_3/NaHCO_3$	
Concentration	g/L	20/10		40/10		40/10	
Leach time	h	48		48		48	
Oxidant	Type	Air		H_2O_2		H_2O_2	
Oxidant addition	kg/t	5.0		6.1		2.2	

In carnotite, the uranium is in the hexavalent state and is readily soluble under mild alkaline leaching conditions. Elevated temperature is used to increase the rate of reaction, but it can be done at 95°C in a cascade of atmospheric pressure leaching tanks. However, coffinite and uraninite (as occurs in the Bigrlyi ore) contain uranium in the tetravalent oxidation state, therefore stronger oxidizing conditions are required for good uranium dissolution. Historically alkaline leaching of uranium ores has been done at 120-140°C in pressure vessels using oxygen as the oxidant and it is likely that this form of leaching will be applicable to the Bigrlyi ore.

Vanadium dissolution is generally poor and it is only under strong acid and oxidizing conditions that extractions around 80% were obtained. This is consistent with the fact that vanadium(V) is fairly insoluble in the pH range 1.5-2.5 under oxidizing conditions. In addition, in the presence of ferrous/ferric ions, insoluble iron-vanadate is forms and precipitates. This indicates that under acid leaching conditions that are favourable for uranium dissolution, vanadium extraction will be problematic.

Solid-Liquid Separation

Settling tests done by FLD Smidth confirmed that both the acid and alkaline leach residue slurries settled at similar slow rates (0.53 tph/m^2), despite the finer size of the alkaline residue. The final underflow density was about 60% solids and the yield stresses were low, making pumping easy.

Filtration showed more promise as a method of solid-liquid separation and washing. ANSTOs tests indicated that the alkaline slurry filtered up to four times more rapidly than the acid slurry, and filtration tests done by Bernard Guyot on the acid slurry provided an overall form, wash and dry rate of 1.1 tph/m^2 to produce a filter cake washed with 99.5% efficiency and with 20% residual moisture.

Vanadium Recovery

The poor performance of the tried and tested salt roast process to rend the vanadium water soluble is surprising. The fact that roasting made the uranium more refractory indicates that the test conditions may not have been properly controlled. Nevertheless, this approach was abandoned by the vanadium producers on the Colorado Plateau in favour of a hydrometallurgical approach and may not be worth pursuing for Bigrlyi.

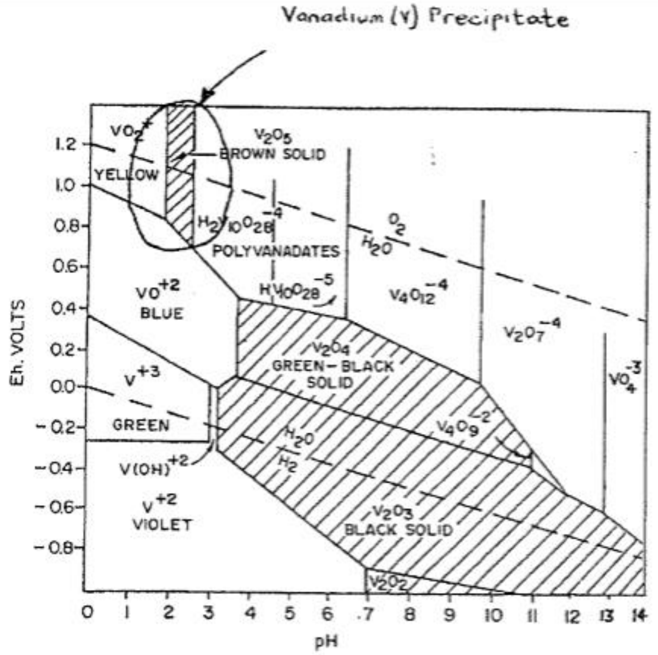

Figure 1. Stability Regions of Vanadium Ions and Solid Oxides (At 25°C. and 10^{-4}M.[V]. Adapted from Ref.[4].)

As commented above and shown in the stability diagram, V_2O_5 has limited solubility under the optimized acid leach conditions identified by ANSTO.

From the above diagram it can be seen that vanadium (III) (as occurs in montroseite and nolanite) should be readily oxidized by oxygen/air to vanadium(V) at the higher pH levels associated with carbonate leaching (pH 9.5 to 10.5). However, only 30-40% vanadium dissolution was achieved by the alkaline leach tests to date, and it seems that leaching at 90^{o}C with air at atmospheric pressure as the oxidant is insufficient for good vanadium extraction.

Process Feasibility Study

In 2011 Hydromet Pty Ltd was commissioned to conduct a pre-feasibility study on the Bigrlyi prospect by Energy Metals (14). Based on their judgement at the time it was decided that the acid route should be used for the study, however, the study seemed to be based on optimistic assumptions one of which was that the acid consumption would be

65 kg/t when the ANSTO data indicated 85 kg/t, with one sample consuming 300 kg/t another was that the uranium price was US$80/lb.

