PRELIMINARY ECONOMIC ASSESSMENT ON THE BERLIN DEPOSIT, COLOMBIA

Report Prepared for

U3O8 Corp.

Report Prepared by

Tenova Mining & Minerals (Australia) Pty Ltd

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1 SUMMARY

U3O8 Corp. ("U3O8 Corp." or "the Company") is a Toronto-based, Toronto Stock Exchange (TSX: UWE) and OTCQX (OTCQX: UWEFF) listed company focused on exploration and resource expansion of uranium and associated commodities in South America. The Company has an advanced portfolio of uranium projects in the region with initial deposits defined in Colombia, Argentina and Guyana and a clear strategy to continue to drive further resource expansion.

U3O8 Corp. engaged Tenova Mining & Minerals (Australia) Pty Ltd (formerly Bateman Engineering Pty Ltd) ("Tenova") to undertake an independent Preliminary Economic Assessment ("PEA") on its 100% owned flagship property, the Berlin Project, which is located in Caldas Province of central Colombia.

Subsequent to acquiring the Berlin Project in April 2010, U3O8 Corp. advanced an intensive exploration program, which has resulted in a maiden uranium resource supported by a suite of by-products including phosphate, vanadium, rare earths (yttrium and neodymium) and other metals delineated on one-third of the property to date. The Company has also achieved positive metallurgical results derived from extensive test work conducted by four independent laboratories that show the uranium and the suite of other commodities of economic interest at Berlin can be efficiently and effectively extracted using a ferric iron leach method. The PEA incorporates a complete flow sheet for processing of the Berlin material from beneficiation and extraction to recovery of the individual commodities.

The PEA provides a base case valuation for the project on the initial uranium resource estimated in accordance with National Instrument 43-101 ("NI 43-101") of 1.5 million pounds ("Mlb") at 0.11% U_3O_8 Indicated and 19.9 Mlb at 0.11% U_3O_8 Inferred, defined on three kilometres ("km") of the 10.5 km mineralised trend at Berlin.

All figures in the report in are in US$ ("$"), unless otherwise noted. The PEA study used the base case uranium price of $60 per pound ("lb"), which is the average reported price for long-term contracts over the previous 12 months (sources: UxC Consulting, TradeTech). The PEA is based on an average 1.2 Mlb of uranium produced annually over a 15 year mine life from a 500,000 tonne ("t") per year underground mine. The Berlin Project is expected to yield revenue of approximately $429 /t of mineralised material against an operating cost of $233 /t and generate cumulative cash flow of $915 million over the mine life. Uranium revenue is cash-flow positive as revenue from the by-products, principally phosphate, vanadium, nickel and yttrium, should more than pay for mining and extraction of the uranium. Berlin's pre-tax net present value ("NPV") is $192 million at a 10% discount with an internal rate of return ("IRR") of 17%. The project would require a capital investment of $450 million (including sustaining capital of $43 million and a $41 million contingency) with a pay-back period of 4.9 years.

Capital and operating costs were also estimated for an alternative case in which the mineralised material is not beneficiated using acetic acid before undergoing leaching by an acidic ferric iron leach, termed the non-acetic option. In the non-acetic option, capital expenditures would decrease to $441 million (including sustaining capital and a $41 million contingency). Revenue would decrease to $406 /t as no revenue would be generated from the gypsum produced as a by-product in the acetic step, against a lower operating cost of $201 /t. The non-acetic alternative is modestly more economic yielding an NPV of $223 million at a 10% discount with an IRR of 19%.

An increase in resources should enhance the economics of the project for both base cases by extending the life of the mine and/or increasing the mining rate. Although the PEA indicates robust economics on the initial resource, given the large size potential of the Berlin deposit, recommendations are to concentrate first on expanding the size of the resource over the entire 10.5 km trend, and then upgrading of resources from Inferred to Indicated category towards advancing to pre-feasibility. In addition, ongoing metallurgical test work should continue to confirm, build on and refine the process to further reduce capital and operating costs that may have a positive impact on project economics. Hydrological and geotechnical studies are also recommended for incorporation in future conceptual mine designs.

This report has been prepared in compliance with NI 43-101 and Form 43-101F1. The PEA is preliminary in nature as it includes Inferred mineral resources that are considered too speculative geologically for economic consideration that would enable them to be classified as mineral reserves. Mineral resources are not mineral reserves and have not demonstrated economic viability. There is no certainty that the results obtained from the PEA will be realised.

1.1 Project Description

The Berlin Project is an advanced exploration project in Caldas Province, Colombia with defined Inferred and Indicated mineral resources of uranium, phosphate, vanadium, rare earths and other commodities contained within the same mineralised layer. The property is located about 80 km northeast of the provincial capital, Manizales, and approximately 150 km northwest of the national capital, Bogota. The project covers an area of 12,665 hectares ("Ha") comprising five contiguous concessions, which are 100% owned by U3O8 Corp. through its wholly-owned subsidiary Gaia Energy Investments Ltd. ("Gaia Energy"). Two of the properties (664-17 and 736-17) within the Berlin Project are in the process of being transferred to Gaia Energy from Anglo Gold Ashanti Limited.

1.2 Geology and Mineralisation

The Berlin Project lies on the eastern flank of the Cordillera Central where remnants of a mid-Mesozoic fluvio-marine sedimentary sequence overlie basement schists of the Cajamarca Complex. The sedimentary sequence that contains the mineralised unit at Berlin defines an upward-fining progression. This transgressive continental to marine sequence forms part of a large basin that extends from Colombia through Ecuador into Peru and the black shales constitute an important source bed for hydrocarbons in the region. The Colombian Andes developed in response to roughly east-west shortening in the mid-Pleistocene. Related deformation in the Berlin area resulted in the formation of the syncline that hosts the mineralisation in the project area.

The Cretaceous strata form a 10.5 km long, canoe-shaped fold (syncline) at Berlin. Folding of the lower Cretaceous sedimentary sequence at Berlin is assumed to have taken place in response to inversion of the basin which started in the Paleogene. The large extent of the alaskite batholith on the west, and the location of smaller alaskite batholiths on the east flank of the syncline at Berlin are believed to have played a key role in the mineralisation of the sedimentary units at Berlin.

The Berlin Project is located within the zone of influence of the Palestina Fault System that forms the western bounding structure to the Cretaceous sequence in the Berlin area. The fault strikes 010° to 020° and can be traced over a distance of more than 400 km. The eastern margin of the Cretaceous sequence in the Berlin area is marked by the San Diego Fault that is a north-striking splay that merges with the Palestina Fault near the northern tip of the Creteaceous sequence at Berlin.

The mineralised unit encountered in the drilling to date on the Berlin Project is in a sedimentary layer that lies beneath an organic-rich, black shale. The mineralised layer changes in composition from a sandstone in the near-surface oxidised zone to a carbonate rock in the unoxidised zone at depth. Mineralisation lies at the top of this variable unit. Mineralisation in the other commodities such as phosphate, vanadium and yttrium occurs in the same uranium-bearing layer.

Drilling has confirmed that mineralisation encountered in trenches at surface extends to depth where it follows the asymmetric "U"-like cross-sectional shape of the fold. Microscopic study of drill core samples shows that uranium occurs mainly as the mineral uraninite that has a close association with organic carbon. The majority of the phosphate occurs as fine, crystalline fluorapatite ($Ca_5(PO_4)F$) masses in the sandstone, carbonate-bearing siltstone and carbonate rock. Most of the metals of potentially economic interest occur as phosphate minerals, or are associated with fluorapatite.

The mineralised unit has an average thickness of three metres ("m") in the maiden resource area. Immediately north of the resource area, the mineralised unit thins in a 500m wide swath that extends across the syncline. North of this, the mineralised unit thickens and attains of exceeds the average thickness of the mineralised zone in the resource area.

The Berlin Project has shown remarkable geological continuity with the mineralisation consistently intersected in a specific and easily identifiable limestone-sandstone unit in both the exploration and resource areas. The mineralised layer is sandwiched between conspicuous marker units that can be traced throughout the 6.3 km of the 10.5 km Berlin trend that has been drilled to date.

1.3 Status of Exploration

Prior exploration on the Berlin Project was conducted by the French company, Minatome, between 1978-1981 and culminated in the drilling of 11 bore holes for a total of 2,136 m, the excavation of 20 trenches and three adits. Minatome made a historic resource estimate of 12.9 million tonnes ("Mt") at 0.13% U_3O_8 (38 Mlb U_3O_8) on the southern 4.4 km of the 10.5 km long keel-shaped fold at Berlin (Castaño, 1981). Minatome's historic estimate was not done in accordance with NI 43-101; and therefore, should not be construed as a current mineral resource, but is merely included for historical context of the project. Historic work did not include estimates for commodities other than uranium. Historic data from trenching also showed that anomalous grades of uranium continue along strike to the north.

U3O8 Corp. began exploration on the Berlin Project when it acquired the property in April 2010. Due to the stratiform nature of the mineralisation at Berlin, the principal objective was to define the extent and consistency of the known mineralised layer through trenching and drilling. The project is in steep terrain in which trenches are excavated by hand in areas where the mineralisation comes to surface and drilling is conducted from platforms cut into hillsides.

Trench sites were identified using historic data and geological maps from the Minatome exploration that indicated areas of outcropping mineralisation. The majority of the trenches are located on the more accessible southern part and eastern flank of the syncline, where mineralisation has been shown to occur over a strike distance of 8.5 km. To date, 38 trenches have been excavated and assay results support follow up with drilling along the entire 10.5 km trend at Berlin.

U3O8 Corp's 2010-2011 drill program culminated in the drilling of 82 bore holes for 18,523 m from which the initial Inferred and Indicated mineral resources were defined on the southern 3 km of the 10.5 km mineralised trend at Berlin. Additional wide-spaced exploration drilling of 15 holes for 6,445 m of which 11 intersected mineralisation, has shown similar grades of uranium and the other elements that extend over a further 3.3 km of the trend and this area is ready for infill drilling. Trenching shows that the remaining 4.2 km of the Berlin trend is mineralised and this portion has yet to be drilled.

Further exploration drilling is planned to show the size potential of the entire Berlin trend, to be followed by infill drilling in due course towards the goal of increasing the current mineral resource.

1.4 Mineral Resource Estimates

The PEA is based on a NI 43-101 resource estimate prepared by Coffey Mining Pty Ltd. ("Coffey Mining") and reported in the March 2, 2012 technical report. The initial resource estimate on the Berlin deposit was delineated on a 3 km sector of a 10.5 km mineralised trend. Mineral resources were estimated for uranium, phosphate, vanadium, yttrium, neodymium, nickel, molybdenum, rhenium and silver contained within the 0.04% U_3O_8 mineralised shell (Coffey Mining, 2012).

As a result of the metallurgical test work, Coffey Mining was requested to estimate resources for zinc and calcite using the same model applied in the 2012 resource estimate (Coffey Mining, 2012), which have been included in this report. Metallurgical testing showed that zinc is efficiently extracted by the acidic ferric iron leach and easily recovered at little additional cost, and hence, although the grade of zinc in the mineralised material is not high, it provides a modest, positive contribution to revenue. Gypsum is an additional by-product generated from calcite when acetic acid is used in the beneficiation step of the metallurgical process on the Berlin Project. Therefore, a resource was estimated in order to incorporate gypsum revenue in the cash flow model for this PEA.

A recommended cut-off grade of 0.04% U_3O_8 has been used for the reported resource estimates summarised in Table 1-1.

Table 1-1: Resource estimate summary for uranium and other commodities in the Berlin deposit at a cut-off grade of 0.04% U_3O_8

NI 43-101 Resource	Tonnes (million)	Uranium		Phosphate		Vanadium		Yttrium	
		Grade	(Mlb)	Grade	(Mt)	Grade	(Mlb)	Grade	(t)
Indicated	0.6	0.11%	1.5	8.4%	0.5	0.4%	6.0	460ppm	294
Inferred	8.1	0.11%	19.9	9.4%	0.8	0.5%	91.0	500ppm	4,066

NI 43-101 Resource	Tonnes (million)	Neodymium		Nickel		Molybdenum		Rhenium	
		Grade	(t)	Grade	(Mlb)	Grade	(Mlb)	Grade	(t)
Indicated	0.6	110ppm	70	0.2%	3.1	570ppm	0.8	6ppm	4
Inferred	8.1	100ppm	813	0.2%	42.1	620ppm	11.0	7ppm	55

NI 43-101 Resource	Tonnes (million)	Silver		Zinc		Calcite	
		Grade	(Moz)	Grade	(Mlb)	Grade	(Mt)
Indicated	0.6	2.8ppm	0.06	0.3%	4.4	48.8%	0.29
Inferred	8.1	3.4ppm	0.89	3.0%	45.0	36.5%	3.00

No mineral reserve estimate has been undertaken for the Berlin Project at the date of this PEA.

1.5 Mining Methods

The PEA is based on an underground mine on the maiden resource and assumes that approximately 80% of the resource is mined with 20% left as pillars for mine support. After a first year mine production of 250,000 t of mineralised material, the mine would ramp up to a production rate of 500,000 t during a 15 year mine life. Planned daily output from the operation is 1,430 t of mineralised material and 715 t of waste.

Mineralisation at Berlin lies in a specific layer that is “U”-shaped in cross section. The steeply inclined parts of the deposit require mining by cut and fill methods while the shallowly inclined parts use room and pillar mining techniques. Mine access is from a portal located at an elevation of 805 m above mean sea level (“amsl”) via a 760 m ramp at a 15% inclination.

Crushing and milling would be done in an underground chamber so that dust can be controlled to the highest safety standards. Initial tests show that the mineralised material is amenable to semi-autogenous grinding (“SAG”). . In the base-case scenario in which mineralised material is treated with an acetic acid pre-leach, the volume of tailing is reduced to the extent that they could all be accommodated as backfill in the underground mine. In the alternative process, which eliminates the acetic acid pre-leach step, excess tailing would gravitate to a long-term storage facility located approximately 14 km from the mine site.

1.6 Recovery Methods

A conceptual flow sheet was developed from extensive metallurgical test work on intercepts from 35% of all bore holes drilled in the initial resource area at Berlin. The process route has been designed to efficiently extract multiple commodities, to be versatile in terms of reagent consumption, to be compatible with standard recovery methods and to create an environmentally benign tailing. The three main components of this process are:

1. Beneficiation of the crushed mineralised material using acetic acid (vinegar) to remove calcite and concentrate the valuable commodities into 40-47% of the original mass, which makes the subsequent extraction and recovery processes more efficient, reduces capital and operating costs and decreases the volume of tailings by 50-60%;

2. Extraction of the metals and phosphate into a pregnant liquor solution (“PLS”) by an acidic ferric iron leach method. The rates of extraction achieved for each metal and phosphate is shown in Table 1-2; and

3. Recovery of the individual elements from the PLS by ion exchange (“IX”), solvent extraction (“SX”) and direct precipitation.

Table 1-2: Extraction rates of the metals and phosphate from the Berlin deposit

Commodity	Extraction %
Uranium	96.1
Phosphate	98.9
Vanadium	66.3
Yttrium	86.1
Neodymium	59.6
Nickel	65.9
Molybdenum	51.4
Rhenium	32.8
Zinc	95.9
Silver	25.0

1.7 Project Infrastructure

The Berlin Project is in Caldas Province of central Colombia, and is favourably located between the country’s largest cities – 140 km from Bogota and 100 km from Medellin. The town of La Dorada is 60 km east of the project and lies on the principal paved road between Bogota and Medellin. La Dorada provides port facilities on the Magdalena River, which is navigable by barge to the coastal port of Barranquilla. Barranquilla is the largest port in Colombia and provides access to the export destinations of the Caribbean, Central America, the southern U.S. and northern South America. A defunct railway line also runs from La Dorada to the port town of Santa Marta on the Caribbean coast. The Colombian government is reported to be planning to have the railway line operational in 2015, which would offer an alternative link between the project and the Caribbean coast.

Large volumes of quality water are available in the project area, although majority of the water used in the operation would come from the underground mine. The PEA indicates that about 75% of the required electricity for the plant can be produced from heat generated from a sulphuric acid plant that forms an integral part of the processing facility. The plant could supply about 46% of the power requirement for the entire Berlin operation. In addition, the project is planned to be linked to the 395 megawatt (“MW”) La Miel hydroelectric dam located about 12 km from the Berlin Project. La Miel would serve as an additional power source.

1.8 Preliminary Economic Assessment

Based on the 12 month trailing average long-term uranium price of approximately $60 /lb, the PEA provides an independent valuation of a base case on the initial mineral resource defined to date on 3 km of the 10.5 km mineralized trend at Berlin.

The Berlin Project is expected to generate $3.0 billion in revenue with free cash flow of $915 million over the 15 year life of the mine. Highlights of the PEA are summarised in Table 1-3.

Table 1-3: PEA summary (pre-tax, base case at $60 /lb uranium price)

Annual mill throughput		500,000 t
Total uranium produced		16.3 Mlb
Annual uranium production		1.2 Mlb
Mine life		15 years
Cumulative free cash flow		$915 million
NPV at 10% discount		$192 million
IRR		17%
Pay-back period		4.9 years
Cash cost per lb of U_3O_8, net of by-products		<$0 /lb
Capital investment	Initial capital	$366 million
	Sustaining capital	$43 million
	10% contingency	$41 million
	Total Capital	**$450 million**

The PEA is preliminary in nature as it includes Inferred mineral resources that are considered too speculative geologically for economic consideration that would enable them to be classified as mineral reserves. Mineral resources are not mineral reserves and have not demonstrated economic viability. There is no certainty that the results of the PEA will be realised.

Berlin’s pre-tax NPV at a 10% discount is $192 million. The project’s NPV and IRR are shown at various discount rates and uranium prices in Table 1-4. Berlin NPV’s sensitivity to a plus or minus 10% change in capital and operating expenditures is shown in Table 1-5.

Uranium (33%), phosphate (29%), nickel (14%), vanadium (9%) and yttrium (6%) would represent the most significant contribution to revenue at Berlin. The financial model shows that revenue from the by-products covers the cost of extracting the uranium, resulting in Berlin having a production cash cost of less than $0 per pound of uranium. Gypsum (5% of revenue) is an additional by-product generated from calcite when acetic acid is used in the beneficiation step of the process. Commodity prices of the other elements used in the PEA are shown in Table 1-6.

An increase in resources is likely to result in a higher IRR from the current 17% by providing flexibility to extend the mine life and/or increase the mining rate.

Table 1-4: Sensitivity analysis of Berlin NPV (in $ million) to uranium price

Uranium Price		$40	$50	$60 (Base Case)	$70	$80
Discount Rate	0%	$595	$755	$915	$1,074	$1,234
	5%	$246	$346	$447	$546	$647
	10%	$59	$126	$192	$258	$325
	15%	($45)	$1	$47	$94	$140
IRR		12%	15%	17%	20%	22%
Pay-back period (years)		6.3	5.5	4.9	4.4	4.0

Capital expenditures are less sensitive to discount rate due to the front-end nature of the initial capital costs.

Table 1-5: Berlin NPV (in $ million) sensitivity to ±10% change in capital and operating costs

Discount Rate	Base Case NPV	Effect on NPV of 10% Change in Operating Costs	Effect on NPV of 10% Change in Capital Costs
0%	$915	$915 +/- $149	$915 +/- $45
5%	$447	$447 +/- $93	$447 +/- $40
10%	$192	$192 +/- $62	$192 +/- $38
15%	$47	$47 +/- $44	$47 +/- $35

Figures may not add due to rounding.

Table 1-6: Commodity prices used in the revenue estimates for the Berlin PEA

Commodity	Price $
U_3O_8	60 /lb
NH_4VO_3	9 /kg
H_3PO_4 (Phosphoric acid)	1 /kg
$Y(OH)_3$	50 /kg
$NiCO_3$	9.25 /lb
$Nd(OH)_3$	50 /kg
Mo	12 /lb
Zn	0.89 /lb
Gypsum	30 /t

Capital Costs

The PEA is based on an annual throughput of 500,000 t of mineralised material with a capital investment of $450 million including $43 million in sustaining capital and $41 million contingency (Table 1-7).

Table 1-7: Summary of Capital Costs

Items	Capital costs (million)
Mining	$74
Process plant	$195
Infrastructure and tailing management	$71
Other (EPCM, indirect costs, etc.)	$69
Contingency	$41
TOTAL	**$450**

Operating Costs

Operating costs for the Berlin Project are expected to average $233 /t of mineralised material including royalties and a 10% contingency (Table 1-8).

Table 1-8: Summary of Operating Costs

Items	Cost per tonne
Revenue-based royalties	$18
Mining and dewatering	$60
Processing	$132
G&A	$4
Contingency	$19
TOTAL	**$233**

Non-Acetic Option

Capital and operating costs were also estimated for an alternative case in which the mineralised material is not beneficiated using an acetic acid before undergoing leaching with acidic ferric iron leach, referred to as the non-acetic option. In the non-acetic option, capital expenditures are estimated at $441 million (including sustaining capital and a $41 million contingency), which are summarised in Table 1-9. Revenue would be approximately $406 /t of mineralised material as no revenue would be generated from the gypsum as a by-product in the acetic step, against a lower operating cost of $201 /t of mineralised material. At a uranium price of $60 /lb, the non-acetic alternative is modestly more economic yielding an NPV of $223 million at a 10% discount with an IRR of 19%. The project's NPV and IRR, assuming a non-acetic option are shown at various discount rates and uranium prices in Table 1-10.

Table 1-9: Summary of capital costs – non-acetic option

Items	Capital costs (million)
Mining	$74
Process plant	$177
Infrastructure and tailing management	$84
Other (EPCM, indirect costs, etc.)	$65
Contingency	$41
TOTAL	**$441**

Table 1-10: Sensitivity analysis of Berlin NPV to uranium price and discount rate – non-acetic option

Uranium Price		$40	$50	$60 (Base Case)	$70	$80
Discount Rate	0%	$663	$822	$982	$1,142	$1,302
	5%	$291	$391	$491	$591	$691
	10%	$90	$157	$223	$290	$356
	15%	($21)	$24	$71	$117	$163
IRR		14%	16%	19%	21%	23%
Payback period (years)		5.9	5.2	4.6	4.2	3.8

1.9 Conclusions

Berlin has shown remarkable geological continuity with the mineralisation consistently intersected in a specific and easily identifiable limestone-sandstone unit that lies beneath an organic-rich black shale in both the resource and adjacent exploration areas. The mineralised layer is sandwiched between conspicuous marker units that are evident over a strike distance of 6.3 km of the Berlin trend.

The mineralised unit at Berlin is similar to the shape of the hull of a canoe and recent drilling has shown that the deepest part of the keel reaches depths of over 700 m below surface.

Grades intersected at depth along the keel are similar to assays obtained in trenches where the mineralised layer reaches surface on the eastern side of the fold. With scout drilling having defined the approximate shape of the mineralised unit along a further 3.3 km segment of the Berlin trend beyond the current resource area, infill drilling would focus on the shallower parts of the eastern flank towards further resource growth.

Based on the similarity of average grades of the maiden resource with exploration drill results in the northern part of the trend, large increases are expected in resources of the other elements that occur with the uranium, namely: vanadium, phosphate, molybdenum, rhenium, rare earths (yttrium and neodymium) and nickel.

Trenching has added to the understanding of both the nature of mineralisation and its continuity. Uranium grades obtained in drilling and trenching completed to date are consistent with that indicated by the historical work. The trench assay results further support the continued drilling of the deposit throughout the 10.5 km Berlin trend.

A conceptual flow sheet has been developed from extensive metallurgical test work on intercepts from 34% of all bore holes drilled in the initial resource area at Berlin. The process comprises three main components: (1) beneficiation as a means of concentrating the commodities of value into as small a mass as possible for further processing; (2) extraction of the metals and phosphate by an acidic ferric iron leach method with excellent results; and (3) recovery of the individual elements from solution by IX, SX and direct precipitation.

The mineralisation at Berlin is in a limestone that contains about 55% carbonate minerals (calcite) that consume acid that is required to leach the metals and phosphate from the mineralised rock. Beneficiation of the crushed mineralised material using acetic acid removes the calcite and concentrates the valuable commodities into 40-47% of the original mass, which makes the subsequent extraction and recovery process more efficient, reduces capital and operating costs and decreases the volume of tailings by 50-60%.

Financial modelling in the PEA shows that the uranium could be mined and recovered from Berlin at a zero cash cost, thanks to the revenue from the associated commodities.

Capital and operating costs were also estimated for an alternative case in which the mineralised material is not beneficiated using an acetic acid before undergoing acidic ferric iron leach, referred to as the non-acetic option.

It is important to note that the economic viability of the Berlin Project is not dependent on beneficiation by acetic acid and, in fact, the more economically attractive method is direct processing of the run-of-mine (“ROM”) material without beneficiation with acetic acid.

Flotation is also being examined as an alternative beneficiation method to acetic acid leach as a means of selective removal of the carbonate from the mineralised material at Berlin. An advantage of using flotation is that the technique uses fewer reagents, although it would not result in a gypsum by-product credit for the project.

The PEA is based on the initial mineral resource defined on 3 km of the 10.5 km mineralised trend at Berlin and provides a base case from which the economics of the project can be improved as the size of the deposit increases through further resource drilling and as efficiencies are realised from ongoing metallurgical test work.

1.10 Recommendations

Although the PEA indicates robust economics on the initial resource, recommendations are to concentrate first on expanding the size of the resource over the entire 10.5 km trend, and then on upgrading the mineral resource from Inferred to the Indicated category towards advancing to pre-feasibility studies. In addition, ongoing metallurgical test work should continue to test and refine the process and improve the efficiencies of extraction and recovery, which may have a positive impact on the economics of the project.

Based on the technical work completed to date, Tenova's primary recommendations (budget estimate by U3O8 Corp.) include:

- Wide-spaced exploration drilling of the northern 4.2 km of the mineralised trend at Berlin that remains to be explored to fully define the size potential of the Berlin deposit (budget $3.3 million);
- Infill drilling of the 3.3 km of the mineralised trend that has already undergone exploration drilling with the aim of increasing the current Inferred mineral resources (budget $6.6 million);
- Infill drilling to upgrade the current and contiguous Inferred resources to the Indicated category (budget $11.0 million);
- On conversion of a significant part of Inferred mineral resources to Indicated, a pre-feasibility study ("PFS") should be undertaken on the potentially larger Berlin deposit; and
- Metallurgical test work should continue with the aim of improving and refining the conceptual process. The focus of this test work should be on efficiently beneficiating the mineralised material, which could lead to capital and operating cost savings while maintaining the revenue stream. This test work should be of a level appropriate to pre-feasibility stage studies.

2 INTRODUCTION

2.1 Scope of Work

This PEA for the Berlin Project in Caldas Province, Colombia was conducted by Tenova Mining and Minerals (Australia) Pty Ltd (formerly Bateman Engineering Pty Ltd), ("Tenova") for Toronto Stock Exchange and OTCQX listed U3O8 Corp.

This PEA is being done at an earlier stage of the project than is normally the case. A PEA would typically be done when the full extent of the deposit is known. The rationale for undertaking a PEA at this early stage, given the multi-commodity nature of the Berlin deposit, is to establish the economic viability of the process as early as possible. Passing this early test provides the justification to continue with exploration to increase the size of the project. In addition, details of the capital and operating cost estimates highlight specific areas where focus is required to achieve greater efficiencies to improve the economics of the deposit.

This PEA is based on the initial resource that was estimated by Coffey Mining (2012) and as updated herein, although mineralisation is open along trend to the north of the resource as it is currently defined, and there is potential for the deposit to grow significantly in size. Coffey Mining personnel have taken two site visits to the Berlin Project. The first was by Mr Neil Inwood from June 13 to 16, 2011 in preparation for the initial mineral resource estimate. The second site visit was by Mr Doug Corley from June 13 to 19, 2012 in preparation for the inclusion of other commodities in the resource estimate.

The PEA incorporates results of extensive metallurgical test work reported in the NI 43-101 resource estimate (Coffey Mining, 2012) as well as subsequent test work undertaken under the direction of Dr Paul Miller, as described herein. Metallurgical test work has been undertaken by four independent, internationally renowned laboratories, three of which are located in Australia and one in Canada.

The mine design and final financial model were prepared by Mr Pedro Véliz who has extensive specialist knowledge of mining in rock formations of the type found at Berlin. Mr Véliz visited the Berlin Project from April 23 to 26, 2012 during which he visited the site and also undertook an extensive review of bore hole core at the core storage and logging facility in Ibague, Colombia.

2.2 Qualifications and Experience

The PEA was completed under the supervision of Mr Louis de Klerk, Pr. Eng., BSc(Eng) (Chemical), P Grad Dip in Advanced Process Design and Mr Pieter Niemann, Pr. Eng., at Tenova. Tenova is part of the Techint Group, a leading global engineering firm, that has been providing process design, development and construction services to the resource sector for over 90 years with extensive and specific process and engineering experience in the extraction of uranium, phosphate, rare earths, nickel and zinc as well as in sulphuric acid production. Clients and projects have included AREVA's Imouraren, Cameco's Key Lake, Harmony, Rossing, Kazatomprom and Arafura's Nolans Bore project (rare earths, phosphoric acid, gypsum production and uranium recovery). Tenova has also constructed phosphate plants in Australia, South America, Africa and the U.S.

Dr Paul Miller, Managing Director of Sulphide Resource Processing Pty Ltd., has overseen the metallurgical test work carried out on the Berlin Project.

He is a metallurgist specialising in hydrometallurgy and has over 30 years' experience in the commercial application of processes for the treatment of sulphide-bearing ore. Dr Miller has a doctorate in Chemical Engineering, is a member of the Institute of Mining and Metallurgy, London, and is also a Chartered Engineer.

Mr Pedro Pino Veliz, President of P&K Projetos e Consultoria LTDA ("P&K"), is a Civil Mining Engineer (P.E.), Eng. Dr. (IT) SME and has more than 35 years of experience in mining operations and executive management. He heads up P&K's 315 person team headquartered in Rio de Janeiro. In operation since 1992, P&K has extensive experience serving its Brazilian and international clientele with rock mechanics, rock characterisation, underground stability and hydrological studies. This has included detailed design of tunnels and stations, methods of excavation and specifications for support methods in the expansion of the Rio de Janeiro subway system in preparation for the 2014 World Cup and 2016 Olympic Games.

Mr Doug Corley is a professional geologist with over 20 years' experience in exploration, mining and resource geology. He is a Member of the Australian Institute of Geoscientists and a Registered Professional Geoscientist (R.P. Geo) Resource Geologist.

The Qualified Persons as defined under NI 43-101 of the Canadian Securities Administrator ("QP") for this report are listed in Table 2-1 along with their area of responsibilities.

Table 2-1: Qualified persons and area of responsibility

QP Name	Area of Responsibility
Louis de Klerk, P. Eng	Project Management Report Sections 1, 2, 3, 4, 5, 6, 7, 8, 9, 10, 15, 19, 20, 23, 24, 25, 26, 27
Pieter Niemann, P. Eng	Recovery Methods Report Sections relevant to recovery methods: 1, 2, 3, 4, 17, 18, 21, 24, 25, 26
Paul Miller, PhD (Ch.E), CEng	Mineral Processing and Metallurgy Report Sections: 1, 13, 25, 26
Pedro Véliz, P.E.	Mining Methods Report Sections: 1, 16, 18, 21, 22, 25, 26
Doug Corley, R.P. Geo	Mineral Resource Estimation Report Sections: 1, 11, 12, 14, 25, 26

2.3 Limitations and Caution

This PEA is preliminary in nature. It is based largely on Inferred resources, a category of mineral resources that is considered too speculative geologically to have the economic considerations applied to them that would allow them to be categorised as mineral reserves. There is no certainty that the PEA will be realised.

Note that mineral resources that are not mineral reserves do not have demonstrated economic viability.

It is important to note that a PEA is defined as a conceptual study of the potential viability of mineral resources. Capital and operating costs reported herein are estimated to an accuracy of ±35%. The confidence level of the economic viability of a deposit in a PEA is less than that of a PFS which has, in turn a lower confidence level than that of a feasibility study (“FS”). Capital and operating cost estimates for PFS, for example, would typically be to an accuracy of ±20%, while in a FS, they would be estimated to an accuracy of ±10 to ±15%.

2.4 Conventions and Standards

All units in this report are according to the International System of Units. All currency values are US$ (“$”). Acronyms and abbreviations used in this report are commonly used in the minerals industry and are listed with brief explanations in Appendix A.

2.5 Independence

Tenova is independent of U3O8 Corp. Tenova is not an insider, associate or affiliate of U3O8 Corp.

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3 RELIANCE ON OTHER EXPERTS

For Sections 4 to 11 of this PEA:

- Tenova has relied on information provided by the Company and as reviewed by Coffey Mining and Dr Hernán Rodríguez.
- Tenova has not performed an independent verification of land title and tenure as summarised in Section 4 of this PEA.
- Tenova did not verify the legality of any underlying agreement(s) that may exist concerting the concessions or other agreement(s) between third parties.
- Tenova relies on U3O8 Corp. to provide correct information regarding the title and legal status on the exploration concessions that constitute the Berlin Project as well as details regarding environmental legislation and requirements, which were provided by Dr Hernán Rodríguez, independent Colombian counsel to U3O8 Corp.
- Tenova was informed by U3O8 Corp. that there are no known litigations potentially affecting the Berlin Project.

Dr Rodríguez is based in Bogota, Colombia and is a partner with Norton Rose Group, an international law firm with offices across Asia, Europe, Canada, Africa, the Middle East and Latin America. Dr Rodríguez has been practicing law since 1995, and has participated in hydrocarbon, mining and infrastructure projects in Colombia, including projects for ports, railways and oil refineries, and has been actively involved in mergers and acquisitions transactions, especially in the mining industry.

For Sections 12, 13 and 14 of this PEA, Tenova has relied on information provided by Coffey Mining and Sulphide Resource Processing Pty Ltd (Dr Paul Miller).

For Section 16 of this PEA, Tenova has relied on information from Mr Pedro Véliz.

For Sections 18 and 19 of this PEA, Tenova has relied on information provided by the Company.

For Section 20 of this PEA, Tenova has relied on information provided by the Company and Dr Hernán Rodríguez.

For Section 21 of this PEA, Tenova has relied on reagent and utility cost information provided by the Company.

For Sections 22, 23 and 24 of this PEA, Tenova has relied on information provided by the Company.

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4 PROPERTY DESCRIPTION AND LOCATION

4.1 Location and Size of the Mineral Properties

The Berlin Project area is located in central Colombia in the province of Caldas some 80 km northeast of the provincial capital, Manizales, and approximately 150 km northeast of the national capital, Bogotá (Figure 4-1). Conspicuous geographical features in close proximity to Berlin, which can be viewed in satellite imagery, are the La Esmeralda Lake (5 38 49.45N 74 57 43.09W), 5.3 km north of Berlin and the La Miel hydroelectric dam, 1.5 km south of Norcasia.

Figure 4-1: Map showing general location of the Berlin Project in Caldas Province, Central Colombia

The Berlin Project covers an area of 12,665 Ha comprising five contiguous concessions in the municipality of Samana, Province of Caldas in central Colombia. The location of the mineralised unit in the Berlin Project is shown relative to the concession boundaries in Figure 4-2.

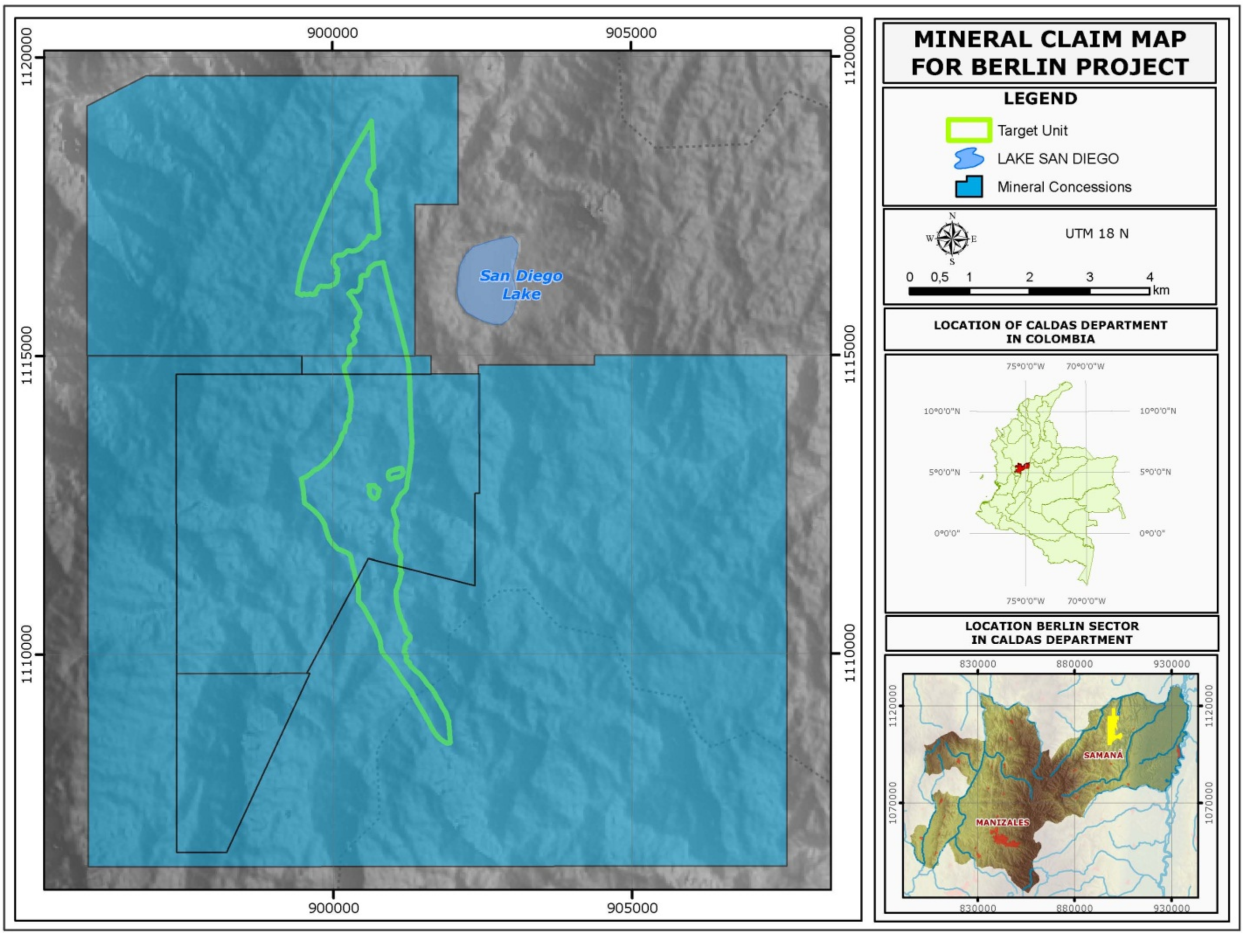

Figure 4-2: Location of the mineral properties that constitute the Berlin Project shown relative to the outline of the mineralised unit

Details of the Gauss-Kruger and UTM coordinates of the corner points of the five mineral properties that make up the Berlin Project are listed in Table 4-1. UTM coordinates are in Zone 18 North, and are in units of metres.

Table 4-1: Berlin Project corner points of the mineral properties that constitute the Berlin Project (Gauss-Kruger and UTM, Zone 18 North Coordinates)

Concession	Corner Point	Gauss- Kruger		UTM	
		Northing	Easting	Northing	Easting
755-17	1	1,111,460	902,000	619,371	504,207
	2	1,111,920	900,210	619,828	502,417
	3	1,110,000	899,180	617,907	501,391
	4	1,110,000	897,000	617,904	499,212
	5	1,115,000	897,000	622,901	499,204

Concession	Corner Point	Gauss- Kruger		UTM	
	6	1,113,000	902,000	620,910	504,205
	7	1,115,000	902,070	622,909	504,272
	8	1,113,000	902,070	620,910	504,275
756-17	1	1,107,000	897,830	615,220	499,671
	2	1,107,000	897,000	615,219	498,842
	3	1,110,000	897,000	618,217	498,837
	4	1,110,000	899,230	618,221	501,066
664-17	1	1,115,310	907,212	623,540	509,035
	2	1,115,310	904,000	623,535	505,835
	3	1,115,150	904,000	623,375	505,835
	4	1,115,150	902,060	623,372	503,886
	5	1,115,000	902,060	623,223	503,886
	6	1,115,000	902,070	623,223	503,896
	7	1,113,000	902,070	621,224	503,899
	8	1,113,000	902,000	621,223	503,830
	9	1,111,460	902,000	619,684	503,832
	10	1,111,920	900,210	620,141	502,042
	11	1,110,000	899,180	618,221	501,016
	12	1,110,000	899,230	618,221	501,066
	13	1,107,000	897,830	615,220	499,671
	14	1,107,000	897,000	615,219	498,842
	15	1,110,000	897,000	618,217	498,837
	16	1,115,000	897,000	623,215	498,829
	17	1,115,000	899,110	623,218	500,938
	18	1,115,310	899,110	623,528	500,937
	19	1,115,310	895,513	623,522	497,343
	20	1,106,763	895,513	614,980	497,356
	21	1,106,763	907,212	614,998	509,049
IFM 08221X	1	1,115,310	899,109	623,215	501,312
	2	1,115,310	900,001	623,216	502,203
	3	1,115,310	901,266	623,218	503,467
	4	1,115,000	901,266	622,908	503,468
	5	1,115,000	899,110	622,905	501,313
736-17	1	1,115,310	901,000	623,218	503,202
	2	1,115,310	895,513	623,209	497,718
	3	1,119,500	895,513	627,397	497,711
	4	1,120,000	896,500	627,898	498,697
	5	1,120,000	901,722	627,906	503,916
	6	1,117,840	901,722	625,746	503,920

Concession	Corner Point	Gauss- Kruger		UTM	
	7	1,117,840	901,000	625,746	503,198
	8	1,115,310	901,000	623,218	503,202

4.2 Nature of Mineral Properties, Ownership and Tenure

4.2.1 Mineral Rights in Colombia

In Colombia, exploration and exploitation of mining resources, such as uranium, are formalised by execution of a concession contract (the "Concession Contract") with the correspondent national mining authority pursuant to the mining legislation Law N° 685/01, duly amended by Law 1382 of 2010.

Since 1940, the mining authority has been the Ministry of Mines and Energy. The Ministry has, in turn, delegated some mining-related matters to national and provincial authorities (the "Mining Authority"). Specifically, the National Institute of Geology and Mining (INGEOMINAS) is responsible for managing royalties and maintaining the national register of Concession Contracts. Specific Provincial Governments are charged with the granting, execution and performance of Concession Contracts and other related administrative proceedings within their respective provincial boundaries. This is the case for the Caldas Province, in which the Berlin Project is located, whereby the Province manages all exploration and mining-related activities for minerals found within the province, except for coal and emeralds, which are managed by INGEOMINAS.

By means of Decree 4134 of November 3, 2011, the Colombian Government created the National Minerals Agency (the "Agency"), which assumed responsibility for the granting, execution and administration of Concession Contracts throughout Colombia in 2012.

Under Colombian law, foreign individuals and corporations have the same rights as Colombian individuals and corporations. Foreign companies are required to constitute a branch, subsidiary or affiliate in Colombia before they may be granted a Concession Contract. U3O8 Corp's wholly-owned subsidiary, Gaia Energy Investments Ltd., has a branch in Colombia called Gaia Energy (Colombia) Ltd., which has Concession Contracts covering the Berlin Project area (Figure 4-3). Gaia Energy Investments Ltd. is a British Virgin Islands-registered, wholly-owned subsidiary of Gaia Energy Inc. Gaia Energy Inc. Is, in turn, an Ontario-registered, wholly-owned subsidiary of U3O8 Corp. (Figure 4-3). On November 26, 2010 the name of Energentia was changed to Gaia Energy Investments Ltd. and the Colombian branch's name to Gaia Energy Investments Ltd. Sucursal Colombia ("Gaia Energy (Colombia) Ltd." or "Gaia").

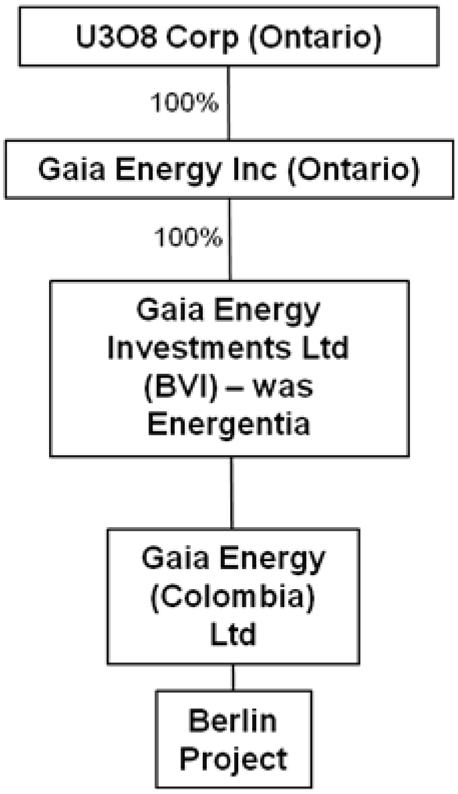

Figure 4-3: Corporate structure through which the Berlin Project is held

The Concession Contracts for the Berlin Project were executed and registered under Law 685 of 2001, prior to Law 1382 of 2010 coming into effect. The Berlin Concession Contracts are valid for a 30 year term that can be extended for 30 additional years. The initial term of the Concession Contracts comprises the following three phases:

- Exploration – three years with possible extension for eight additional years;
- Construction – three years for the construction and assembly of the infrastructure, extendable for one additional year; and
- Exploitation – the remaining years for the exploitation stage, extendable for a further 30 years upon request by the concession holder.

Concession Contracts granted after February 9, 2010, when Law 1382 of 2010 came into effect, have the same initial 30 year term, but are extendable for 20 additional years. In addition, the five year term for the exploration phase may be extended for a total of 11 years prior to the construction phase.

Concession Contracts for exploration convey the right to explore the defined areas for specified metals or minerals. A concession owner has the first right to include additional commodities and metals to the original Concession Contract. The rights of the Concession Contract can be assigned totally or partially to another party, subject to prior notice and authorisation by the Mining Authority and as long as the obligations under the Concession Contract have been duly complied with.

Surface rights are separate from the exploration or mining rights. None of the Concession Contracts covering the Berlin Project imply any surface rights – acquisition of surface rights must be negotiated directly with the landowners. The Concession Contracts are renewed annually, provided that work commitments and property payments due to the Mining Authority have been met.

During the stages of exploration, construction and assembly, the concession holder must pay an annual surface fee to the Mining Authority based on the assigned areas and the number of years of the exploration period. Table 4-2 shows the schedule of fees for Concession Contracts granted prior to February 9, 2010. The fee is based on the Colombian minimum daily wage, which is approximately $10.00 at present.

Table 4-2: Annual concession fees required to maintain a Concession Contract

Concession Size	Fees per Hectare ("Ha") Approximately
Up to 2,000 Ha	$10.00
2,000 Ha – 5,000 Ha	$20.00
5,000 Ha – 10,000 Ha	$30.00

Concession Contracts granted after February 9, 2010 are subject to a different annual fee of approximately $10.00 /Ha for the first five years of exploration, increasing by 25% of the annual fee per hectare for each year thereafter. During the construction phase, the annual fee is frozen at the maximum level paid in the last year of the exploration phase. Prior to the expiration of the exploration period, the concession owner is required to file a Working and Construction Plan from the Mining Authority as well as an Environmental Impact Assessment ("EIA") with the relevant Provincial environmental authority. Upon approval of both filings, the project can be advanced to the construction and assembly stage.

4.2.2 Berlin Project Properties

The five properties that constitute the Berlin Project are Concession Contracts that have an initial 30 year term that, provided that they are maintained in good standing, expire in 2037 (Table 4-3). Applications for the Concession Contracts were made in 2007 and the initial 3 year exploration period ended in 2010, whereupon a 2-year extension was granted and application is underway for an additional 2-year extension to the exploration phase within the 30 year Contract.

Three of the Concession Contracts are held by Gaia Energy and the Mining Authority has issued resolutions authorising the transfer of the other two contracts to Gaia Energy by AngloGold Ashanti (Table 4-4). The history of AngloGold Ashanti's involvement in the project is described in Section 6.

The commodities for which each mineral property applies are listed in Table 4-4. However, as described in Section 4.2.1, the current title holder has the first right to apply for additional commodities in the area which each Concession Contract covers.

Table 4-3: Summary of critical dates relating to the mineral concessions that constitute the Berlin Project

Concession File #	Initial Exploration Application Date	Date of Expiry of the 1st 30 Year Term	Extensions to Exploration Phase		Current Status
			Expiry of Initial Exploration Phase	Initial 2 year Exploration Phase Extension	
755-17	9 November 2007	9 November 2037	9 November 2010	9 November 2012	Application submitted for further 2 year extension to Exploration Phase
756-17	9 November 2007	9 November 2037	9 November 2010	9 November 2012	
664-17	7 December 2007	7 December 2037	7 December 2010	7 December 2012	
IFM 08221X	21 December 2007	21 December 2037	21 December 2010	21 December 2012	
736-17	9 November 2007	9 November 2037	9 November 2010	9 November 2012	

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Table 4-4: Summary of information relating to the mineral properties that constitute the Berlin Project

Concession Reference Number	Size (Ha)	Minerals Specifically Included in Exploration Permit	Annual Fee ($)	Notes to ownership by Gaia Energy (Colombia) Ltd.
755-17	2,122.6	Uranium & Radioactive Elements, Vanadium, Phosphate & Molybdenum	43,000	None
756-17	459	Uranium & Radioactive Elements, Vanadium, Phosphate & Molybdenum	4,700	None
664-17	7,304.9	Uranium & Radioactive Elements, Vanadium, Phosphate, Molybdenum, Gold	223,600	The Mining Authority issued Resolution 0241, dated February1, 2010, authorising the assignment of the Concession Contract from AngloGold Ashanti to Gaia Energy Investments Ltd. Sucursal Colombia. Registration of this resolution with the National Mining Registry is in progress.
IFM08221X	74.5	Uranium &Radioactive Elements, Vanadium, Phosphate, Molybdenum, Gold, Silver, Copper, Zinc, Platinum	760	None
736-17	2,704	Uranium & Radioactive Elements, Vanadium, Phosphate Molybdenum, Gold, Silver, Copper, Zinc, Platinum	55,000	The Mining Authority issued Resolution 2251 dated May 3, 2012, authorising the assignment of the Concession Contract from Anglo Gold Ashanti to Gaia Energy Investments Ltd. Sucursal Colombia. Registration of this resolution with the National Mining Registry is in progress.
Total	12,665		327,000	

4.2.3 Requirements to Maintain the Concession Contracts in Good Standing

Concession Fees

As outlined in general principle in Section 4.2.1 above, annual concession fees are paid to the State for each Concession Contract. The approximate cost of maintaining the Berlin concessions is $327,000 (Table 4-4).