Even under these assumptions the project appeared to be marginal and Paladin (a 47% shareholder at the time) asked for a comparative study using the alkaline processing route be undertaken before further expenditure on a proposal to collect samples (5 t) and run mini-plant trials using the acid process. This request was agreed to, and based on an alkaline flowsheet supplied by Paladin, the capital and operating costs were estimated for both options.

Unfortunately, even after this study the choice between the two routes was not clear-cut, because although the alkaline route appeared to have a lower capital cost, it had a higher operating cost than the acid route. In addition, possibly due to limited data availability for the alkali route, several conservative assumptions were incorporated into the alkaline study (20), while the acid route assumptions (19) tended to err on the optimistic side.

According to Hydromet (20) the capital cost of the alkali route is $147 million compared to the acid route cost of $168 million, a saving of $21 million (13%). They estimated the operating cost of the alkali route as $45.9 million pa compared to the acid route cost of $40.5 million pa, an increase of $5.4 million pa (13%). In addition, the acid route was expected to have an overall uranium recovery of 96.1% and the alkaline route 88.8% (contradicting the leach extractions presented above).

Other possible savings to the Hydromet (2012) alkaline study that may be applicable are listed below. Although they may not all be achievable in practice, they are points that should be considered when comparing the acid and alkaline process routes.

- Recovery of vanadium was incorporated into the alkaline circuit, but not the acid. This may not be essential and in addition to a small saving in capex it will reduce the NaOH requirement by $2.1 million pa. Alternatively, if vanadium is recovered then it could be sold as a byproduct and perhaps offset the lower uranium recovery.
- Heating of the PLS to 70ºC to ensure good uranium precipitation (SDU) may not be essential, this could reduce steam demand and costs by $3.0 million pa. In addition, if HFO (or natural gas) was used instead of diesel for this purpose another $0.7 million pa could be saved.
- The raw water demand is estimated to be about 350 kt/a, rather than the 1050 kt/a used in the study. Apart from saving $0.34 million pa this will simplify water supply logistics.
- The demineralised water demand is estimated to be 79 kt/a rather than 131 kt/a used in the study. Apart from saving $0.24 million pa this will substantially reduce the cost of the demineralization plant.
- $20 million has been allowed to build evaporation ponds. However, if less water is required for the process then the size of the evaporation ponds can be reduced. In addition, forced evaporation could be another way to reduce the size of the ponds and hence the cost (as well as making the process less dependent on nature).
- It is not clear why the labour costs for the alkali route are $1.0 million pa higher than that for the acid route or why the administration costs are $0.3 million pa

higher or why the light vehicles cost is $0.3 million higher. One would expect these items to be similar for both options.

- Alkali reagents are far less aggressive than sulphuric acid on materials of construction therefore there should be savings in spare parts and in engineering maintenance labour for the alkaline route.

Other points to consider include the following.

- Uranium locked in calcareous particles is not accessible to the alkali reagent and therefore finer grinding is required than for the acid route. Despite the finer grind some uranium will remain unliberated in calcareous minerals and will not be extracted by the carbonate reagent. This accounts for a slightly lower uranium extraction in the leach, perhaps 1% to 2%, but not 7%.
- It appears that very poor filter washing has been assumed for the alkali route and in my opinion, this is not justified. Testwork is required to confirm this or not, but even if the filtration characteristics are as poor as implied by the study then additional washing of the filter cake should be incorporated to mitigate this problem.
- Due to the selective nature of the alkaline leach very little apart from uranium and vanadium is dissolved. The only chemicals required for the process are sodium hydroxide, fuel (CO_2) and hydrogen peroxide (coverts to water and oxygen), all of which are in abundance in the natural environment around the mine. On the other hand, the acid route demands sulphuric acid, manganese dioxide and organic solvents, which are foreign compounds that are potential environmental hazards unless properly handled.
- The acid study does not appear to make any provision for the purchase of lime or for slaking it. If the assumption is that the limestone in the area will be used to neutralize the tailings this may not be adequate because the pH has to be raised above 8.5 to precipitate the manganese. The cost of lime could be over $2 million pa.
- The acid study estimates an acid demand of 65 kg/t at a cost of $176/t ($5.2 million pa). It is likely that the average acid consumption will be closer to 85 kg/t and the acid cost for this remote location closer to $250/t. If this is correct then the cost of the acid route will be $4.3 million pa more than indicated.
- For the acid route the mine has to be careful to avoid high calcareous parts of the deposit, especially near the edges of the pit envelope where the uranium grades are low. This type of mining is expensive compared to that that required for the alkaline route where only the uranium grade is important. In general, the area around the mine is calcareous and any other discoveries could be high acid consuming and uneconomic to treat with an acid process, but the alkaline route would be more flexible in this regard.