Environmental Mining Insurance

Within 10 days following the execution of the Concession Contract, an environmental mining insurance policy must be obtained by the concession-holder as a guarantee against non-compliance with mining and environmental obligations.

Failure to meet these obligations may result in the payment of fines and the unilateral termination of the agreement by the Mining Authority. The insured value is calculated as follows for the different stages:

- Exploration – 5% of annual estimated work expenditures;
- Construction – 5% of the annual investment towards mine construction; and
- Exploitation – 10% of the result of multiplying the estimated annual production by the price of the mineral being extracted, as determined by the Government.

The insurance policy must be in full force and effect throughout the life of the Concession Contract and is renewed annually as part of the fulfilment of obligations to maintain the Concession Contract in good standing.

Budget and Work Program

At the time an application is made for an extension to the exploration phase, a budget and work program must be presented to the Mining Authorities.

Reporting

An annual report of activities on each Concession Contract must be submitted to the Mining Authority. Significant differences in expenditure and work undertaken from the planned budget and work program require explanation.

Compliance with Environmental Standards

Exploration is required to be carried out to standards demanded by the Provincial Environmental Authority, Corpocaldas, and included in the mining and environmental guides. Corpocaldas monitors environmental compliance and issues permits for trenching and drilling.

4.3 Material Agreements and Encumbrances

Concession Contracts 664-17, 736-17 and No IFM 08221X were acquired from a subsidiary of AngloGold Ashanti as described in Section 6. AngloGold Ashanti maintains a 2% net smelter royalty ("NSR") on uranium production on commencement of commercial uranium production from these properties. A payment of approximately $250,000 is due to AngloGold Ashanti on registration of Concession Contracts 736-17 and 664-17. This registration is expected to be made in 2013.

There are no back-in rights or other encumbrances on the Concession Contracts that constitute the Berlin Project.

4.4 Royalties and Colombian Tax Regime

4.4.1 Royalties

In addition to the 2% NSR on uranium due to AngloGold Ashanti, as described in Section 4.3, the Colombian Government requires a royalty that varies according to commodity as listed in Table 4-5.

Table 4-5: List of NSR on the mining of various commodities

Commodity	NSR (%)
Uranium	10
Vanadium, Phosphorus, Molybdenum, Yttrium, Rhenium, Iron, Copper, Platinum	5
Gold, Silver	4
Nickel	12
Construction Materials (incl. Gypsum)	1

The royalty is accrued at mine head, meaning that 100% of the ore mined is taken into account to calculate the royalty (product at mine head has not been subject to any process).

When calculating the royalty, the Ministry takes into account the sales price of the mineral extracted from the mine, as well as costs, such as transport and refining/processing, as estimated from various sources, including mining companies and publications.

Royalties are payable in cash or in kind.

When payable in kind, since royalties are calculated at mine head or wellhead, the royalty percentage will never form part of the product to be sold by the producer but will be as from that moment owned by the State. This means that royalties in kind will never form part of income and thus will not be considered as a cost or expense.

When payable in cash, royalties will be taken from income obtained by the producer in the sale of the product to the market. Although royalty is calculated at the mine head or wellhead, it will be payable only once the product has been sold. Thus, the royalty percentage is part of the product sold and then subtracted from the income obtained as a consideration payable to the State for the exploitation of the mine or well. Said consideration is considered as deductible for income tax purposes, as expressly accepted by tax authorities through a ruling issued in 2005.

4.4.2 Withholding Taxes on Income for the Export of Minerals and Hydrocarbons

As a general rule, the Colombian Tax Code established that income originated in the export of goods is not subject to withholding taxes, meaning that when receiving income in Colombia it will be subject to income tax but said tax will be liquidated and paid with the annual income tax return. Nevertheless, Law 1430 of 2010 authorised the government, as an exception, to levy withholding tax up to 10% on gross income for the export of hydrocarbons and other mining products.

Based on that authorisation, according to article 1 of Decree 1505 of 2011, the government introduced the withholding tax at a rate of 1% on gross income obtained in the export of hydrocarbons and other mining products. This withholding tax is more a self-withholding since the beneficiary of payment is the one who has to withhold, declare and pay the withholding tax.

This self-withholding has the following features:

i. The self-withholding of 1% on gross income from the export of hydrocarbons or mining products is treated as a tax credit. Once the income for the export is obtained under the accrual system, that is, when the right to collect the income arises even if not paid yet (and even if amounts for the export are received abroad), the entity receiving it will calculate the 1% and will declare and pay it in the withholding taxes return of the corresponding month, that is, within the first two weeks of the following month.

ii. The self-withholding will be recorded as part of the account receivables since it is a tax credit that may be offset against the final income tax payable calculated for the correspondent taxable year (corresponding to the calendar year). Income tax for a given taxable year has to be declared and paid during the following year (usually during April), at which time the tax credit will be used. If there is a credit balance resulting from the payment of the withholding taxes, a refund request of such credit balance can be filed or the credit can be carried forward without time limit.

This self-withholding tax applies only on income from the export of hydrocarbons and other mining products. If any entity exports manufactured products, although made with minerals or hydrocarbons as raw materials, such export will not be subject to self-withholding tax.

4.4.3 Free Trade Zones

Free Trade Zones (“FTZ”) are delimited geographical areas within Colombian territory in which industrial activities are performed under a special legal regimen for tax and foreign trade purposes.

An industrial user is a legal entity installed or located exclusively in one or various FTZs. Therefore, when investors want to perform activities in a FTZ they need to incorporate a special purpose vehicle that will be located in the FTZ. If the industrial user performs activities outside the FTZ, it may lose its qualification as such.

The main benefits in a FTZ are:

i. The income tax rate is 15% (the general income tax rate is 33%).

ii. Goods entered into the FTZ are considered to be outside of the national customs territory. Thus, the import of goods, including raw material and capital goods, into the FTZ is not taxed with customs duties or VAT. Furthermore, the sale of goods from the Colombian customs territory to the FTZ is not subject to VAT.

In Colombia, there are various FTZs of which the most important and popular ones are the ones located in Barranquilla, Bogota (there are some under construction in neighbouring municipalities, i.e. Mosquera, Cota), Cali and Medellín. To establish a business on a FTZ, the first requirement is to identify the locality and whether there is an adequate space available and then to file documentation, including economic and financial analysis of the activities to be performed, with the operator of the correspondent FTZ.

4.4.4 Importation of Goods

In general, import duties are 0% to 20% depending on the type of goods imported. VAT of 16% is then calculated on CIF value plus import duty.

There are various alternatives for importing goods into Colombia with different types of import duties and VAT benefits:

a) Ordinary import: As a general rule ordinary imports are subject to import duties and VAT. If the machinery is produced in a country that has signed free trade agreements with Colombia, then the goods might have a special import duty rate.

b) "Temporary import for re-exportation in the same condition". Under this type of import, it is possible not to pay any customs duties (import duties and VAT) or at least, it is possible to defer said payment in semi-annual quotas. At the end of the temporary import term, the merchandise must be re-exported or imported under the ordinary regime. The temporary import can be made using one of the following alternatives:

   – Short term temporary import: The maximum term for this type of import is six (6) months that can be extended for another three (3) months and exceptionally for a further three (3) months. Under this type of temporary import no customs duties (import duties and VAT) are accrued. Notwithstanding, said customs duties shall be calculated in the import return in order to determine the amount of the guarantee that has to be issued in favour of the Customs Authorities.

   – Long term temporary import: This type of import allows the import of machinery and equipment that can be depreciated and that from its repeated use in a productive process, a tangible or intangible good is obtained without changing its nature, for a maximum of five (5) years. This term can be longer if the importer proves before Customs Authorities that the work to be done by the machine requires a longer period of time. Customs duties have to be paid and must be calculated when filing the import return with local customs authorities. Custom duty payments can be deferred during five (5) years in semi-annual quotas taking into account the exchange rate of the date of each payment.

c) Temporary import of heavy machinery and its spare parts for "Basic industries": According to article 428(e) of the Colombian Tax Code, VAT is not accrued on heavy machinery and its spare parts for "Basic Industries" imported under the temporary import regime. The mentioned article considers, among other, the oil and mining industries and heavy chemistry (heavy chemistry is not legally defined; should be understood as the industry that transforms raw materials into semi-elaborated products) as "Basic Industries". In order to obtain this benefit, customs authorities must certify that the imported goods are not produced in Colombia.

d) Import of machinery and its spare parts for oil and mining sectors: According to Decree 562 of 2011 modified by Decree 1570 of 2011, there is a 50% import duty exemption applicable to the import of machinery, equipment and spare parts assigned (i) to exploitation, transformation and transport in the mining activities and (ii) to exploitation, transport and oil refining. This benefit will be in force until August 2015.

4.5 Environmental Liabilities

No environmental liabilities on the Berlin Project are known to the author at this time. There are no other known significant factors and risks that may affect access, title, or the right or ability to perform work on the Berlin Project.

4.6 Permits Required to Conduct the Proposed Work

No specific environmental license is required for the exploration stage. However, all work must be done in accordance with environmental guidelines issued by the Ministry of Mines and Energy and the Ministry of the Environment. Three drill-related permits are required for:

- Solid waste management;
- Forestry use;
- Water use; and
- Social permission to establish drill pads.

The water-use permit typically takes about six months to process while the other drill-related permits typically take 2-3 months. Water from one permitted site can be used for various drill platforms in the vicinity.

An EIA must be completed in order to obtain an environmental licence from the respective Provincial environmental authority before mine construction may commence.

4.7 Known Significant Factors and Risks

No significant factors or risks related to the properties that constitute the Berlin Project are known to the authors at this time.

4.8 Other Factors Related to the Mineral Properties

4.8.1 Repatriation of Funds and Payment of Dividends

Companies that have sales in foreign currencies are required to repatriate these amounts in Colombian Pesos through authorised foreign exchange intermediaries. However, under the current exchange regulations, branches of foreign companies undertaking exploration and exploitation of uranium, petroleum, natural gas, coal or ferronickel, are exempt from this repatriation obligation. Instead, branches of companies involved in the exploration and mining of these commodities are required to repatriate the amounts necessary to pay expenses in local currency.

4.8.2 Free Trade Agreements

Colombia signed a free trade agreement with Canada in June 2010 and with the U.S. in 2012.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Access

The Berlin Project can be accessed by road from either Bogotá or Ibague, both of which have commercial airports. The distance to the municipality of La Dorada, an old river port on the Magdalena River, is 191 km from Bogotá and 167 km from Ibague. From La Dorada, a secondary unpaved road leads 56 km westwards to Berlin, passing through the municipality of Norcasia (Figure 5-1).

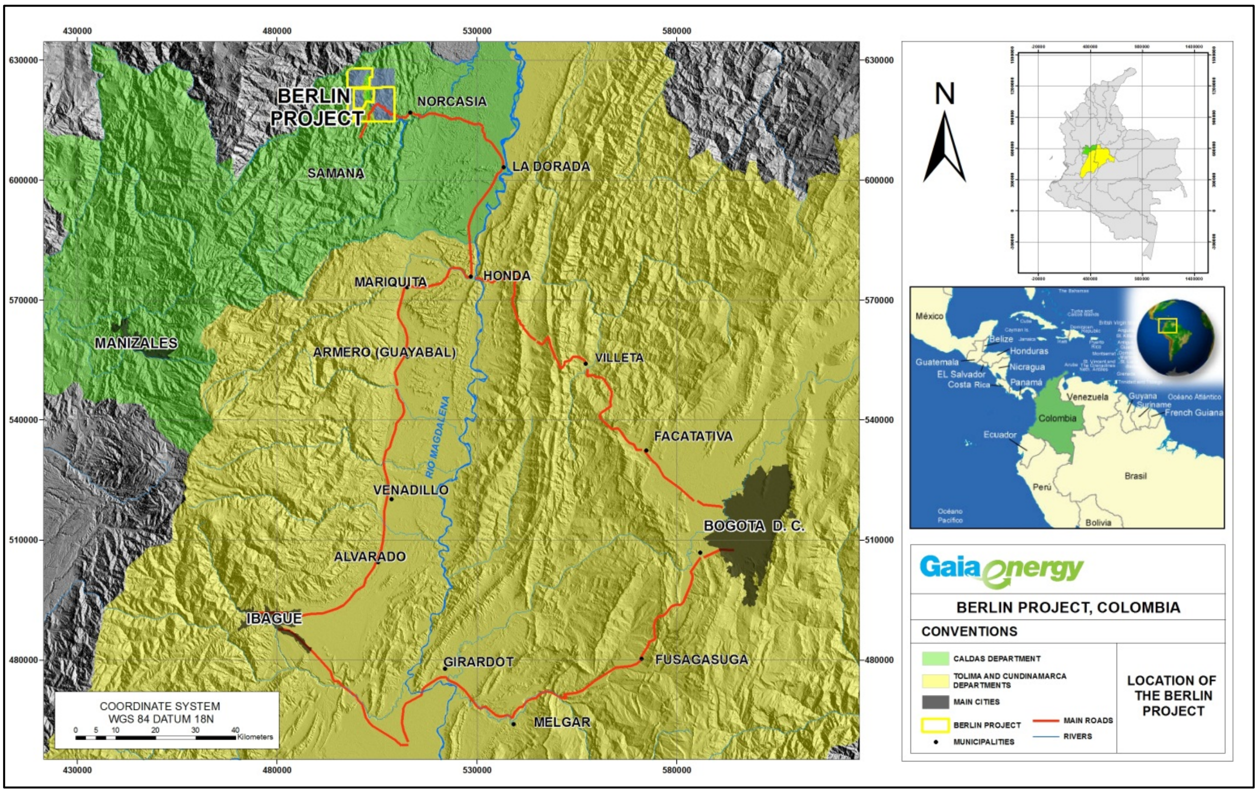


Figure 5-1: Map showing the general location of the Berlin concession areas in Caldas Province relative to local infrastructure and towns

5.2 Topography, Elevation and Vegetation

5.2.1 Vegetation

The mountains and valleys of the project area are mostly covered by rainforest; however, zones of grass and crops can also be seen (Figure 5-2).

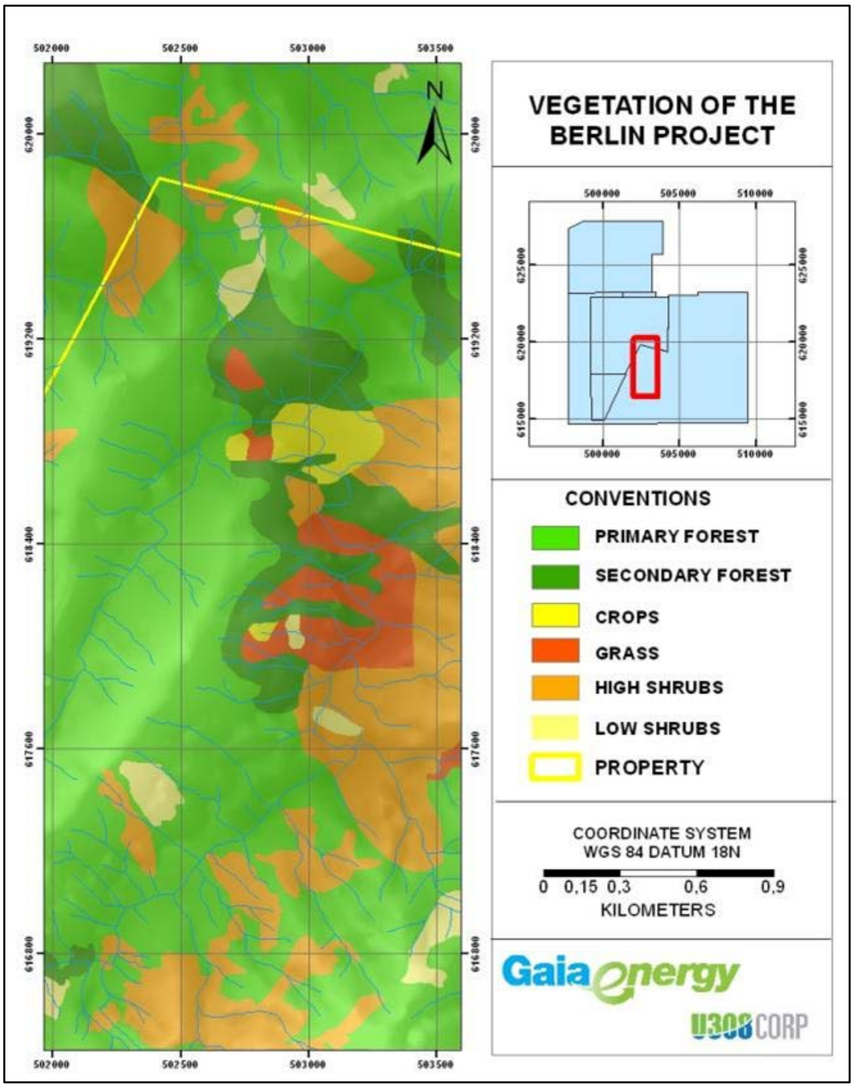


Figure 5-2: Classification of vegetation of part of the Berlin Project

5.2.2 Physiography

The location of the project on the eastern foothills of the Colombian Central Range means that its topography is abrupt with altitudinal variations between 850 m and 1300 m amsl characterised by thin and elongated edges, steep hillsides of between 35 and 70°, and the predominance of “V” – shaped river valleys.

This rugged physiography contrasts with the peneplain of the Magdalena River to the east of the Cordillera.

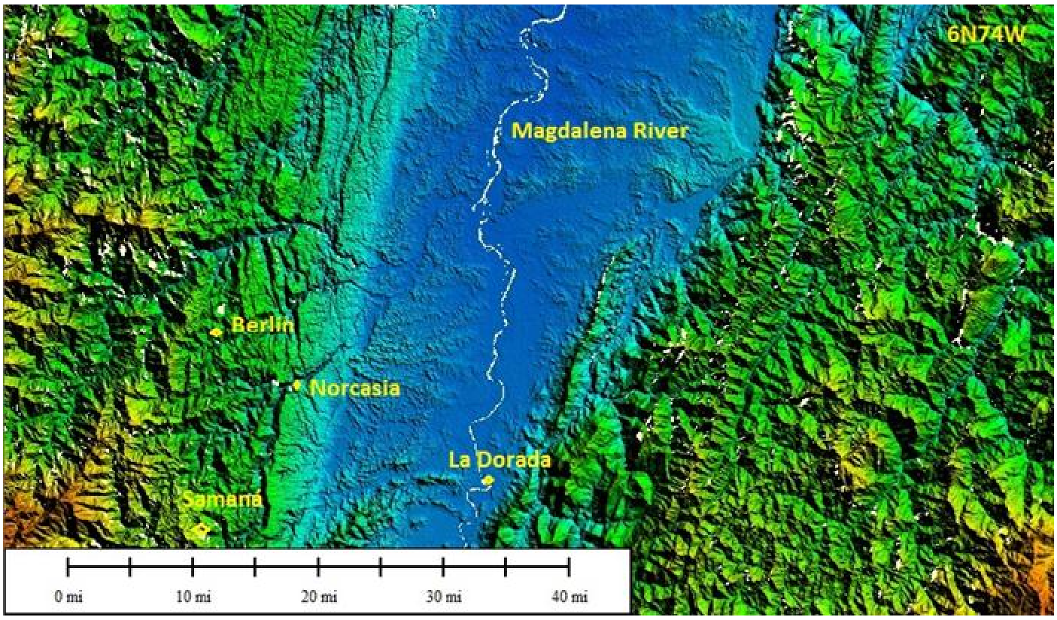


Figure 5-3: SRTM imagery showing the topographic relief in the area of the Berlin Project

Figure 5-4: Typical landscape of the Berlin Project

5.3 Infrastructure, Population and Local Resources

5.3.1 Population and Infrastructure

In 2010, U3O8 Corp. staff carried out a socio-economic analysis of the township of San Diego, which is part of the municipality of Samaná (Figure 5-3 and Figure 5-5). There are some 4,200 inhabitants in 32 settlements linked by dirt tracks, with no vehicular access. Subsistence agriculture is the principal economic activity. San Diego has a police station, church, elementary school, high school, Home for Peasant Youth, Mayor's Office, cemetery, sports court, two hotels, two nightclubs, an old age home, two billiard halls, clinic, centre for community access to internet and shops.

Norcasia, 8 km from Berlin, is the closest urban area to the project, with a population of 7,000 offering shops, a hospital and public transportation.

The Magdalena River is navigable by barge from the town of Puerto Boyoca, some 65 km northeast of the project area to the port of Barranquilla on the Caribbean coast.

A railway line leads from the town of El Dorado on the Magdalena River to the port town of Santa Marta on the Caribbean coast. Although the railway line is not currently in use, the government has flagged it as a priority infrastructure project for completion by about 2015.

5.3.2 Water Supply

High rainfall and a rich tributary system guarantee high volumes of quality water. The river system has been dammed to provide hydropower at La Miel, just south of Norcasia.

5.3.3 Power Supply

The energy consumed in the Berlin Project comes from the sub-station of Norcasia, which is distributed by the CHEC (Caldas Hydroelectric). The 395 MW La Miel hydroelectric dam is located approximately 12 km from the central part of the project area (Figure 5-5).

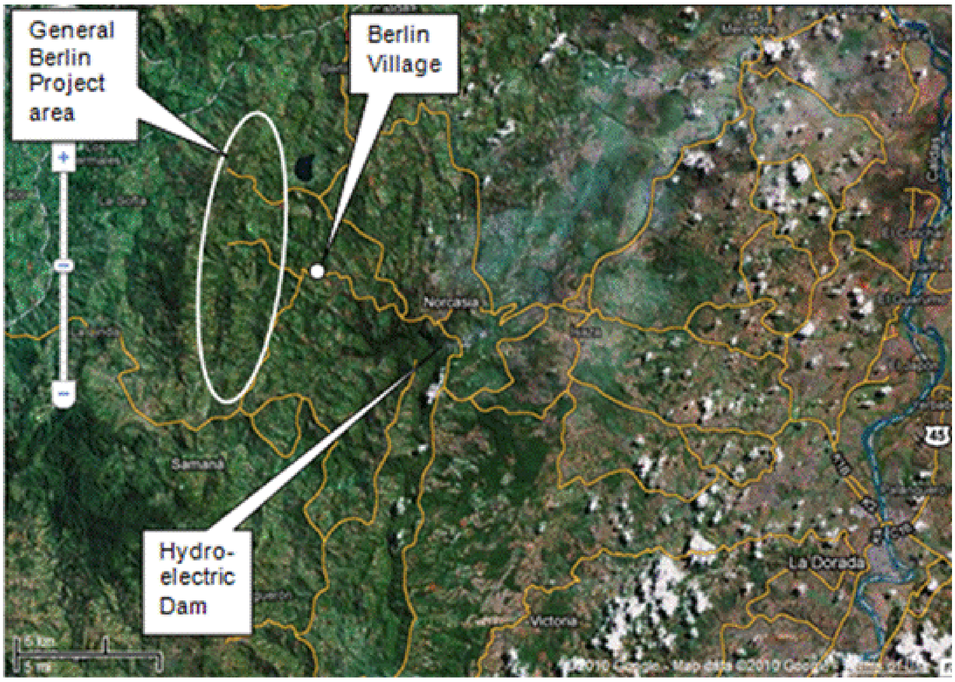


Figure 5-5: Road network and location of La Miel hydroeletric plant in relation to Berlin

5.4 Climate and Operating Season

The mean temperature in the area of the project varies between 21 and 25°C throughout the year, with average annual rainfall of 5,000 to 8,000 millimetres ("mm") per year (Corpocaldas, 2001). Rainfall occurs throughout the year, with lower rainfall between December and February and from June to August, and higher rainfall between March and May and between September and November. These conditions allow for operations year-round.

With a view to obtaining accurate, site-specific data, the U3O8 Corp. installed a Weatherhawk meteorological station on its property at Vereda Alto San Juan in August 2012 (Figure 5-6).

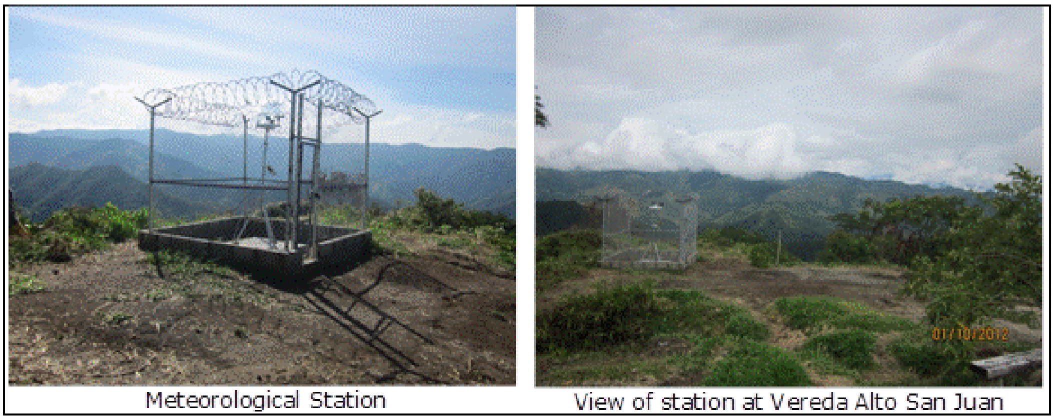


Figure 5-6: Meteorological station at Vereda Alto San Juan

The weather station continuously records temperature, dew point, humidity, atmospheric pressure, solar radiation, precipitation, wind speed and wind direction.

6 HISTORY

6.1 Prior Ownership

Minatome, a French exploration company, which has now been incorporated into AREVA, undertook detailed exploration on mineral concessions that covered the Berlin Project between 1979 and 1981. Its work involved basic geological investigation followed by exploration drilling, initial metallurgical test work and estimation of a uranium resource (not NI 43-101 compliant). Minatome is reported to have withdrawn from the Berlin area when the company was nationalised by the French government in 1981, which coincided with a slump in uranium prices.

After Minatome's withdrawal from the project, the concessions reverted to the State. The United Nations Development Program ("UNDP") reviewed the technical work undertaken on the project in 1982 and focused on the potential to recover uranium, molybdenum, vanadium and phosphate from the Berlin area. The UNDP suggested that Dense Media Separation may play a part in recovery of the various commodities.

Energentia Resources Inc. ("Energentia"), formerly KPS Ventures Ltd., applied for two exploration concessions directly from the State (Concession Contracts 755-17 and 756-17 as more fully described in Section 4) and also entered into an agreement with AngloGold Ashanti to acquire three properties (Concession Contracts 664-17, 736-17 and IFM08221X) in the Berlin Project area in 2007. Specifically, the agreements that Energentia entered into for these three properties was with a wholly-owned subsidiary of AngloGold Ashanti, Sociedad Kedahda SA. Due to the slow nature of related legal procedures, the assignment of these three properties to Energentia was ongoing when Mega Uranium Ltd. ("Mega") purchased Energentia on May 1, 2008. U3O8 Corp. then purchased the South American assets from Mega in a deal that closed on April 10, 2010. Among other assets, U3O8 Corp. purchased Energentia as a wholly-owned subsidiary. The Colombian State has issued resolutions approving the transfer of the concessions that originally belonged to AngloGold to Gaia Energy. The registration of the assignment of the concessions to Gaia Energy has been completed for one concession and the process is ongoing for the other two.

6.2 Historical Exploration

Uranium was identified in the phosphatic strata in a regional radiometric prospecting program undertaken by the Colombian Instituto de Asuntos Nucleares ("IAN") between 1977 and 1983. Minatome obtained permission from IAN to explore the Berlin Project area for uranium in 1979. Field-based exploration carried out by Minatome identified a sedimentary unit near the base of the Cretaceous sequence in the Berlin area as having significant uranium grades. Rock-chip sampling resulted in the identification of highly anomalous uranium values over the entire strike length of the synform in the Cretaceous sequence in the Berlin area (Figure 6-1, Figure 6-2 and Table 6-1).

Minatome's assay results from surface rock-chip sampling were substantiated by independent sampling undertaken by Naranjo (1983), at the Universidad National in Bogota, who reported mapping the uraniferous unit over a strike length of approximately 3 km in the southern part of the synform in the Cretaceous sequence. Analyses from seven channel rock-chip samples and one point sample were reported as shown in Table 6-2.

Minatome's exploration concentrated on the southern 5 km of the 10.5 km long syncline where access is easier and where outcrop of the sedimentary sequence is generally better in comparison with the north. The apparent consistency of uranium grades along strike led Minatome to excavate 20 trenches and three adits, the latter with the objective of confirming mineralisation in fresh exposures beneath the saprolite. The location of the adits is shown in Figure 6-2 and the assay results reported by Minatome are listed in Table 6-1.

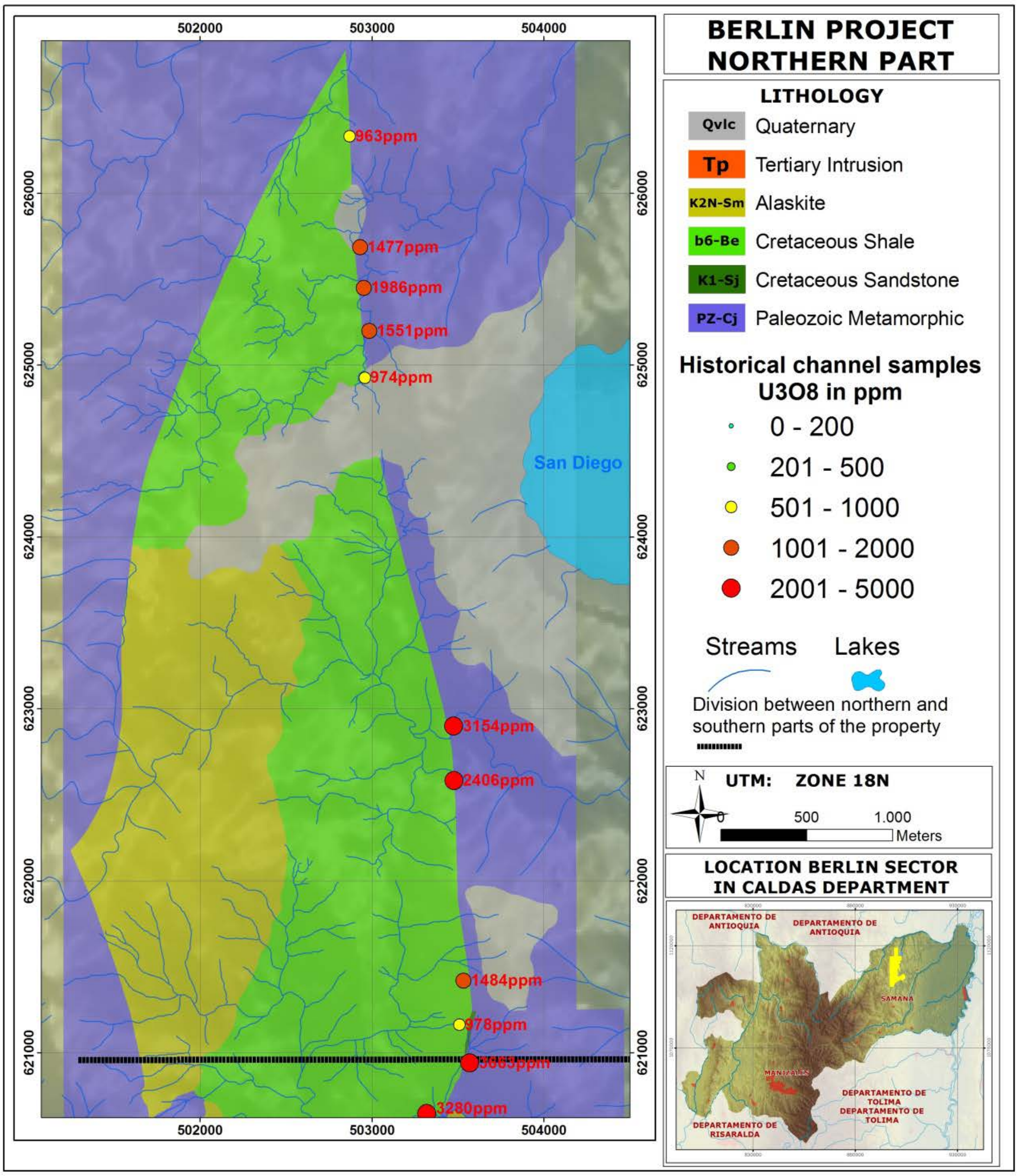


Figure 6-1: Channel sample uranium grades in uraniferous unit, northern Berlin Project area

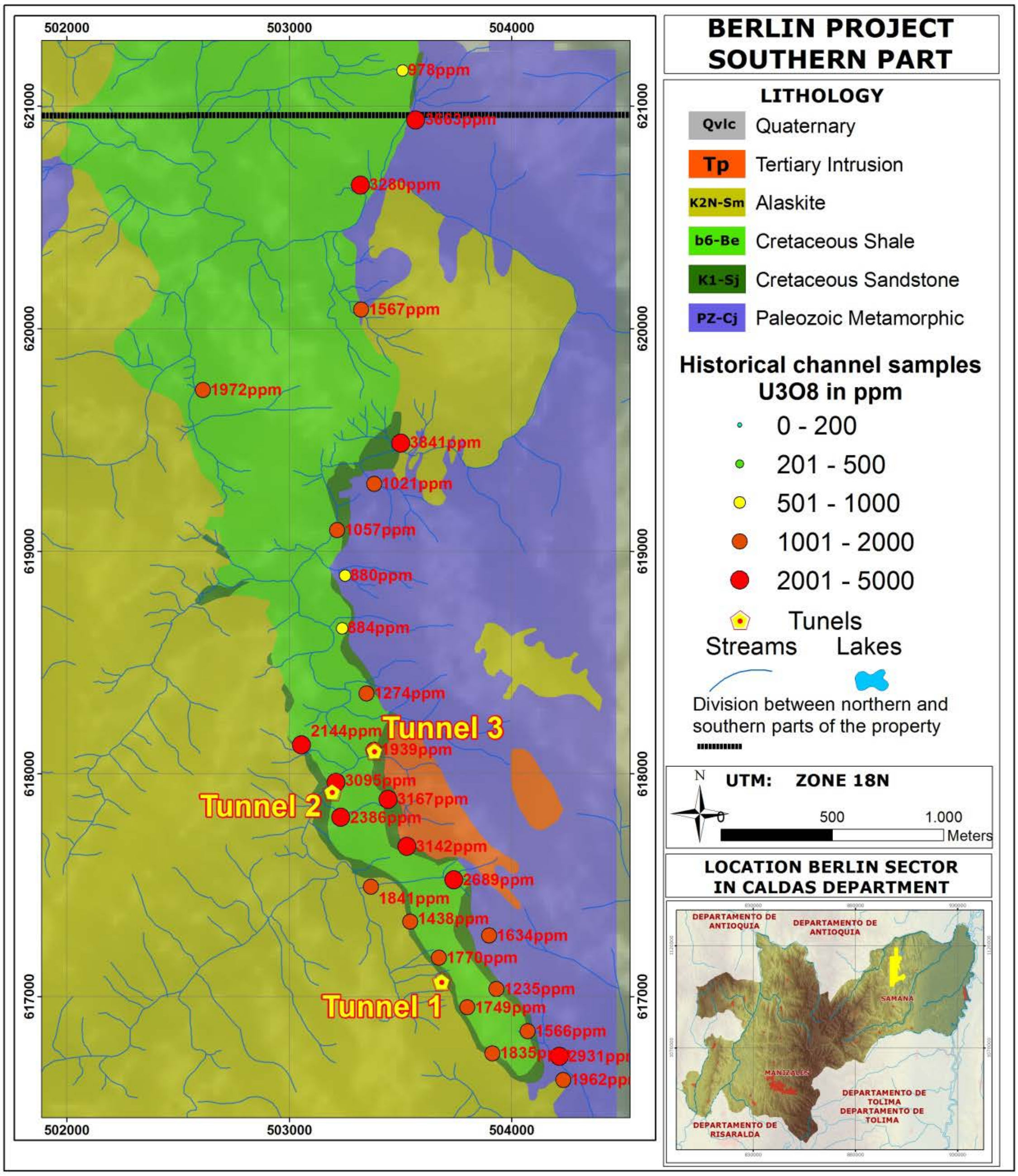


Figure 6-2: Channel sample uranium grades in uraniferous unit, southern Berlin Project area

Table 6-1: Berlin Project rock-chip sample U_3O_8 assay values and thickness of mineralised unit reported by Minatome (Castano, 1981)

Sample No.	Location - Coordinates		U_3O_8 Grade (%)	Sample Thickness (m)
	UTM North	UTM East		
BW-375	619,725	502,610	0.197	0.2
6W-200	618,132	503,055	0.214	1.4
BW-175	617,962	503,209	0.310	1.8
BW-150	617,805	503,231	0.239	2.3
BW-125	617,494	503,367	0.184	1.1
BW-100	617,337	503,543	0.144	1.0
BW-75	617,174	503,673	0.177	1.8
BW-50	616,953	503,800	0.175	0.7
BW-25	616,744	503,913	0.184	2.4
BE-0	616,625	504,230	0.196	1.0
BE-01	616,731	504,213	0.293	2.0
BE-25	616,843	504,071	0.157	1.1
BE-50	617,034	503,930	0.124	0.9
BE-75	617,274	503,899	0.163	2.6
BE-100	617,523	503,739	0.269	1.2
BE-125	617,675	503,527	0.314	0.9
BE-150	617,886	503,443	0.317	2.3
BE-175	618,108	503,379	0.194	1.1
BE-200	618,362	503,346	0.127	1.3
BE-225	618,654	503,237	0.088	0.7
BE-250	618,891	503,250	0.088	1.1
BE-275	619,095	503,215	0.106	0.4
BE-300	619,304	503,381	0.102	1.4
BE-325	619,487	503,499	0.384	1.5
BE-400	620,086	503,322	0.157	0.2
BE-450	620,647	503,319	0.328	1.3
BE-475	620,939	503,568	0.366	1.1
BE-500	621,162	503,508	0.098	0.9
BE-525	621,419	503,532	0.148	1.3
BE-675	622,910	503,509	0.315	0.4
BE-650	622,622	503,618	0.241	2.5
BE-875	624,966	503,185	0.097	1.3
BE-900	625,226	503,131	0.155	1.2
BE-925	625,480	503,117	0.199	0.0
BE-950	625,731	503,109	0.148	0.2
BE-1025	626,344	503,243	0.096	0.2

Sample No.	Location - Coordinates	U_3O_8 Grade (%)	Sample Thickness
Coordinates are in UTM zone 18 north measured in metres. The location of these samples is shown in figures 6-1 and 6-2. Source: (Castaño, 1981)			

Table 6-2: Berlin Project corroborative rock-chip sample assay results from radioactive unit, southern Berlin Project area (Naranjo, 1983)

Sample No.	Coordinate		Height (amsl)	Radiometry cps (SPP2)	Thickness (m)	U_3O_8 %	P_2O_5 %	Mo %	V_2O_5 %
	UTM N	UTM E							
1	618,341	503,684	720	9,000	2.60	0.544	7.14	n/a	n/a
2	619,475	503,482	680	6,000	1.50	1.305	10.50	n/a	n/a
3	619,060	503,188	690	6,000	2.00	0.258 (b)	10.15	n/a	n/a
4	618,161	503,469	820	6,000	2.00	0.174 (b)	n/a	n/a	n/a
5	617,655	512,386	860	15,000	1.00	0.985	n/a	n/a	n/a
6	617,561	503,690	840	15,000	Punctual	0.836 (c)	n/a	n/a	n/a
7	617,411	503,710	940	15,000	1.50	0.495	n/a	n/a	n/a
8 (a)	623,508	503,401	810	6,000	2.00	0.648	11.10	0.49	1.15
Co-ordinates are given in UTM zone 18 north measured in metres. (Cps = radiometric counts per second and n/a = not analysed).									

Table 6-3: Summary assay results of channel samples taken through the uraniferous unit in the three adits excavated by Minatome (Locations are shown in Figure 6-2).

		Tunnel 1	Tunnel 2	Tunnel 3
Length	(m)	48	24	40
Thickness	(m)	1.8	3.75	3.25
U_3O_8	(ppm)	362	1,100	588
V	(ppm)	3,490	11,069	10,966
Mo	(ppm)	406	2,306	175
P_2O_5	(%)	6.51	4.7	8.9

Minatome then drilled 11 bore holes from five widely-spaced drill pads for a total of 2,136 m in 1980 (Figure 6-3). Although six of these drill holes are reported to have reached the target depth, nine are reported to have intersected anomalous uranium values. IAN is reported to have drilled six bore holes in the Berlin Project area in 1982 and 1983 and three of these holes are reported to have reached the target horizon and to have intersected grades similar to those reported by Minatome over similar true widths (SRK, 2006).

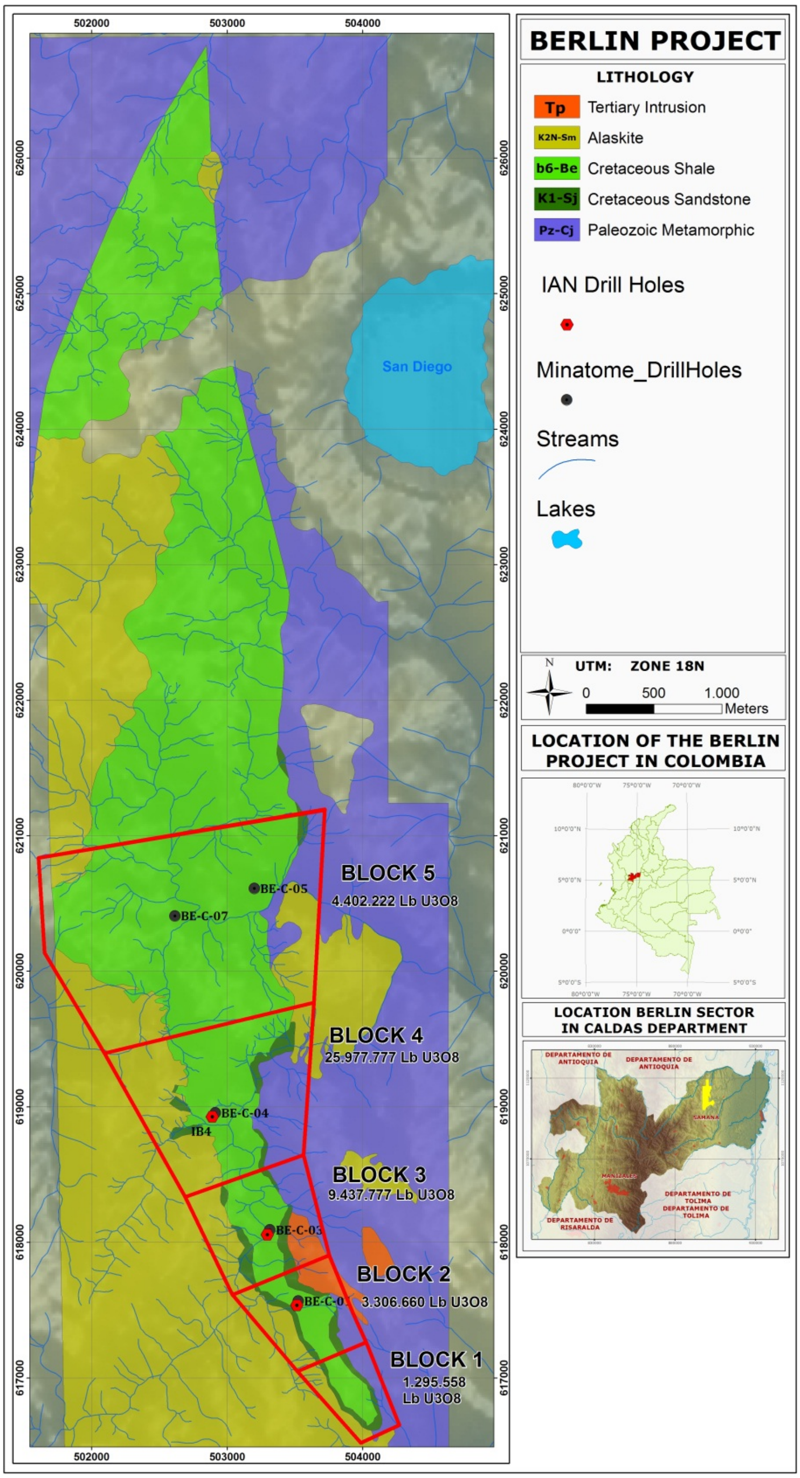


Figure 6-3: Historical drilling in the Berlin area

6.3 Historical Mineral Resource Estimates

Minatome undertook a provisional, NI 43-101 non-compliant grade estimate, using polygonal methods based on assays of mineralised intervals of the BE series of bore holes whose collar positions are shown in Figure 6-3 and rock-chip channel samples from the three adits (Table 6-1) in 1981.

The historic estimate by Minatome, which covered the southern 4.5 km of the 10.5 km-long mineralised trend in the Berlin area, was 12.9 Mt at a grade of 0.13% U_3O_8 for a total of approximately 38 Mlb of contained U_3O_8 (Castaño, 1981). Minatome's estimate was not done in accordance with NI 43-101 and; therefore, should not be construed as a mineral resource or an indication of the resource endowment of the project. The estimate has been included only for historical context of the project.

6.4 Historical Metallurgical Test Work

Minatome conducted various metallurgical tests on the mineralized rock from the Berlin Project in 1979 in Nancy, France (Roussemet & Houot, 1979). Simple acid leaching resulted in approximately 75% extraction of uranium, but with the consumption of 130 kilogram ("kg") of phosphoric acid per tonne of mineralised material.

Provisional metallurgical test work on a 35 kg rock-chip sample from the adits showed the following distribution of uranium (Roussemet & Houot, 1979):

- 5-10% of the uranium occurs on the surface of coarse fragments and is liberated during crushing of the host-rock;
- 55-60% of the uranium is associated with phosphate in the 40-200 micron ("µm") fraction. Test work shows that the phosphate is amenable to flotation; and
- Approximately 30% of the contained uranium occurs with the fine fraction, suspected to be adsorbed onto illite and other clays, and was liberated on ultrafine grinding to a nominal grain size of 8 µm.

The conclusion from Minatome's metallurgical test work was that uranium recovery was approximately 85% using a combination of flotation and ultrafine grinding (Roussemet & Houot, 1979). This work did not include an estimate of cost of processing.

7 GEOLOGICAL SETTING AND MINERALISATION

7.1 Regional Geology & Tectonic Framework

The Berlin Project lies on the eastern flank of the Cordillera Central where remnants of a mid-Mesozoic fluvio-marine sedimentary sequence overlie basement schists of the Cajamarca Complex (Figure 7-1).

The basement in the central part of the Cordillera Central consists of greenschist to lower amphibolite facies metamorphic rocks correlated with the Precambrian to Early Mezosoic Cajamarca Complex (Bürgl and Radelli 1962; Moreno-Sanchez et al., 2008). Cediel et al. (2003) place the Cajamarca Complex in the Cajamarca-Valdivia terrane. The Cajamarca-Valdivia terrane is composed of politic and graphitic schists, amphibolites, intrusive rocks and mafic to ultramafic rocks of ophiolitic origin. The Cajamarca-Valdivia terrane is a wedge-shaped tectonic unit that tapers to the south (Figure 7-1). In central Colombia, the Cajamarca-Valdivia terrane is sandwiched between the Eastern Cordillera block in the east and the Dagua-Piñon and San Jacinto terranes in the west. In southern Colombia, the Cajamarca Complex is found lying directly against the western edge of the Archaean Guyana Shield that extends from there throughout northern South America. The Cajamarca-Valdivia terrane, which is one of numerous accreted terranes in western Colombia, was accreted onto the western edge of the paleo - South American continent in Ordovician – Silurian time (Figure 7-2; Cediel et al. 2003).

Part of the extensive rift system responsible for the separation of North and South America in the Triassic and Jurassic extended through Colombia, Ecuador and Northern Peru (Jalliard et al., 1990; Kerr et al., 1997). Associated half grabens were filled with growth sequences of clastic sediments and volcanic material of dominantly andesitic composition. Evidence of igneous activity that accompanied this period of crustal extension in the Triassic to Jurassic is provided by the metaluminous I-type calc-alkaline Sonsón Batholith, which lies some 20 km west of the Berlin project. The Sonsón Batholith intruded rocks of the Cajamarca Complex.

An orogenic magmatic arc was developed along the eastern flank of the Central Cordillera at approximately 120 million years (“Ma”) (McCourt et al., 1984), with major intrusions of calc-alkaline affinity and compressional events at 112 ± 7 Ma (McCourt et al., 1984).