CONCLUSIONS

The uranium, vanadium and carbonate grades vary widely over short distances (from ppm levels to several percent). This will make selective mining difficult and dilution of the feed to the plant will be inevitable. Testwork using scrubbing with classification, gravity separation, flotation and photometric sorting has not been successful in rejecting waste material.

The ore is relatively soft and closed-circuit ball milling can be used to achieve the fine grind required (80%<75 µm) for liberation of the uranium and vanadium values.

Drill cores and their composites have been used to develop process options for the Bigrlyi deposit. A large number of laboratory acid leaches and a much smaller number of alkaline leaches have been completed. Exceptionally good uranium extractions were achieved with both approaches. However, for acid leaching the acid consumption is variable and high.

The settling characteristics of both acid and alkaline leach residues is poor, however filtration offers a better alternative for solid-liquid separation and washing. ANSTO found that the alkaline leach residue filtered 3 to 8 time more quickly than the acid leach residue, but this should be confirmed.

No experimental work has been done on the recovery of uranium or vanadium from the leach liquors. For the acid route it has been assumed that amine solvent extraction with sodium carbonate stripping followed by sodium diuranate (SDU) precipitation, acid digestion and hydrogen peroxide precipitation of uranium tetroxide will serve the purpose. For the alkaline route the pregnant liquor proceeds, after concentration by evaporation, directly to SDU precipitation.

Vanadium is leached with the uranium to a similar extent (30-40%) under both acid and alkali conditions and steps need to be included in the process to ensure that it does not build up in the circuit or contaminate the uranium product. However, if one prefers to recover the vanadium as a saleable product then it would be desirable to increase the vanadium dissolution in the leach. For the acid leach this requires leaching at a much lower pH and significantly increasing the acid and oxidant consumption. For the alkaline route, leach testwork will be required to confirm that at higher temperatures and pressures high vanadium extractions are possible.

RECOMMENDATIONS ON THE WAY FORWARD

There are three pivotal decisions that need to be made before a process route can be selected.

1. The first is whether vanadium recovery as a co-product is required,
2. the second is whether acid or alkaline leaching is the most cost-effective process option, and
3. the third is whether radiometric and/or X-ray Transmission sorting is viable as a means of rejecting barren material and upgrading the plant feed.

Vanadium Recovery

On average, the vanadium grade of the deposit appears to be higher (about 20%) than that of the uranium. Historically the vanadium price was very low compared to uranium, but in recent times with the depressed uranium market and the vanadium price increasing from a long-term average of around US$6/lb to US$30/lb the potential value of these two elements in the Bigrlyi deposit are similar.

The major capital and operating costs in developing the Bigrlyi deposit lie in mining, comminution, leaching and filtration and the additional cost of recovering vanadium will be relatively small. This suggests that recovering vanadium as a co-product will significantly improve the financial viability of the project.

It is ***recommended*** that alkaline pressure leaching tests should be undertaken in order to establish whether this approach is able to increase vanadium extraction above the 30-40% ceiling found to date. If it is, this would favour the alkaline route and further investigations will need to be undertaken to flesh out a process route for the combined recovery and sale of on specification uranium and vanadium products.

Acid Consumption

The acid route operating cost study used an acid consumption of 65 kg/t this is well below the consumption indicated by the ANSTO leach tests (85-100 kg/t). It is also not certain whether the composites used by ANSTO for their testwork are representative of the deposit as a whole.

The main acid consuming gangue mineral is calcite. The calcite content of the samples used for leaching range from1.5% to 13%. However, geochemical data (18) indicates that the average calcite concentration in the deposit is over 18%. If this is correct the average acid consumption would be over 180 kg/t and this will rule out acid leaching as an option.

It is ***recommended*** that a geometallurgical study should be undertaken to determine the likely acid consumption distribution over the entire ore body with grades above selected cut-offs.

Ideally, a mining block model with grades of all the key elements, should be used as the basis for the study. Nevertheless, in conjunction with the available acid leaching data for this ore, there are numerous drill cores with downhole multi-element assays done at 0.5 m intervals that could be used as a proxy.

Ore Sorting

Although radiometric sorting appears to be a technically feasible method of rejecting material below the cut-off grade with respect to uranium, it is a relatively slow sorting technology and may lose a significant fraction of the vanadium. X-ray sorting has been used successfully for diamond recovery and has potential for rapidly identifying heavy minerals in the rock particles.

It is ***recommended***

a) that the radiometric data previously collected should be reanalysed to determine the vanadium deportment, and

b) X-ray sorting should be investigated as a means of upgrading the ore in terms of uranium and vanadium by a least a factor of two.

Sorting technology could be a game changer because it could either halve the size of the processing plant and reduce the capital outlay accordingly, or it could double the uranium and vanadium production rates so doubling the revenue stream (albeit for a shorter mine life) and decreasing the payback period.