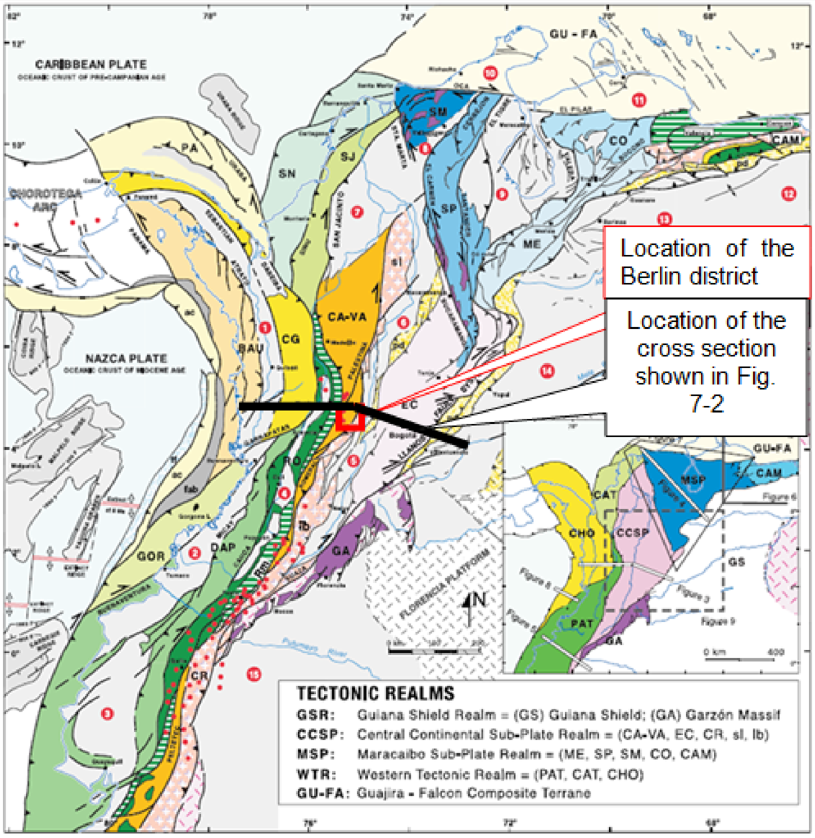
The sedimentary sequence that contains the mineralised unit at Berlin defines an upward-fining progression. The lower part of the stratigraphic sequence corresponds with alluvial fan facies that are interpreted to have formed against fault scarps during early phases of rift development. The sub-aerial fan facies grade upwards into finer-grained marine sands that are overlain by a limestone unit that passes upward into a black shale sequence which is several hundred metres thick. Fossil bivalves and gastropods in the limestones indicate a late Albian (Early Cretaceous) age and, together with ammonite fossils in the overlying black shale sequence, confirm a marine environment of deposition. This transgressive continental to marine sequence forms part of a large basin that extends from Colombia through Ecuador into Peru and the black shales constitute an important source for hydrocarbons in the region.

Cretaceous seafloor sequences of the Dagua-Pinon terrane were accreted onto the western edge of the Cajamarca-Valdivia terrane in the Aptian to Paleocene. This accretion was accompanied by intrusive activity, represented in the Berlin district by the Antioquia Batholith that has been dated at 90-58 Ma (middle to late Cretaceous; Cediel et al., 2003). It has a similar metaluminous, I-type, calc-alkaline composition to the Sonsón Batholith.

The Samaná Batholith, which is also mid- to late Cretaceous in age, is located immediately to the west of, and is intrusive into, the sedimentary sequence at Berlin.

During the Oligocene through to the Pliocene, several other terranes were accreted into the western seaboard of Colombia.

The Colombian Andes developed in response to roughly east-west shortening in the mid-Pleistocene. Related deformation in the Berlin area resulted in the formation of the syncline that hosts the mineralisation in the project area.

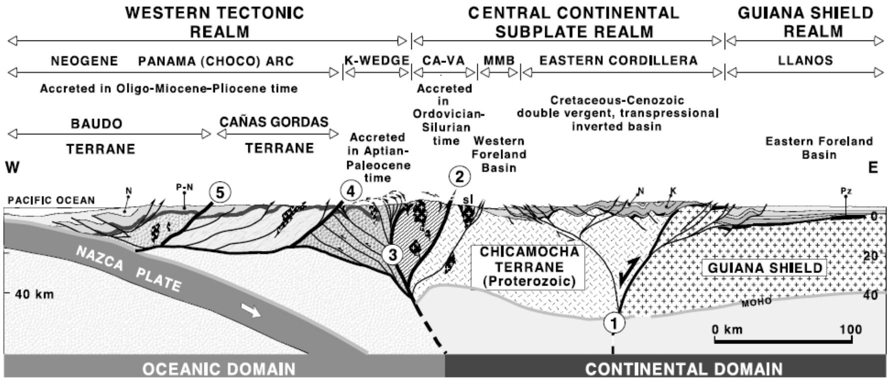


GS = GuianaShield; GA = Garzonmassif; SP = Santander massif – Serrania de Perija; ME = Sierra de Merida; SM = Sierra Nevada de Santa Marta; EC = Eastern Cordillera; CO = Carora basin; CR = Cordillera Real; CA-VA = Cajamarca-Valdivia terrane; sl = San Lucas block; ib = Ibagueblock; RO = Romeral terrane; DAP = Dagua-Piñon terrane; GOR = Gorgona terrane; CG = Cañas Gordas terrane; BAU = Baudoterrane; PA = Panamaterrane; SJ = San Jacinto terrane; SN = Sinuterrane; GU-FA – Guajira-Falconterrane; CAM – Caribbean Mountain terrane; Rm = Romeral melange; fab = forearcbasin; ac = accretionaryprism; tf = trenchfill; pd = piedmonte;

1 = Atrato (Choco) basin; 2 = Tumaco basin; 3 = Manabibasin; 4 = Cauca-Patiabasin; 5 = Upper Magdalena basin; 6 = Middle Magdalena basin; 7 = Lower Magdalena basin; 8 = Cesar-Rancheriabasin; 9 = Maracaibo basin; 10 = Guajira basin; 11 = Falconbasin; 12 = Guaricobasin; 13 = Barinas basin; 14 = Llanos basin; 15 = Putumayo-Napo basin; red dot = Pliocene-Pleistocenevolcano.

(Cediel et al, 2003)

Figure 7-1: Main tectonic components of Colombia (after Cediel et al, 2003)



Regional context of the Berlin Project area that lies adjacent to the Palestina Fault system (2) on the east flank of the Cordillera Central (from Cediel et al., 2003).

Principal sutures: 1 = Grenville (Orinoco) Santa Marta – Bucaramanga – Sauza faults; 2 = Ordovician-Silurian Palestina fault system; 3 = Aptian Romeral-Peltetec fault system; 4 = Oligocene-Miocene Garrapatas-Dabeiba fault system; 5 = late Miocene Atrato fault system.

Abbreviations: K-wedge = Cretaceous wedge; CA-VA = Cajamarca-Valdivia terrane; MMB = Middle Magdalena Basin; sl = San Lucas block; (Meta-)Sedimentary rocks: pz = Palaeozoic; K = Cretaceous; P = Paleogene; N = Neogene.

Figure 7-2: Cross Section through the Colombian Cordilleras showing the regional context of the Berlin Project (after Cediel et al, 2003)

7.2 Stratigraphy

7.2.1 Metamorphic Basement

The metamorphic basement in the vicinity of the Berlin Project is assigned to the Cajamarca Complex (Maya & González, 1995), which correlates with units known by other names such as Cajamarca Group (Nelson, 1957; Nunez et al., 1979), metamorphic rocks of the Cordillera Central (Feininger et al., 1972), Cajamarca Terrain (Etayo-Serna et al., 1986) and Tahamí Terrain (Toussaint & Restrepo, 1988) (Figure 7-3). The Cajamarca Complex, which consists of greenschist-grade metamorphic rocks that had sedimentary and igneous protoliths (Nelson, 1957), is bounded by the Otú-Pericos Fault on the eastern flank of the Cordillera Central and the Jerónimo Fault in the Cauca River Valley on the west.

Basement rocks in the Berlin Project area consist of quartz–sericitic schist, graphitic schist, slate and quartzite, locally with disseminated pyrite.

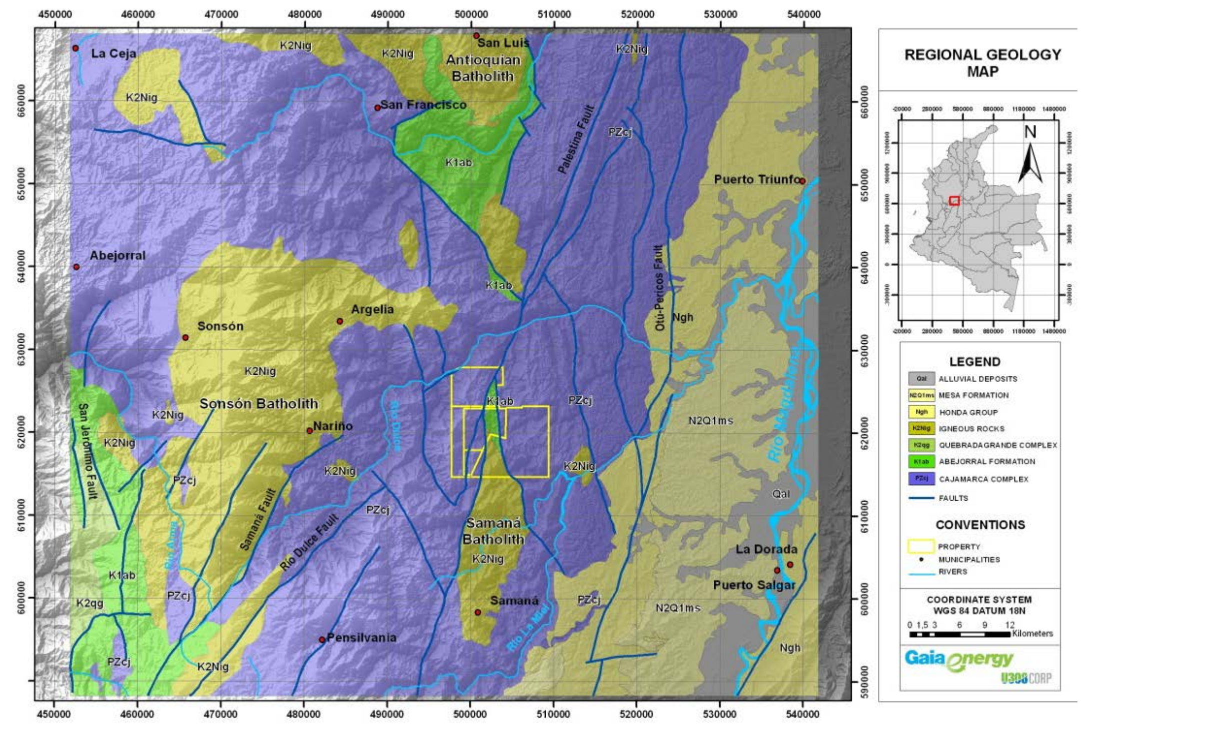


Figure 7-3: Regional geological setting of the Berlin Project. Berlin Project concessions are shown in yellow

7.2.2 Abejorral Formation

The Abejorral Formation (Bürgl and Radelli, 1962) is a Cretaceous sequence that is preserved in outliers in Caldas Province, and in the adjacent Antioquia Province to the north. This formation is discordant over the rocks of the Cajamarca Complex and has a faulted contact with the Valle Alto Formation. Facies sequence investigation by González (1980) in the provinces of Antioquia and Caldas led to the interpretation of a shallow continental shelf depositional environment with local euxinic conditions. The development of facies deposited in transitional and shallow shelf and external shelf environments occurred in the Early Cretaceous (Etayo-Serna et al. 2003). Based on ammonites, González (1980) concluded that this formation is Late Aptian – Middle Albian in age. On a regional basis, the clastic component of the Abejorral Formation would correlate with the Caballos and Hollin formations, while the limestone and black shale sequence would correlate with the Simiti and overlying Villeta formations of Pindell and Tabbutt (1995).

Mineralisation at Berlin occurs in limestone facies that occur immediately beneath the black shale sequence that was correlated with the Abejorral Formation by Bürgl and Radelli (1962).

7.2.3 Honda Group

The Honda Group, which occupies an elongate area that trends north along the Magdelena River, was defined by Hettner (1982). The Honda basin is located to the east of the Berlín Project and consists mainly of intercalated red sandstones, mudstones and polymict conglomerates (Barrero and Vesga, 1976). It lies unconformably on Cretaceous sedimentary rocks. Guerrero (1993) concluded that the Honda Group is Middle Miocene in age and interpreted it to have accumulated in a braided and meandering fluvial environment.

7.2.4 Mesa Formation

The Mesa Formation is broadly upward-fining, with conglomerates with clasts of mainly volcanic rocks at its base, overlain by tuffaceous sandstone, lapilli deposits and tuffs. It is discordant over the Honda Group and overlain by recent surficial deposits. Based on radiometric studies (Thouret, 1989) and fossil species (Dueñas and Castro, 1981), the Mesa Formation is Pliocene in age.

During the Quaternary, explosive volcanism resulted in volcaniclastic deposits covering a large part of the Cordillera Central and the Magdalena River Valley. These deposits are related to volcanoes associated with large terrain-bounding faults, such as the Palestina Fault. The active Ruiz Volcano, for example, lies adjacent to the Palestina Fault (Collins et al., 1981).

7.2.5 Recent Deposits

The cone-shaped volcanic vent that contains the San Diego Lake on the north-eastern margin of the Cretaceous sequence in the Berlin area is surrounded by an apron of lithic tuffs that have a polymict clast assemblage which reflects the underlying stratigraphy.

7.3 Sedimentary Facies Description and Analysis

7.3.1 Facies Sequence

Mineralisation at Berlin is made up of two types of host rock. Mineralisation near-surface is hosted in a sandstone unit, while at depth, mineralisation is confined to a carbonate unit (Figure 7-4). This distribution is ascribed to the sandstone host being a weathered version of the primary carbonate facies occurring at depth.

7.3.2 Arrangement of Unweathered Facies Sequence

The sedimentary sequence in which mineralisation at Berlin is located is made up of a number of distinct facies listed from bottom to top, or oldest to youngest, as follows (Figure 7-5):

- Unit A: conglomerates that grade upward into sandstone beds;
- Unit B: footwall carbonate – fossiliferous biomicrite to packed biomicrite (Folk classicification (Folk, 1959));
- Unit C: sparse biomicrite – this is the main mineralised horizon;
- Unit D: laminated carbonaceous mudstone; and
- Unit E: black mudstone.

The two limestone facies and the overlying Unit D laminated mudstone are remarkably consistent across the area in which the resource was estimated. These three units, which are so closely tied to mineralisation throughout the resource area, each constitute marker beds of remarkable continuity. The five facies groups are described in detail below.

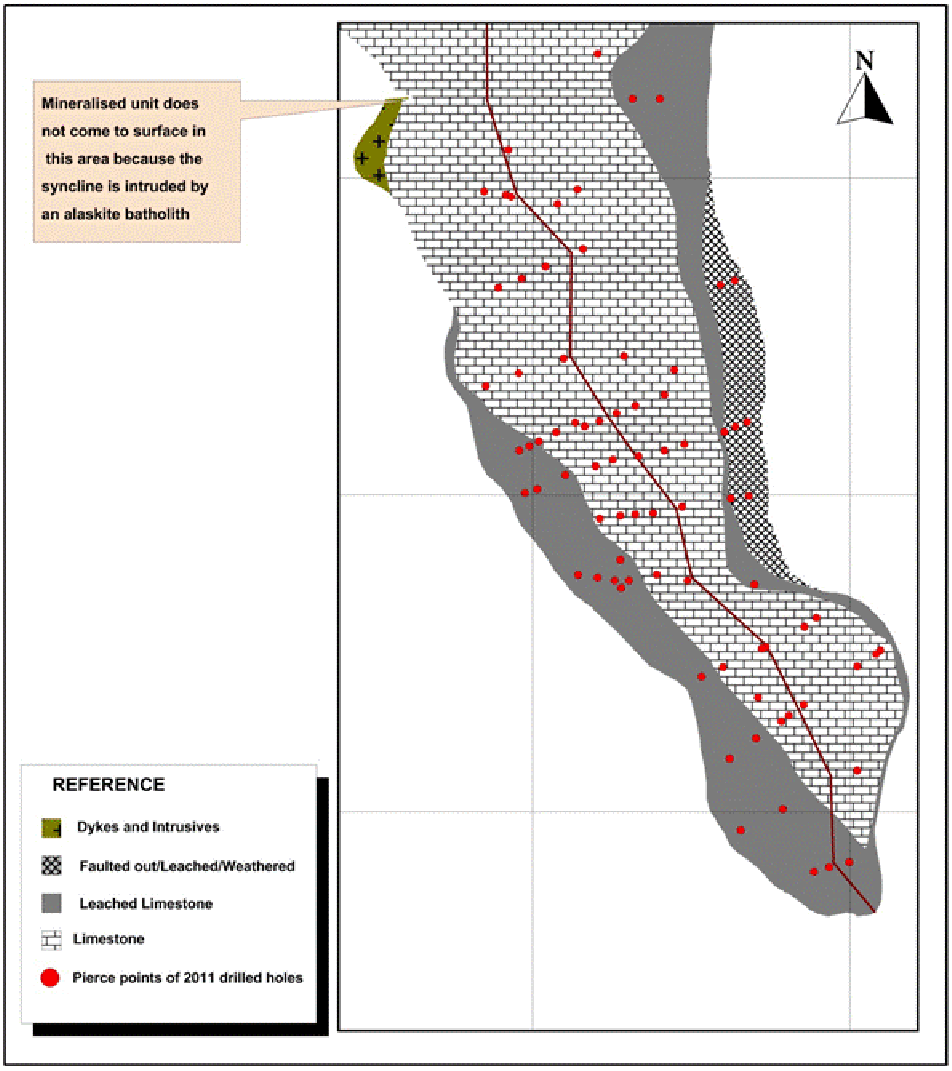


Figure 7-4: Map, in which the "U"-shaped mineralised unit has been unfolded into a flat layer, showing the distribution of sandstone- and carbonate-hosted mineralisation

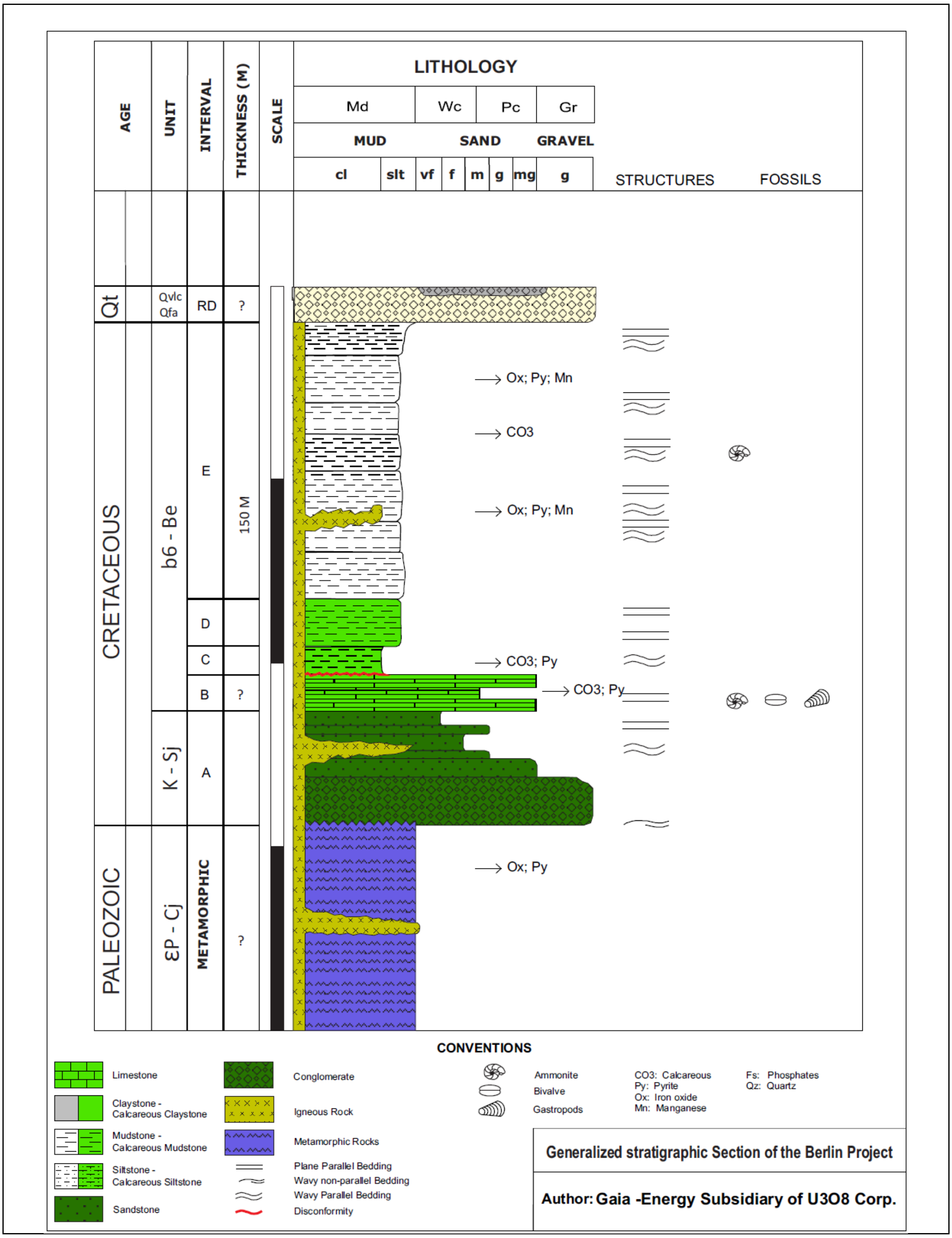


Figure 7-5: Detailed stratigraphic column defined from drill core from the Berlin Project

7.3.2.1 Unit A: Clastic Sequence

The basal unit is a crudely bedded, dominantly clast-supported conglomerate with a maximum thickness of 20 m. The clast assemblage consists of metamorphic and igneous rocks. Clasts are sub-rounded and are poorly sorted. The matrix consists of coarse sand- to granule-sized material.

Conglomerate facies pass upward and laterally along strike into coarse lithic arenite that fines upward into medium- to coarse-grained arenite that constitutes a unit up to 13 m thick. The arenite consists of tabular and lenticular beds that vary in thickness from a few tens of centimetres to a few metres. Basal units of the sandstone are made up of poorly sorted, subangular grains with low sphericity, whereas the upper units are better sorted and have a higher proportion of quartz grains to lithic grains. Primary sedimentary structures include cross-bedding, parallel lamination and slightly wavy lamination. Sandstone facies are intercalated with tabular beds of siliceous siltstone in the upper parts of the sandstone unit.

7.3.2.2 Unit B: Footwall Carbonate

The lowermost part of the carbonate sequence, Unit B, contains a fossil assemblage dominated by gastropods and cephalopods. Unit B typically has an erosional contact with the underlying sandstones and constitutes a unit 2 m-6 m thick. Cobble-sized bioclasts are up to 60 centimetres ("cm") in diameter and make up between 30% and 60% of the rock. Several types of clast occur, including fossil-bearing fine-grained carbonate facies, carbonate mudstones and coarse granular carbonate facies. Many clasts contain soft-sediment deformation features in which flame-like structures of matrix embay the margins of the clasts or where tongue-like protrusions from the clast project into the matrix. The matrix that supports the clasts varies from massive mud (micrite and fossiliferous biomicrite) to granular material with shells and shell fragments (packed biomicrite) (Figure 7-6).

The top of Unit B is marked by an inconspicuous erosional surface that is overlain by Unit C.

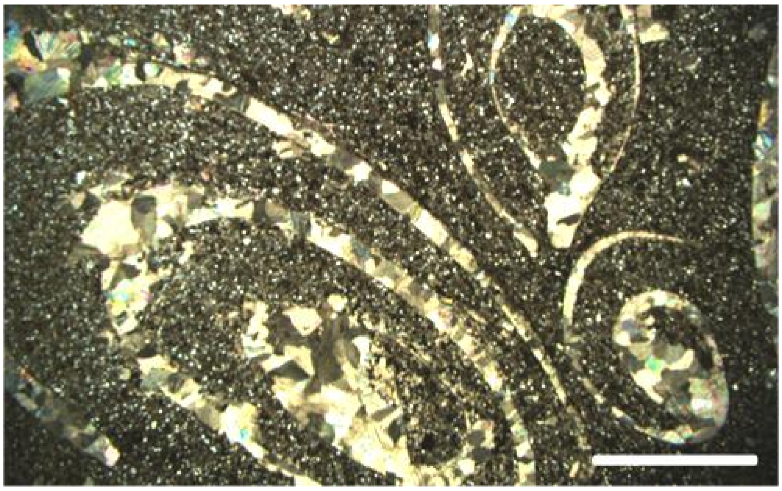


Figure 7-6: Cross-polarised transmitted light photomicrograph illustrating Unit B (scale bar 3 mm), showing packed biomicrite with fossil cephalopod shells

7.3.2.3 *Unit C: Sparse Biomicrite*

Unit C consists of sparse biomicrite with abundant fossil shell fragments that are relatively uniform in size although shell size does typically fine upward through most drill intercepts. Shell fragments in the basal part of the unit seldom exceed 2 cm in length, while those towards the top of the unit are generally a few millimetres long and tend to be thinner than those at the base. Some intercepts show that Unit C consists of several stacked beds in which shell fragment size decreases upward. The fossil fragments consist mainly of brachiopods and mollusk shells. Unit C has subtle plane and slightly wavy lamination and varies between 20 cm and 9 m thick with an average thickness of 2.5 m.

Review of this facies in thin section shows that the biogenic material is arranged in foliated and prismatic microstructures (Nichols, 2009), (Figure 7-7). Unit C is phosphate-rich and also has a high organic carbon content. The carbon ranges from amorphous bitumen to material that shows an incipient graphitic crystal structure.

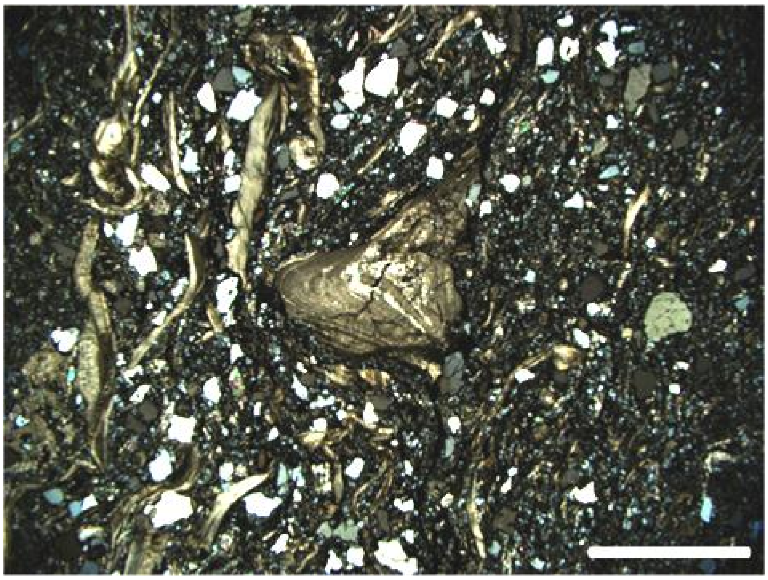


Figure 7-7: Cross-polarised transmitted light photomicrograph illustrating Unit C – sparse biomicrite (scale bar 3 mm) - with numerous fossil fragments

7.3.2.4 *Unit D: Laminated Carbonaceous Sandstone*

Unit D has a gradational contact with the underlying Unit C and the overlying black mudstone Unit E. Unit D averages 4 m in thickness.

Unit D is a plane- to slightly wavy-laminated carbonaceous mudstone interlayered with fine sand or silt arranged in alternating pale and dark laminae. Some intersections show that Unit D consists of stacked, upward-fining units that range in thickness from 5 cm to 40 cm. The more conspicuous upward fining units have a very fine sand at the base, that grades upward into siltstone or mudstone. In hand specimen, Unit D appears devoid of fossils, although petrographic study shows that it does contain fossils. Petrographic examination shows that the rock is largely clastic in nature, consisting of a mixture of fine- to very fine quartz and calcite grains with bioclasts, cemented by carbonate (Figure 7-8).

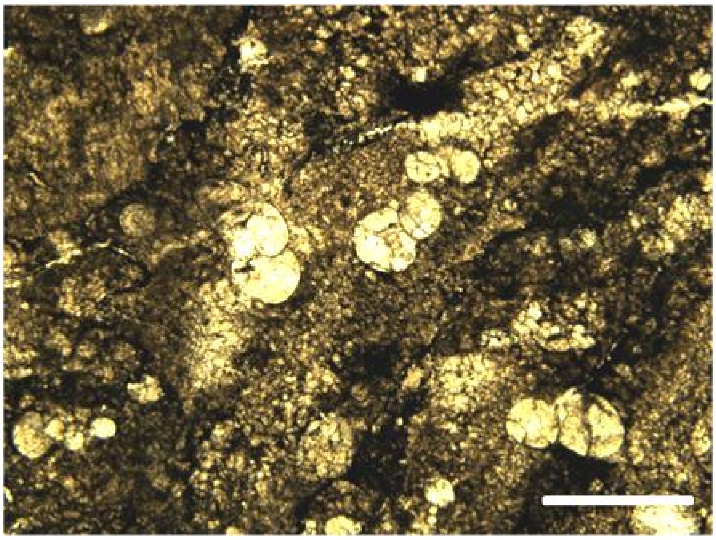


Figure 7-8: Plane-polarised transmitted light photomicrograph illustrating microscopic bioclasts within Unit D (scale bar 0.15 mm)

Carbonate facies have not been found in surface outcrops, nor in trenches, nor in shallow drilling. Near-surface mineralization is contained in sandstones and intercalated tan-coloured clay-rich beds that lie immediately beneath Unit E, the black shale facies. The sandstone is typically calcareous and contains variable amounts of interstitial bituminous hydrocarbon. The intercalated clay-rich and sandy layers fine upward over about 30 cm into a laminated mudstone-siltstone unit in which uranium grades typically drop to several tens of parts per million ("ppm") U_3O_8 over about a metre from the upper contact of the sandstone. The thickness of the mineralised zone varies between 1 m and 6.7 m and averages approximately 2.3 m thick in trenches that have been channel sampled.

7.3.2.5 Unit E: Organic-Rich Black Shale / Mudstone Facies

Mudstone facies constitute a monotonous unit up to 600 m thick in the Berlin project. Parts of the sequence show conspicuous bedding defined by paler basal silty facies that fine upward into mudstone. These upward-fining units are typically a few decimeters thick. This unit appears largely fossil-free except for widely scattered ammonite shells. Thin sections, however, show that the mudstone contains well-sorted bioclasts.

Descriptions of the hydrocarbon content of Unit E are provided in Section 7.6.

7.3.3 Description of the Weathered Sequence

Unit A, as described in Section 7.3.2.1, is virtually unchanged in the weathered part of the sequence. Units B and C cannot be distinguished from one another in the weathered environment – appearing as a single unit of interlayered sandstone and siltstone beds. Unit D, the black shale, is weathered to a pale ochre colour that extends from surface for a few tens of centimetres up to a few tens of metres. Units B and C, in contrast, are weathered to a depth of more than 100 m in the southern part of the Berlin syncline.

Unit B is recognisable in some bore hole intersections by the presence of relict shell fragments in the sandstone unit that forms the immediate footwall of the mineralised zone. These shell casts are of the same type as the fossils found in the carbonate rock that constitutes Unit B in intersections from beneath the weathered horizon (Figure 7-9).

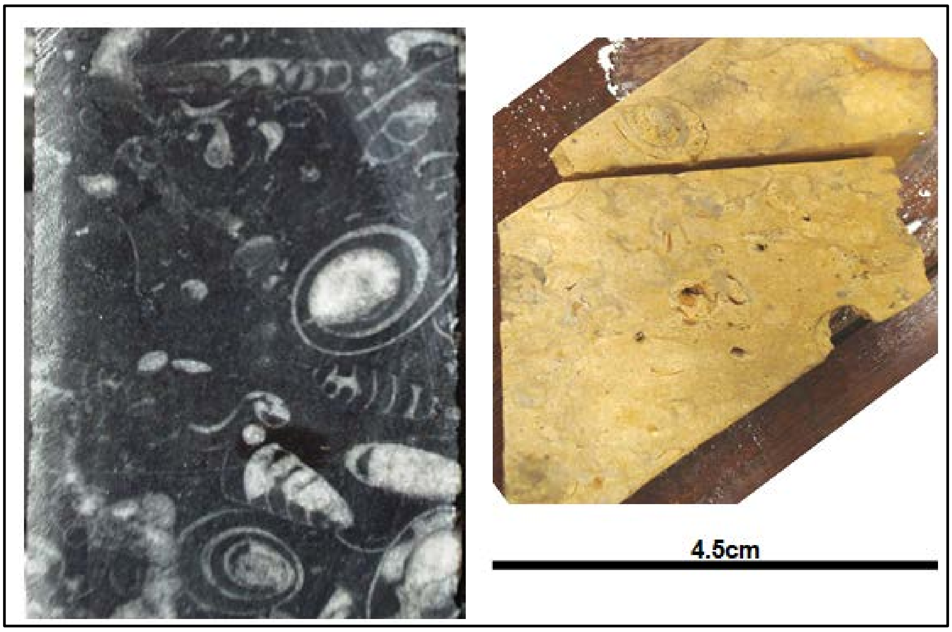


Figure 7-9: Comparison of fossil shell fragments in unweathered carbonate constituting Unit B with shell casts of the same fossils in “sandstone” in the weathered environment

7.3.4 Lateral Facies Distribution

The two limestone facies and the overlying Unit D laminated mudstone are remarkably consistent across the area in which the resource was estimated. These three units, which are so closely tied to mineralization throughout the resource area, each constitute marker beds of remarkable continuity, recognizable over a strike distance of 6.3 km (Figure 7-10).

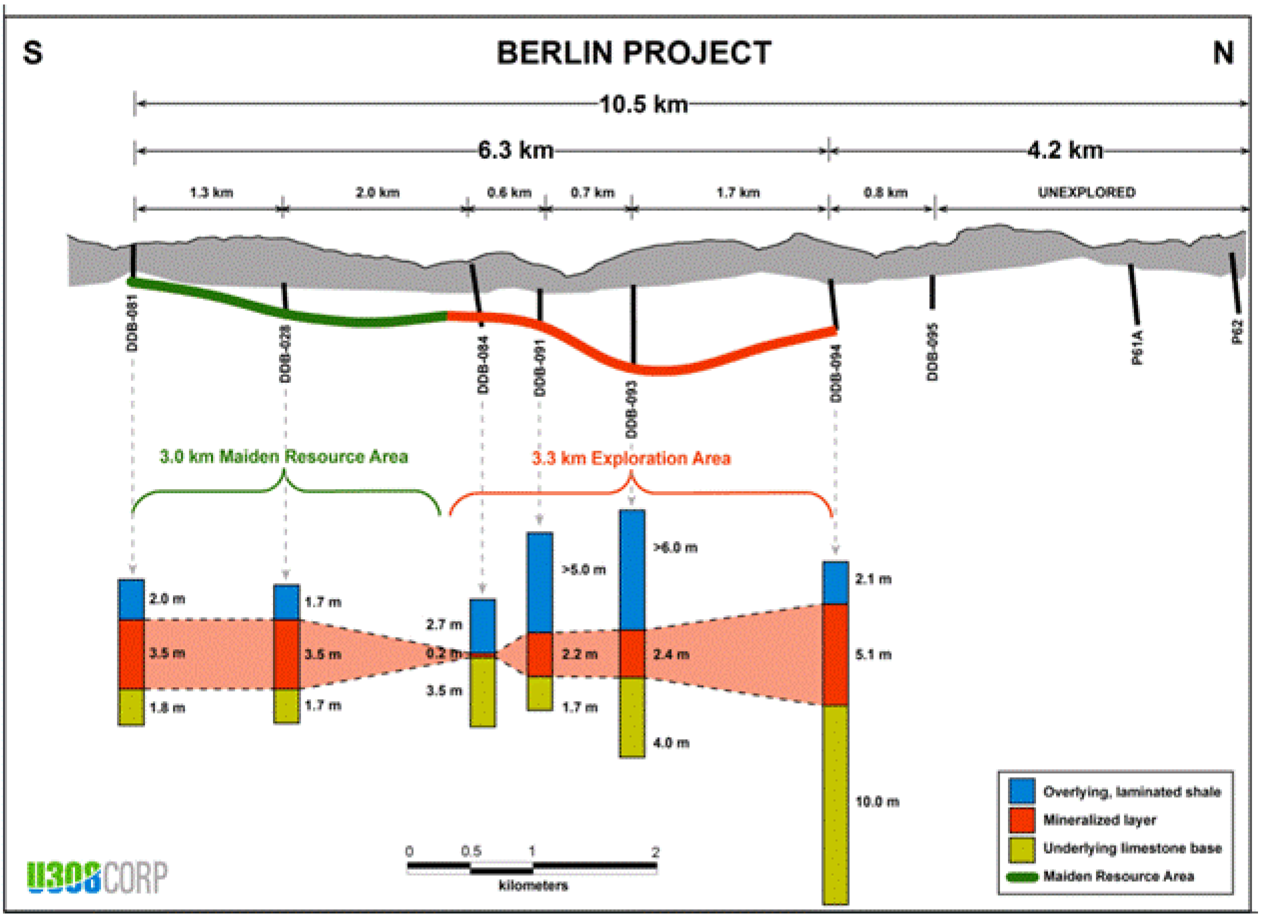


Figure 7-10: Long section along the Berlin trend showing the consistency of mineralised Unit C-D (in orange) sandwiched between underlying Unit B (yellow) and the overlying black shales (Unit E, in blue)

7.3.5 Facies Sequence Interpretation

Basal conglomerates of Unit A are interpreted to represent alluvial fans that accumulated where rivers draining a dominantly metamorphic basement discharged into a basin formed adjacent to a fault scarp. The upward-fining nature of the clastic sequence suggests that the fault scarps became less pronounced with time, or retreated, such that the streams carried finer sand as opposed to a pebble load. The presence of limestone facies with marine fauna overlying the sandstones is consistent with a marine transgression that drowned the fluvial system. The nature of Unit B of the limestone facies is consistent with mass flow deposits, with large fossil-bearing clasts and boulders supported in a limestone mud. Such mass flow facies, coupled with cephalopod (squid family) fossils is consistent with a shelf or shelf-break environment. Unit B could be interpreted as the basal mass-flow component of a Bouma sequence (Bouma, 1962).

Furthermore, the upward-fining arrangement of carbonate, laminated carbonaceous mudstone and the black shale mudstone facies, together with their plane – and slightly wavy – laminated nature (Units C, D and parts of E) are reminiscent of stacked turbiditic sequences. Such sequences typically develop in deep, tranquil subaqueous environments in which the parallel lamination is caused by mass flow from which each upward-fining unit is derived.

Units B, C and D weather into a rock that is devoid of carbonate and has a sandy to silty texture which gives it the appearance of sandstone that is interbedded with siltstone facies. It appears that carbonate has been removed from the original carbonate rock in the weathered layer by organic acids generated by the decay of organic material in the humic layer in the very high rainfall environment at Berlin.

7.4 Igneous Rocks

The mineralised sequence at Berlin lies between converging faults at the northern end of the Samaná Batholith (Figure 7-3). This igneous complex measures about 30 km north-south by approximately 8 km east-west. The rocks consist mainly of diorite and gabbro (~60% of the complex) with less extensive granodiorites, granites and tonalities (Muñoz, 1983). Barrero and Vesga (1976) obtained a K/Ar age of 119+/-10 Ma from hornblende in the Samaná Batholith (Barremian – Aptian). Field relationships suggest that an alaskitic component was emplaced late in the development of the igneous complex (Muñoz, 1983). A contact metamorphic aureole extends some 30 m to 150 m into enclosing sedimentary rocks. Igneous rocks of the complex have a homogenous texture with local development of a cleavage defined by the alignment of biotite plates. With the exception of the alaskite component, the rest of the complex is characterised by an abundance of xenoliths of gabbro and basalt. Recent drilling has shown that the Cretaceous rocks in the Berlin Project were intruded by alaskitic dykes and sills as well as granodiorites of Cenozoic age. The alaskite has an equigranular, faneritic, holocrystalline texture and is composed predominantly of plagioclase and quartz with minor biotite.

Two intrusive stocks lie near the eastern margin of the Berlin syncline where they intrude the Cretaceous sedimentary sequence. The stocks are mesocratic, porphyritic rocks that are made up of plagioclase, quartz and amphibole.

7.5 Structural Geology

The Berlin Project is located within the zone of influence of the Palestina Fault System that forms the western bounding structure to the Cretaceous sequence in the Berlin area. The fault strikes 010° to 020° and can be traced over a distance of more than 400 km, with evidence of activity during the Late Cretaceous and the Paleogene. Dextral displacement of about 28 km occurred during these times, with lesser displacement having occurred during the Neogene and Recent (Feininger, et al., 1972). The eastern margin of the Cretaceous sequence in the Berlin area is marked by the San Diego Fault, which is a north-striking splay that merges with the Palestina Fault near the northern tip of the Cretaceous sequence at Berlin.

The Cretaceous sedimentary sequence in which mineralisation in the Berlin Project occurs has been folded into a doubly-plunging syncline. Cross sections developed from field mapping and interpretation of drill intersections in the resource area show that the syncline is asymmetric: it has a steep to over-turned eastern limb dipping to the east, while the western limb is generally moderately inclined to the east (Figure 7-11 and Figure 7-12). The axis of the syncline is exposed at surface in the southern and northern parts of the property, consistent with the fold having the shape of the hull of a canoe. A north-south cross section shows a very open, flat-bottomed syncline with the southern limb inclined gently to the north and the northern limb inclined gently to the south.

A pervasive cleavage is strongly developed in the black shales; many of the cleavage surfaces are marked by lustrous, black carbonaceous material which petrographic studies show to be organic carbon that is bituminous to sub-graphitic in composition. Cleavage is dominantly east-dipping, being axial planar to the asymmetric syncline. West-dipping cleavage is locally developed in exposures of the west limb where it is associated with east-verging asymmetric folds that are antithetic to the regional syncline.

East-dipping faults have been mapped along the eastern margin of the syncline. In some areas, Palaeozoic schists are in faulted contact with the black mudstone, and kinematic indicators show an east over west sense of motion. This is consistent with west-verging thrust faults eliminating parts of the overturned stratigraphic sequence.

The asymmetric geometry of the Berlin syncline shows that the San Diego Fault system, while being subordinate to the Palestina Fault on a regional scale, exerted a stronger influence on the Berlin area than the Palestina Fault itself.

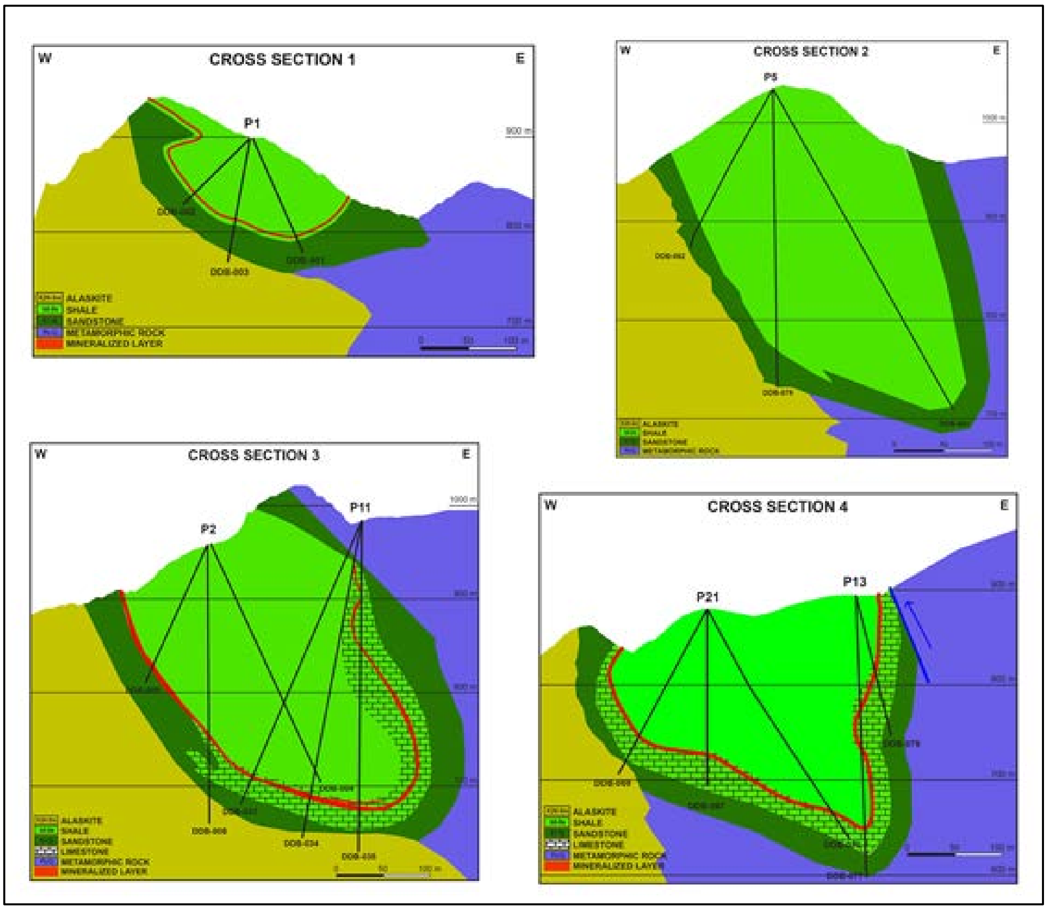


Figure 7-11: West-east cross sections (1 to 4) through Berlin syncline and plan view

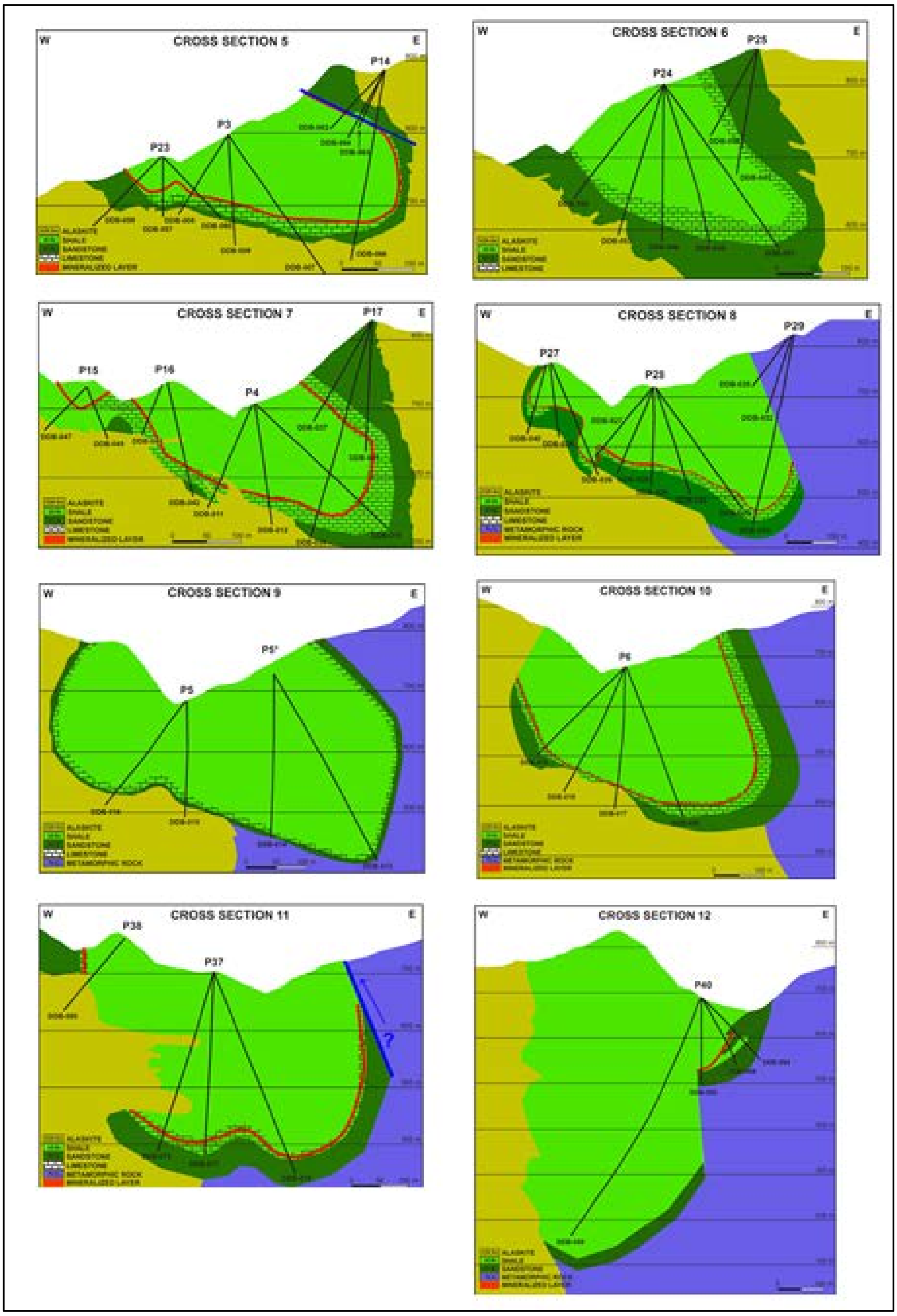


Figure 7-12: West-east cross sections (5 to 12) through the Berlin syncline

7.6 Mineralisation

7.6.1 Analytical Methods

Four petrographic studies of the mineralised unit have been commissioned by U3O8 Corp:

- Renaud (2010a, b and c) used polished thin sections for study by transmitted and reflected light with a petrographic microscope. Areas of interest were investigated using the energy dispersive system ("EDS") of a microprobe and specific mineral compositions were obtained using wavelength spectrometers. Backscatter electron detector images of relevant minerals and textural relationships were collected digitally.
- Royal Ontario Museum (2010) study of weathered mineralised material using a scanning electron microscope ("SEM") equipped with an EDS.
- The Australian Nuclear Science and Technology Organization ("ANSTO") study involved X-Ray diffraction ("XRD") to identify the mineralogical phases present with quantification undertaken using Siroquant. Major and minor mineralogical phases were also assessed using a SEM equipped with an EDS.
- Caceres Bottia (2012), as part of an M.Sc. thesis, used reflected and transmitted light, SEM and cathodoluminescence ("CL") in his study of textures and mineral compositions from the limestone-hosted mineralisation.

7.6.2 Distribution of Mineralisation

At depth in the Berlin syncline, uranium mineralisation has a very strong spatial relationship with Unit C, described in Section 7.6 above (Figure 7-13). Phosphate and most metals show a sharp decrease in grade at the base of Unit C, whereas grades decrease more gradually into the hanging wall. Vanadium, molybdenum, zinc and silver show a marked persistence of grades into the hanging wall. A similar distribution of mineralisation is apparent in the near-surface environment in which the carbonate appears to have been removed from the rock by weathering.

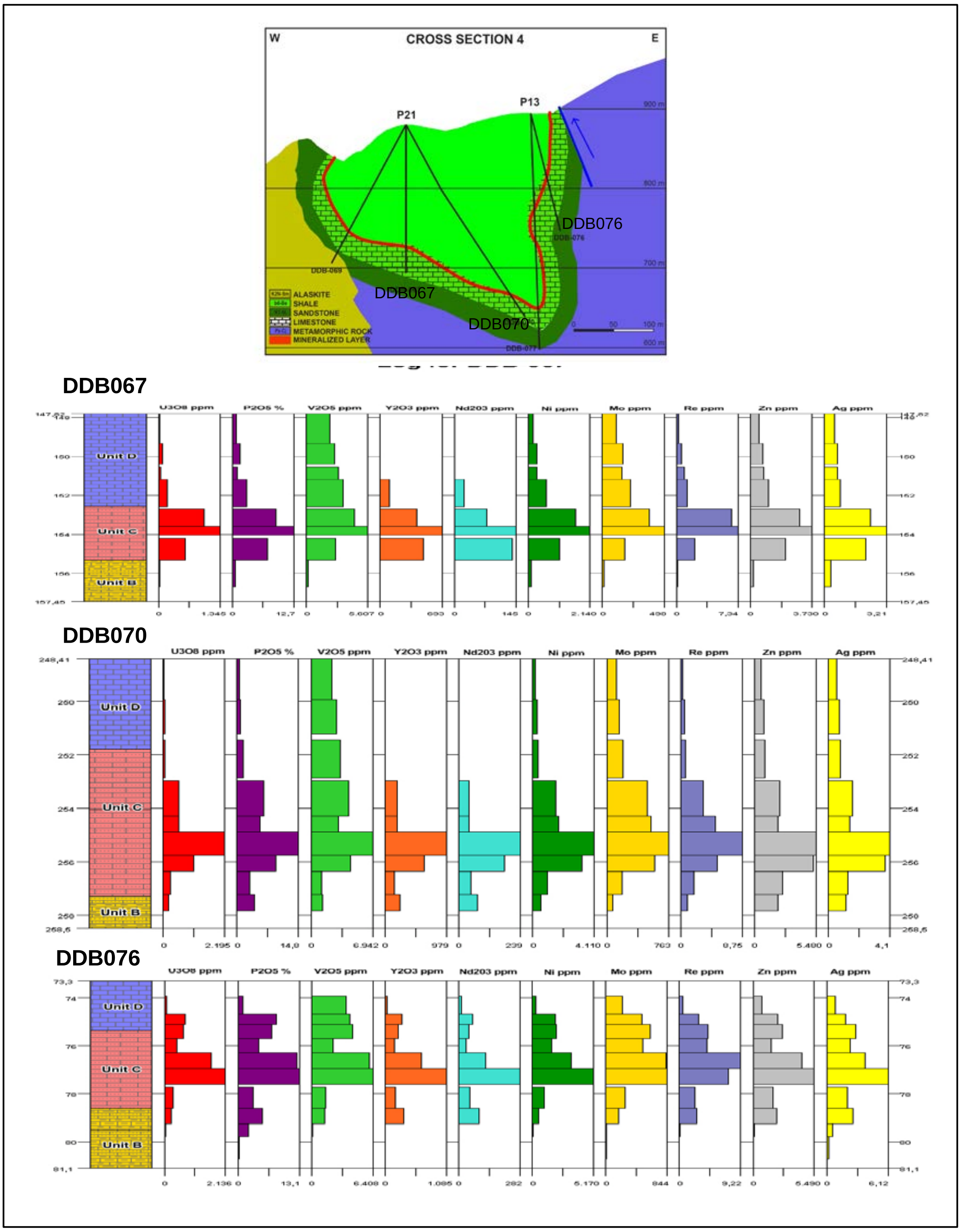


Figure 7-13: Histograms showing the distribution of metals in selected bore holes drilled in the resource area at Berlin

The lateral distribution of metals and phosphate shows a good correlation as indicated by grade-thickness values that are shown on a map in which the “U”-shaped mineralised layer has been unfolded into a flat sheet (Figure 7-14).

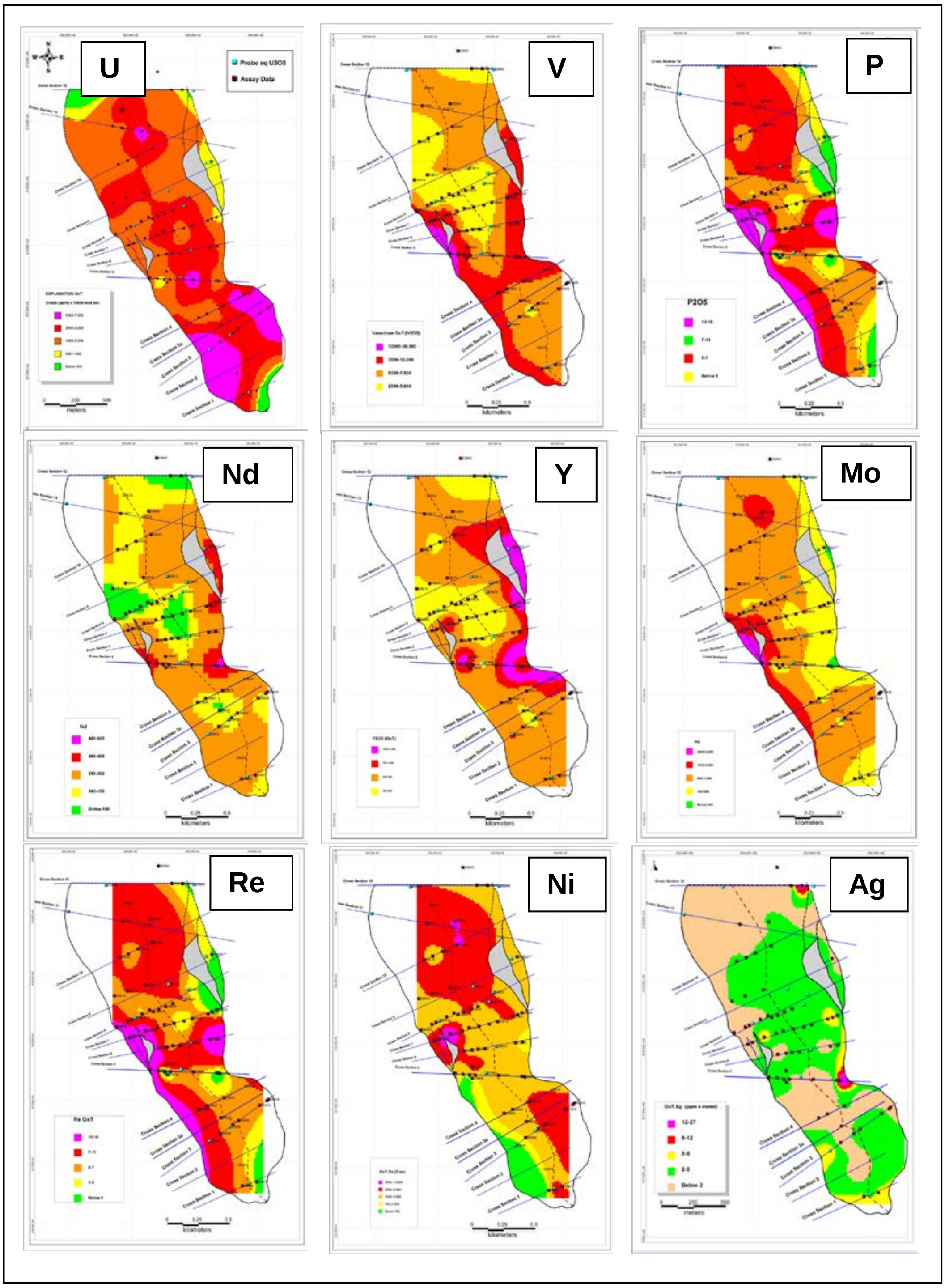


Figure 7-14: Map showing contoured values of grade-thickness for various elements from the resource area at Berlin. The "U"-shaped fold in which mineralisation occurs at Berlin has been unfolded and is illustrated as a flat sheet in this diagram

7.6.3 Petrography of the Mineralised Unit

7.6.3.1 Primary Mineralisation

7.6.3.1.1 Quartz and muscovite

Quartz grains and muscovite flakes, along with bioclasts are representative of the original constituents of this rock. Many quartz grains are embayed, a feature that is indicative of partial dissolution of the grains.

7.6.3.1.2 Calcite

Textural features show that there are two phases of calcite in the mineralized unit. Despite this, CL showed no compositional differences between the two types.

Stage 1 calcite replaces fossil shell fragments and also occurs as irregular patches of cement between adjacent grains (Figure 7-15a, b). Stage 2 calcite occurs as partial replacements of apatite (Figure 7-15c, Figure 7-16) as well as in flame-shaped embayments into bitumen (Figure 7-17c).

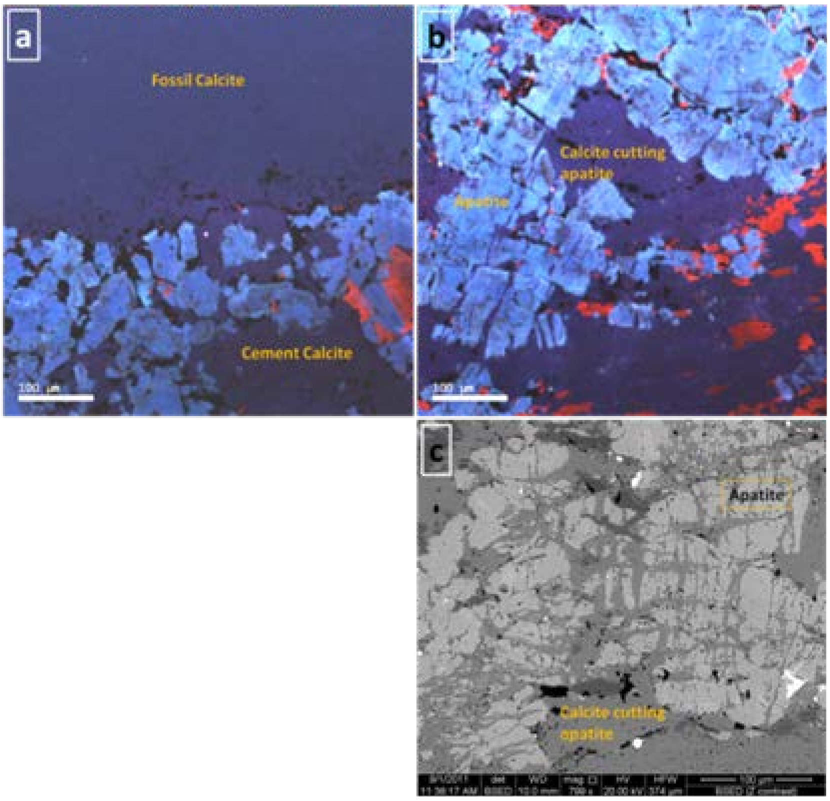


Figure 7-15: Colour cathodoluminescence imaging: calcite stage 1 (a) within fossil and in cement. Calcite stage 2 (b) cross-cutting apatite. Back-scattered electron image: calcite stage 2 (c) cutting apatite

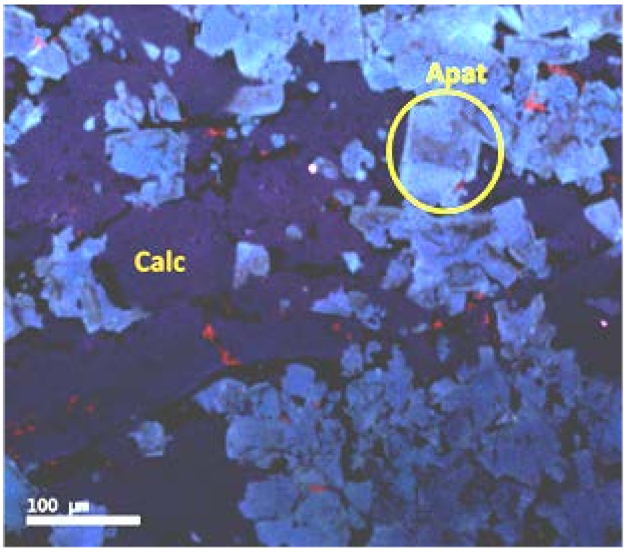


Figure 7-16: Colour cathodoluminescence imaging of apatite (Apat) and calcite (Calc). Compositional variations are shown by the darker versus brighter areas of apatite grains. Samples are from mineralised layer from drill cores of holes DDB16 and DDB18

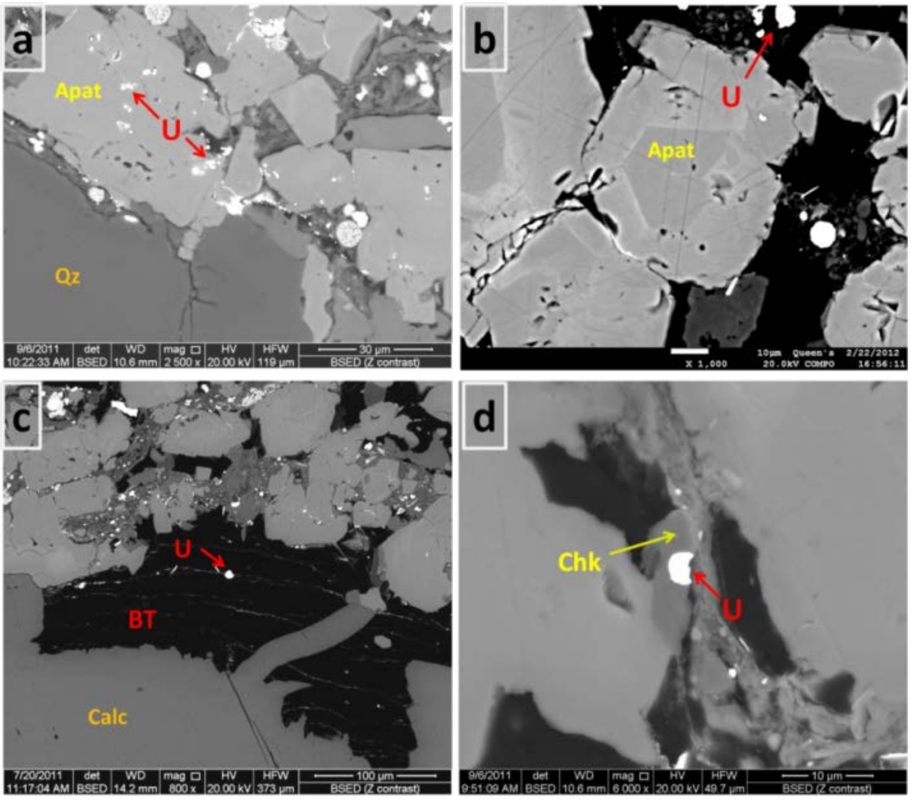


Figure 7-17: Back-scattered electron image: (a) textural relationships between uraninite and apatite (Apat); (b) Apatite zonation and increase of U concentration in the rim; (c) uraninite (U) within bitumen (BT) and bitumen being cross-cut by calcite; (d) uraninite and chernykhite (Chk)

7.6.3.1.3 Chernykhite

Chernykhite, a vanadium-barium mica, occurs as a partial replacement of muscovite laths (Figure 7-18) and is observed to be in contact with bitumen and uraninite (Figure 7-16d).

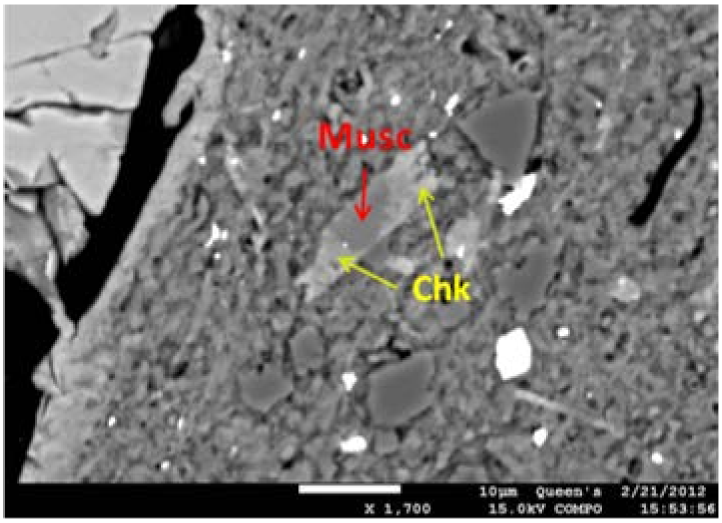


Figure 7-18: Back-scattered electron image: muscovite (Musc) replaced by chernykhite (Chk)

7.6.3.1.4 Apatite

Apatite occurs in both euhedral or subhedral crystals and irregular masses (Figure 7-17a, b). Many of the crystalline apatite exhibits compositional zoning around euhedral cores (Figure 7-16, Figure 7-17b).

7.6.3.1.5 Bitumen

Bitumen occurs as in interstitial filling, commonly in contact between apatite and calcite and it also crosscuts irregular masses of apatite (Figure 7-19). Some of the organic carbon has a subtly banded texture indicative of an incipient graphitic crystalline structure, and may be partially interlayered with amorphous organic matter of lower thermal maturation (Figure 7-19a).

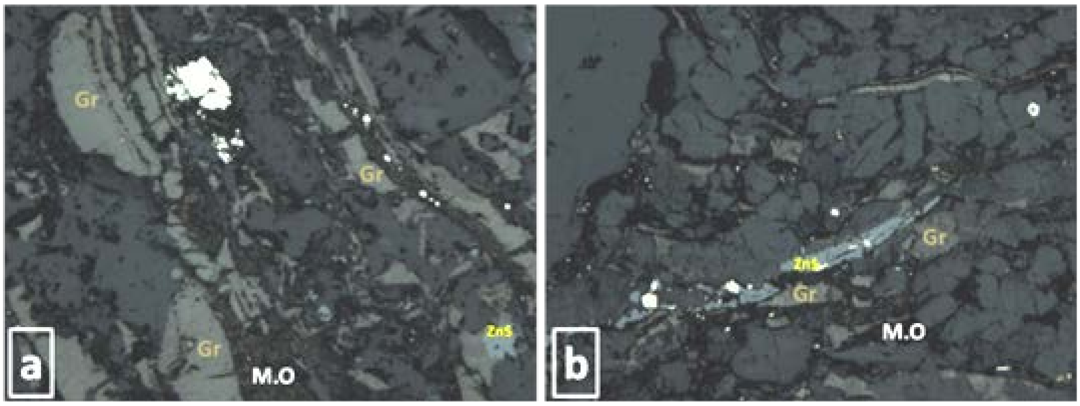


Figure 7-19: Graphite In Reflected Light Microscopy Images. Organic carbon occurs as amorphous masses in the plane of the cleavage, a) and also is matrix fill in microbreccias, b) Sphalerite follows the same oriented patterns as graphite. (Gr = Graphite; ZnS = Sphalerite; M.O = Bitumen)

7.6.3.1.6 Uranium Minerals

The only uranium mineral identified by Caceres Bottia (2012) is uraninite, although ANSTO reported rare occurrences of brannerite and coffinite in addition to the dominant uraninite. Uraninite occurs in a number of mineralogical associations as follows:

- as zones of enrichment, with yttrium, close to the exterior margins of some apatite crystals – which is consistent with the occurrence of uranium within the crystal lattice of the apatite (Figure 7-17b);
- intergrown within apatite crystals and concentrated at the margins of crystals (Figure 7-17a);
- within bitumen (Figure 7-17b and c); and
- in contact with chernykhite (Figure 7-17d).

Uraninite occurs as very small blebs that average less than 10 µm (Figure 7-20). An approximation of composition of the uraninite is provided in Table 7-1. This composition is approximate because the microprobe beam is likely to have overlapped the edge of the tiny uraninite blebs, thereby including some adjacent material in the analyses.

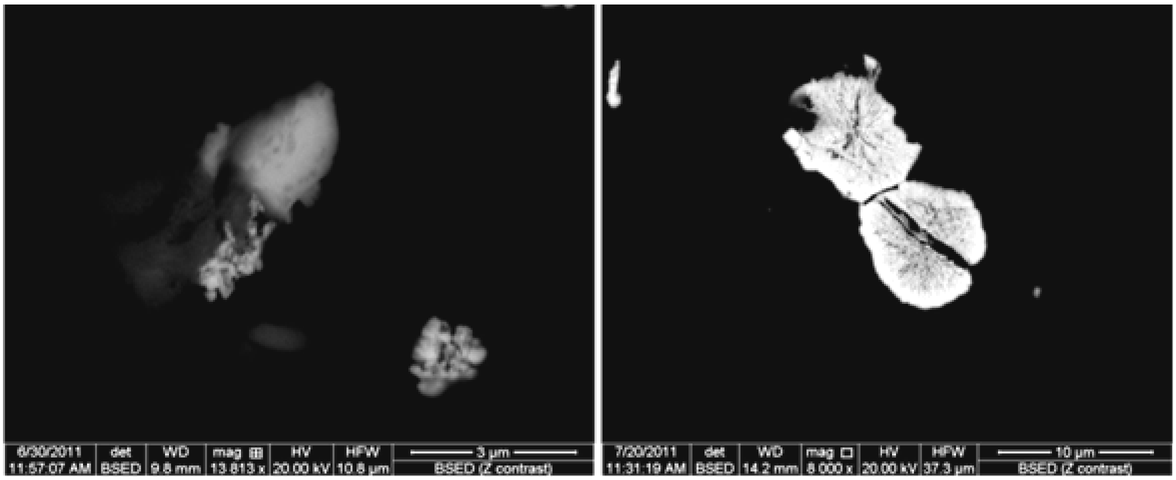


Figure 7-20: Back-scattered electron image of uraninite grains

Table 7-1: Electron-microprobe analyses and calculated chemical U-Pb ages for uraninite from the Berlin Project. Compositions are reported in %wt

	SiO_2	TiO_2	V_2O_3	Cr_2O_3	MnO	P_2O_5	FeO	UO_2	ThO_2	PbO	Y_2O_3	CaO	Total	Age (Ma)
11 uran 1	0.90	0.60	0.55	0.03	0.06	0.16	0.50	87	0.02	0.17	0.91	3.60	94.46	14.8
11 uran 2	1.08	0.69	1.09	0.05	0.07	1.13	0.54	80	0.00	0.00	0.74	4.42	90.07	0
11 uran 3	0.35	0.57	0.26	0.01	0.00	0.07	0.15	92	0.07	0.71	0.92	2.25	97.82	56.5
18 uran 1	0.62	0.58	0.43	0.05	0.10	0.39	0.68	88	0.02	0.13	0.72	2.76	94.84	11
18 uran 2	0.54	0.64	0.55	0.07	0.07	1.20	0.35	85	0.02	0.19	0.68	3.47	92.75	16.1
18 uran 3	0.86	0.66	0.68	0.10	0.09	1.01	0.53	81	0.00	0.11	0.67	2.43	88.84	9.9
16 uran 1	0.45	1.19	0.19	0.01	0.00	0.20	0.06	89	0.08	0.70	0.57	2.65	95.09	58.7
median	**0.62**	**0.64**	**0.55**	**0.05**	**0.07**	**0.39**	**0.50**	**86.94**	**0.02**	**0.17**	**0.72**	**2.76**	**94.46**	

7.6.3.1.7 Sulphides

Pyrite occurs in a number of forms including:

- Small, spherical framboids (Figure 7-21);
- Euhedral crystals within apatite crystals;
- In fractures in calcite where it occurs with chlorite, black isotropic apatite, quartz and sphalerite; and
- With zircon and monazite in calcite cement in calcite-quartz-apatite domains in carbonate facies.

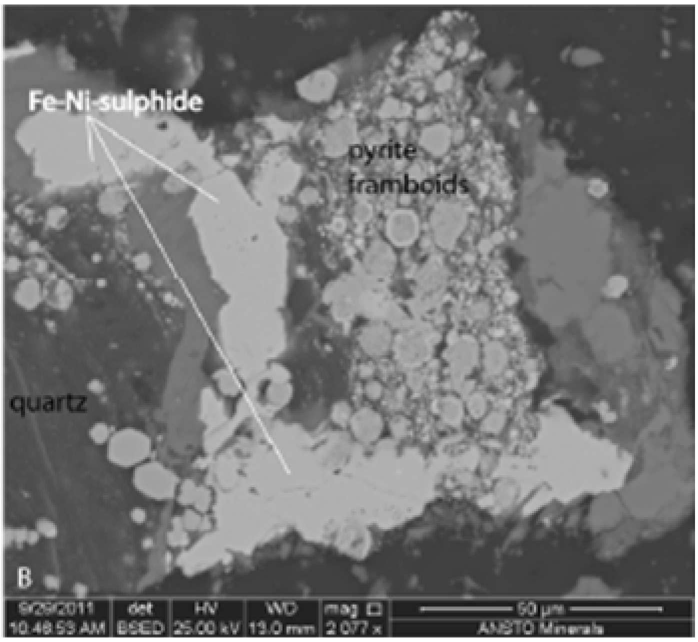


Figure 7-21: BSE image showing pentlandite (Fe-Ni sulphide) and framboidal pyrite in carbonate facies

Nickel-arsenic sulphides occur as subhedral crystals and as irregular masses as a partial replacement of sphalerite (Figure 7-22a). Sphalerite occurs in close association with bitumen, commonly forming bands that are parallel to subtle banding evident in the bitumen (Figure 7-22b).

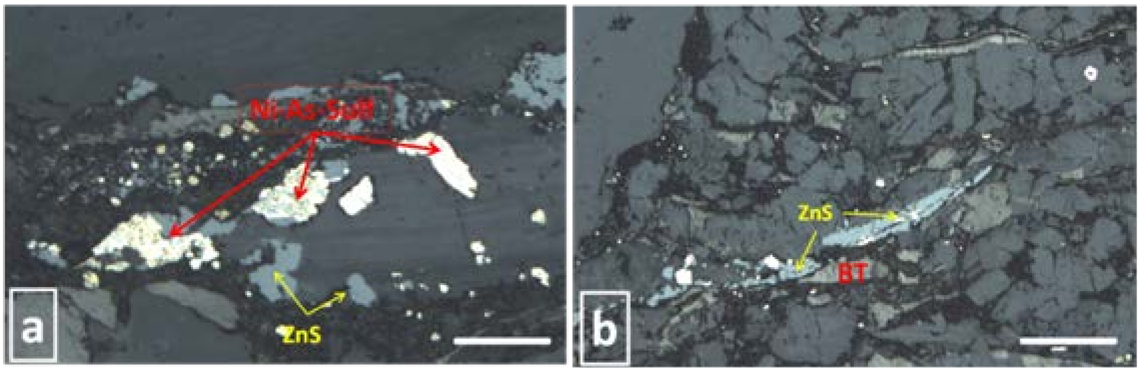


Figure 7-22: Plane-polarised reflected light: Sphalerite replaced by Ni-As sulfides (a); sphalerite (ZnS) following the pattern of bitumen (b) (scale bars, left 0.15 mm; right 0.3 mm)

7.6.3.1.8 Interpreted paragenesis

Quartz, muscovite and bioclasts are interpreted to be the only vestiges of the original rock composition. Mineral associations and cross-cutting relationships have been used to develop the paragenetic sequence illustrated in Table 7-2.

Table 7-2: Generalized paragenesis for the Berlin Project

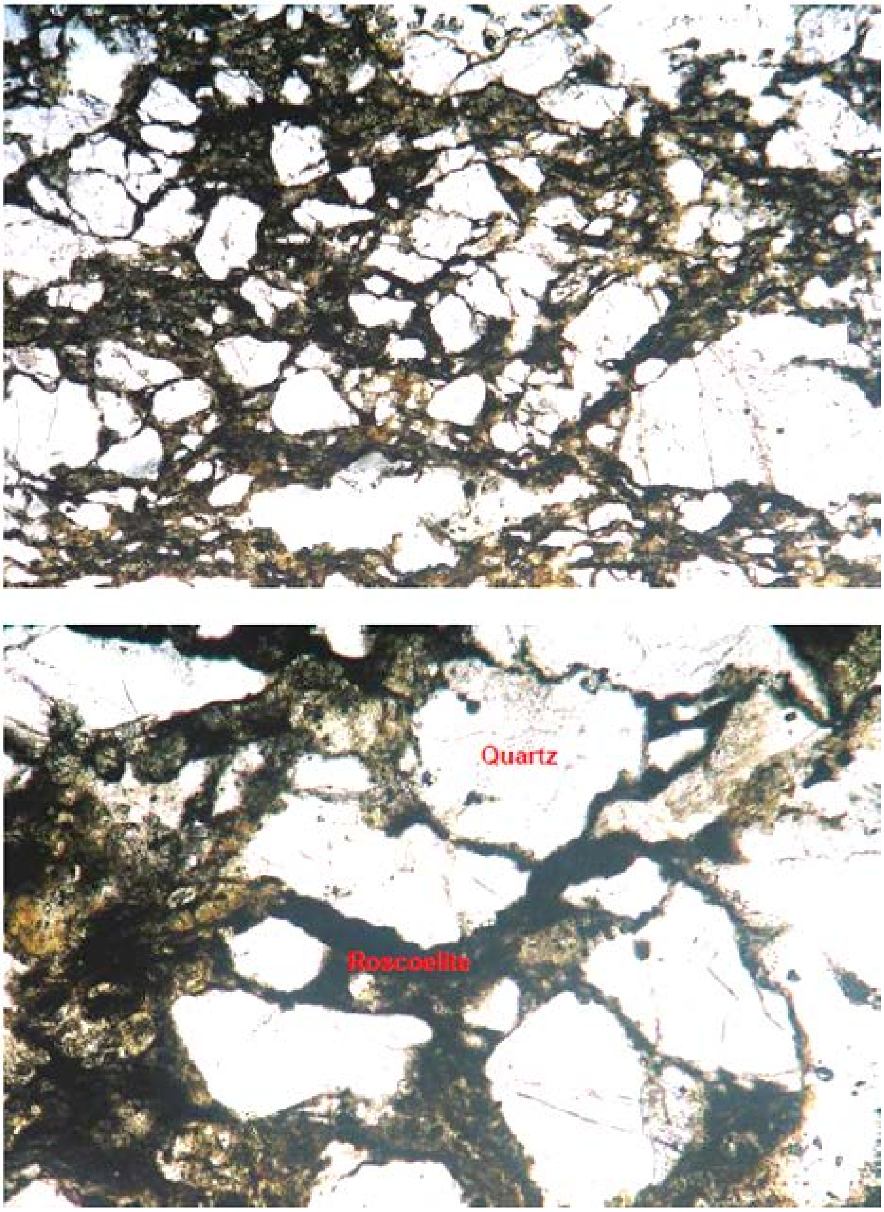
	Detrital minerals	Diagenesis (early → late)
Quartz	---------	
Moscovite	----	
Pyrite		-----------
Calcite Stage 1		--------
V-Ba rich mica (chernykhite)		-------------
Apatite		--------------
Bitumen		--------------
Uraninite		----------
Sphalerite		------
Calcite Stage 2		---------
As-Ni-S Minerals		-------
Calcite Vainlets		-----

7.6.3.2 Mineralisation in the weathered zone

Fracture-fillings in the sandstone near the southern closure of the synform have typically green to pale blue to orange, botryoidal coatings that have been identified as variscite ($AlPO_4.2H_2O$) and childrenite ($(Fe,Mn)AlPO_4(OH)_2.H_2O$) (ROM, 2010). These minerals typically develop in the weathering environment from the primary phosphate minerals, and occur within interstices in the sandstone. Similar fracture-fill secondary mineralisation extends up to 20 m into the sandstones that underlie the mature sandstone that hosts the stratiform mineralisation in some sectors of the syncline. The absence of similar footwall mineralisation in intercepts from beneath the weathered zone suggests that some redistribution of uranium may have occurred in the surficial environment.

Petrographic studies show that the sandstone samples taken from trenches are composed mainly of coarse quartz grains with minor magnetite and hematite, barite and minor chromite (Renaud, 2010a). Interstices are filled by apatite, Fe-Al-Ti-Cu-Ca-Cr-bearing phosphates, roscoelite (a V-Ba mica), Y-phosphates including churchite, monazite (REE-phosphate) and numerous U-bearing phosphates of the autunite ($Ca(UO_2)_2(PO_4)_2.10\text{-}12H_2O$) and meta-autunite subgroups (Renaud, 2010a). Mineralisation was observed to be parallel to bedding planes and planes of weakness in the rock.

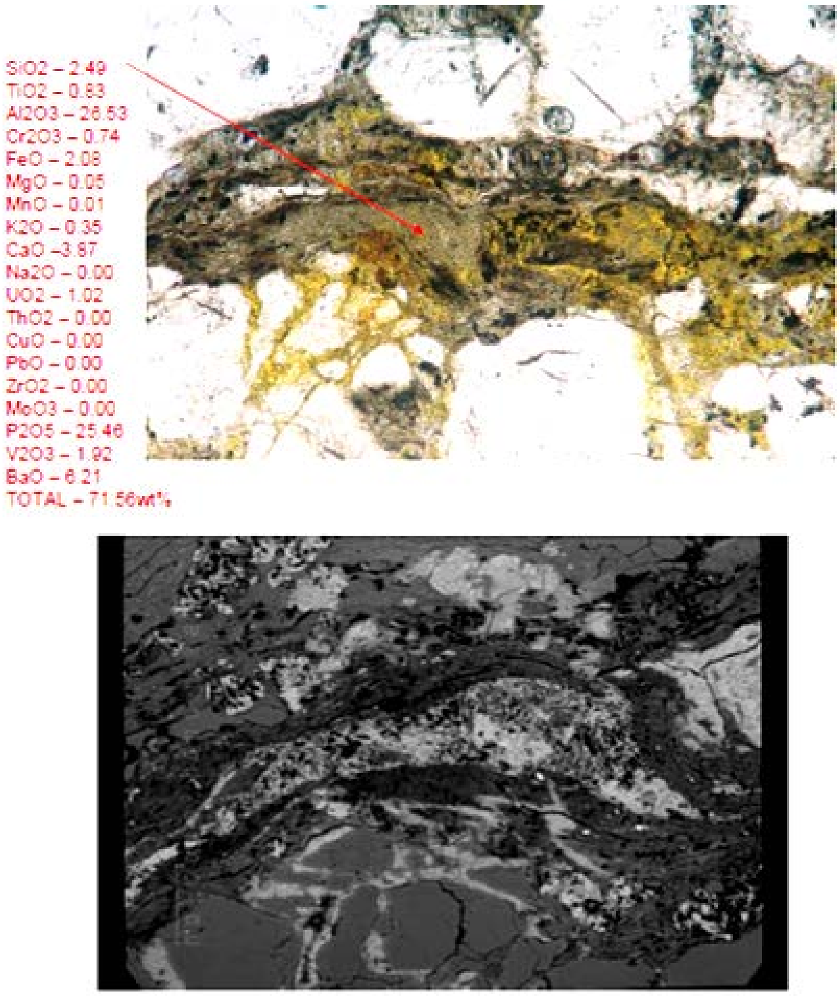
Some sandstones consist of micro - mosaic-brecciated quartz grains. These are breccias, that if the matrix was removed, the fragments would fit together like a jig-saw puzzle to reconstitute the original grains (Figure 7-23). The matrix between these fragments consists of fine-grained apatite, churchite, Fe-Al-phosphate +/- U, fine-grained zircon, Fe-Al-phosphate, roscoelite and Fe-Al+/-Ca-Ti-V-Cr-phosphate.



A common texture in these sandstones is mosaic-brecciation of the larger grains (the fragments fit together like a jigsaw puzzle) that are cemented with yellow cacoxenite and black-green V-Ba mica (Roscoelite).

Figure 7-23: Sandstone

In one weathered sandstone sample, the rock is dominated by quartz grains with interstitial matrix consisting of apatite, Ba-V-mica, and phosphates defined by alternating bands of black and yellow. The black bands are dominated by quartz with an interstitial matrix of apatite, Ba-V-mica, and minor Fe-Al-phosphates (cacoxenite). The interstitial apatites commonly contain fine grained REE-phosphate (monazite). The yellow bands are dominated by quartz with interstitial matrix dominated by apatite, cacoxenite and lesser V-bearing mica (Figure 7-24).



Photomicrographs in plane and crossed polarised light illustrating a complex domain of yellow cacoxenite intergrown with pale green Al-Ba-Fe-Si-Ca-U-phosphate. From Renaud, 2010a.

Figure 7-24: Sandstone photomicrographs in plane and crossed polarised light

Elongate quartz grains with yellow cacoxenite define the pervasive foliation in some samples. Cacoxenite in another sample of sandstone occurs in association with intergrowths of Fe-Al-V-phosphate and U-V-Al-Fe-Ca-phosphates (Renaud, 2010a).

Microscopic investigation shows an early generation of quartz veinlets is folded with primary layering (bedding), and is cross-cut by a later generation of stockwork veinlets that are not folded (Renaud, 2010a). This suggests that there were two phases of silicification of the sandstone – the early one being pre-folding, and the more pervasive one being post-folding.

In samples of the sandstone facies from drill holes DDB1 and DDB3, there are coarse clastic domains dominated by quartz with interstitial domains of mica-apatite-chlorite-pyrite-rutile-zircon in contact with more apatite-rich domains which seal finer-grained quartz, mica, iron-oxide, zircon, rutile, and a host of metals (Renaud, 2010c). These apatite-rich domains host such metals as silver-poor tetrahedrite, Ni-S (millerite), pyrite, sphalerite, and fine-grained U-Ti-bearing minerals.

Pyrophyllite was observed to be replacing mica and quartz grains in drill hole DDB3, where it constitutes up to 20% of the interstitial mineralogy between quartz grains. The pyrophyllite is occasionally associated with pyrite grains, but has not been noted with any other metals in these sections.

7.7 Summary

Uranium mineralisation is stratabound and is tightly confined to a carbonate unit that lies between sandstones in the footwall and mudstones in the hanging wall. Grades of all metals of potential economic interest and phosphate drop off sharply into the Unit B carbonate footwall. Uranium is closely associated with Unit C in the carbonate sequence, and grades decrease rapidly through Unit D into the hanging wall. Rare earths, yttrium and neodymium, have a similar distribution to uranium. In contrast, vanadium, nickel, molybdenum, zinc, silver, and to some extent, phosphate extend further into the hanging wall Unit E.

Limestone has been found neither at surface nor in shallow drilling. Mineralisation, however, occurs at the same stratigraphic position between the sandstone foot wall (Unit A) and the overlying black shales (Unit E). It appears that the carbonate has been leached by organic acids, leaving the clastic vestiges of the original rock hosting the mineralisation in the weathered zone.

Textural relationships indicate that apatite crystallized towards the middle of the paragenetic sequence and that uranium mineralisation is associated with organic carbon, most of which is strongly aligned in the pervasive cleavage of the host rock. The organic carbon is bituminous and is likely to have formed when the black shales passed through the oil window, reaching the temperature and pressure conditions conducive to hydrocarbon generation.

8 DEPOSIT TYPES

8.1 Berlin Deposit

Historic information led Spencer and Cleath (2010) to conclude that the mineralisation at Berlin was hosted by black shales and at that time it was classified as a deposit of this type, such as the Alum Shale in Sweden for example. Its relatively high grade was explained as being anomalous in that type of mineralised system. The nature of the intercepts of mineralisation encountered in U3O8 Corp's drilling of the deposit, as described in Section 7, indicate that the deposit is better described as stratabound limestone-hosted uranium-phosphate mineralisation. Mineralised intercepts from Berlin recently underwent extensive, detailed study for a Master of Science thesis at Queen's University in Ontario under the supervision of renowned uranium expert, Dr Kurt Keyser (Caceres Bottia, 2012). The conclusions drawn from the study as to the nature of the Berlin mineralisation are as follows (Caceres Bottia, 2012):

- Mineralisation is principally hosted in a permeable limestone unit located between an impermeable carbonate in the footwall and a largely impermeable, organic-rich black shale in the hanging wall;
- Organic carbon in the shale ranges in composition from bitumen to partially structured pseudographite. The shale reached this thermal maturity during a tectonic event that formed the pervasive cleavage or by intrusion of adjacent batholiths;
- Mineralisation occurred in the permeable unit where it lies in contact with the chemically reactive carbon-bearing black shale; and
- Uraninite, the principal uranium-bearing mineral, was dated directly at between 52-67 Ma. The host rocks are correlated, on the basis of their fossil assemblage, with the Aptian – Albian sequence (~100-120 Ma). The alaskitic intrusive that occurs in close proximity to the Berlin deposit has been dated at 60-80 Ma.

In addition to the composition of the host rocks at Berlin being different from those of typical black shale deposits, the mix and concentration of elements of potential economic interest are also significantly different. Black shale deposits are typically enriched in uranium, vanadium, molybdenum, nickel, zinc and manganese (Dahlkamp, 1993), while Berlin's suite of enriched elements is uranium, vanadium, phosphate, molybdenum, nickel, zinc and two rare earth elements, yttrium and neodymium.

8.2 Analogous Deposits

Deposits that are somewhat analogous to Berlin include the Santa Quiteria phosphate-uranium deposit in northeastern Brazil and the Nolans Bore deposit in Australia. The common factor in these deposits is the phosphate content of approximately 10% with associated uranium and rare earth mineralisation.

The Santa Quiteria Batholith forms part of a Neoproterozoic continental magmatic arc that contains several types of granitoid (Angeiras, 1988). The Santa Quiteria deposit lies within a metamorphosed carbonate-dominated sedimentary sequence that is composed of marble, calcsilicates, karstic dissolution breccias, carbonaceous breccias and collophanites (microcrystaline apatite) associated with mylonitised gneisses.

The deposit contains approximately 200 Mlb of uranium and the associated phosphate occurs in both stratiform bodies and in cross-cutting veins in a limestone host rock near alkaline intrusives. The reported non-CIM, non- NI 43-101 compliant resource is 80 Mt at 11% phosphate and 0.1% U_3O_8 (Porte, 2012). It is not known whether Santa Quiteria contains significant quantities of rare earth elements.

Nolans Bore is being developed principally as a rare earth deposit with numerous by-products including uranium and phosphate by Arafura Resources. The deposit is located near Alice Springs in the Northern Territory of Australia, at the southern end of the Reynolds Range in the central Aileron Province of the Arunta Region. The Aileron Province comprises greenschist to granulite facies metamorphic rocks that have been dated at 1,865-1,710 Ma.

The Nolans Bore deposit is composed of steep-dipping massive fluorapatite veins hosted by mylonitised gneissic granites. The deposit consists of sub-parallel sets of fluorapatite veins and stockworks of fluorapatite, allanite and carbonate veins with associated calc-silicate alteration. The deposit contains JORC-compliant resources of 4.3 Mt of 3.3% REO (rare earth oxide), 13% P_2O_5 and 0.026% U_3O_8 in Measured resources and 21 Mt of 2.6% REO, 12% P_2O_5 and 0.019% U_3O_8 (www.arafuraresources.co.au).

In terms of their association with carbonate rocks, phosphate and uranium, there is more similarity between Berlin, Santa Quiteria and Nolans Bore than there is between Berlin and the black shale deposits.

9 EXPLORATION

9.1 Responsibility for Exploration

All of the recent exploration described in this report has been done by employees and consultants of Gaia Energy (Colombia) Ltd, an indirectly held, wholly owned, subsidiary of U3O8 Corp. Only U3O8 Corp's assay data and drill results have been used for the resource estimate.

9.2 Approach

Due to the stratiform nature of the mineralisation at Berlin, the principal objective was to define the extent and consistency of the known mineralised layer through trenching and drilling. Exploration by U3O8 Corp. commenced in April, 2010.

9.3 Trenching

Trenches were excavated by hand on outcropping mineralisation on the flanks of the Berlin syncline. Trench sites were identified in two principal ways:

- With the use of geological maps made by Minatome in the late 1970's and early 1980's that indicated areas of outcropping mineralisation, coupled with the help of field assistants whom had worked for Minatome in its prior exploration of the area; and
- Reconnaissance transects made roughly perpendicular to the axis of the syncline in which outcrops and subcrops of mineralisation were identified from radioactivity detected with hand-held spectrometers.

Once the anomalously radioactive stratum was identified, the trench was cut perpendicular to strike. Sample locations were then defined based on lithology and levels of radioactivity. Radioactivity was measured with a hand-held GR 135 spectrometer. Sample collection was described in detail in a previous NI43-101 Technical Report (Coffey Mining, 2012)

To date, 38 trenches have been excavated. The majority of the trenches are located on the more accessible southern part and eastern flank of the syncline. Mineralisation has been shown to occur over a strike distance of 8.5 km. A summary of assay results obtained from the trenches is shown in Table 9-1: Figure 9-1 and Figure 9-2 display the location of the trenches.

9.4 Discussion

Exploration has been undertaken entirely through field work without support by airborne geophysics, for example. Given the continuity of mineralisation discussed in Section 7, the priority is to define where the mineralised layer may be have been interrupted by intrusions, such as the alaskitic intrusive bodies located on the western margin of the Berlin syncline, and by faults. Airborne magnetic and/or gravity surveys would assist in the definition of the margin of intrusive bodies at depth. An equally effective and significantly cheaper option would be magnetic traverses across the mapped contacts of the stocks and batholiths, supported by forward modelling of the form of the contact at depth with magnetic susceptibility values from bore hole core. Field-based gravity surveys would be slow and expensive due to the extreme topography of the area. Ultimately, drilling is the key exploration tool.

Table 9-1: Assay results from the mineralised intervals of trenches at a 0.4% U_3O_8 cut-off grade

Trench ID	Estimated True Thickness	U_3O_8 (%)	U_3O_8 (lb/st)	V_2O_5 (%)	P_2O_5 (%)	Mo (ppm)	Re (ppm)	Ag (ppm)	Ni (ppm)	Zn (ppm)	Y_2O_3 (ppm)	Nd_2O_3 (ppm)
TB-003	1.35	0.085	1.88	0.862	6.1	173	0.1	0.9	25	114	264	429
TB-001	1.34	0.114	2.51	1.066	3.9	769	0	1.7	13	284	135	216
TB-023	1.9	0.069	1.53	0.908	12.2	60	0	0.9	430	612	235	933
TB-002	2.32	0.172	3.79	0.887	4.9	140	0	1.6	37	40	205	853
TB-019	2.24	0.066	1.46	1.121	3	165	0	1.4	196	480	265	1341
TB-010	0.88	0.065	1.43	0.984	15.7	126	0	0.4	409	511	171	860
TB-011	0.99	0.039	0.85	0.821	3.6	16	0	0.7	57	115	353	1145
TB-009	2.64	0.134	2.95	0.828	14.6	146	0	0.4	43	85	237	840
TB-026	0.42	0.068	1.5	0.956	12.3	343	8	3.9	1873	2908	71	336
TB-025	0.88	0.081	1.79	1.011	13	216	0	0.5	13	6	249	925
TB-020	1.32	0.145	3.19	1.323	16.6	27	0.1	2.9	24	33	215	921
TB-000	0.73	0.114	2.51	0.995	2.3	142	0.1	2.3	432	712	245	1078
TB-004du	2.55	0.093	2.05	1.1	12.8	88	0	0.7	54	116	257	851
TB-027	5.47	0.11	2.41	0.855	17.4	238	0	1.9	783	2213	223	971
TB-004	1.68	0.083	1.83	1.075	3.7	172	0	0.3	278	570	241	1180
TB-016	1.53	0.321	7.06	1.382	25.9	1584	35.9	10.6	7063	3109	283	1252
TB-029	0.9	0.197	4.33	0.83	8.7	120	0.1	9.3	133	17	242	752
TB-028	2.72	0.134	2.94	0.464	6.7	153	0	1.2	33	138	163	683
TB-013	0.5	0.077	1.69	0.963	19.6	81	0.1	0.7	382	502	249	960
TB-006	1.86	0.1	2.21	0.701	13	40	0.1	1.3	19	39	197	621
TB-014	0.71	0.063	1.38	0.591	4.9	58	0.2	1.7	45	90	26	123
TB-012	2.88	0.081	1.79	0.406	3.7	216	0	3.6	42	80	127	454
TB-030	0.59	0.048	1.05	0.295	7.1	36	0.1	1	27	53	150	505
TB-031	0.92	0.05	1.09	0.632	10.1	102	0.1	1.1	22	29	125	448
TB-032	0.8	0.063	1.38	0.63	8	74	0	1.3	312	191	155	660
TB-032	0.59	0.05	1.09	0.42	2.3	38	0	10	324	129	21	133
TB-034	1.9	0.1	2.2	0.563	5.7	99	0.1	3.4	77	322	73	333
TB-036	0.7	0.035	0.77	0.63	0.9	141	0	9.8	362	415	0	0
TB-008	1.7	0.075	1.65	0.745	11.4	30	0	0.8	17	34	162	733
TB-005	2.01	0.136	2.99	0.568	12.2	105	0	0.9	32	74	176	637
TB-021	2.2	0.142	3.13	0.862	15.1	34	0.1	1.9	209	262	264	1016
TB-033	2.47	0.129	2.84	0.742	18.1	53	0.1	1.1	11	18	321	1110
TB-022	2.39	0.085	1.87	0.935	8.7	179	1.8	2.8	30	48	107	332
TB-018	1.95	0.064	1.41	0.684	12	38	0	0.4	50	73	249	755
TB-037	2.62	0.114	2.51	1.055	9.5	292	0	0.8	537	481	424	1871
TB-038	2.08	0.086	1.9	0.932	13.3	70	0	0.6	79	243	176	729

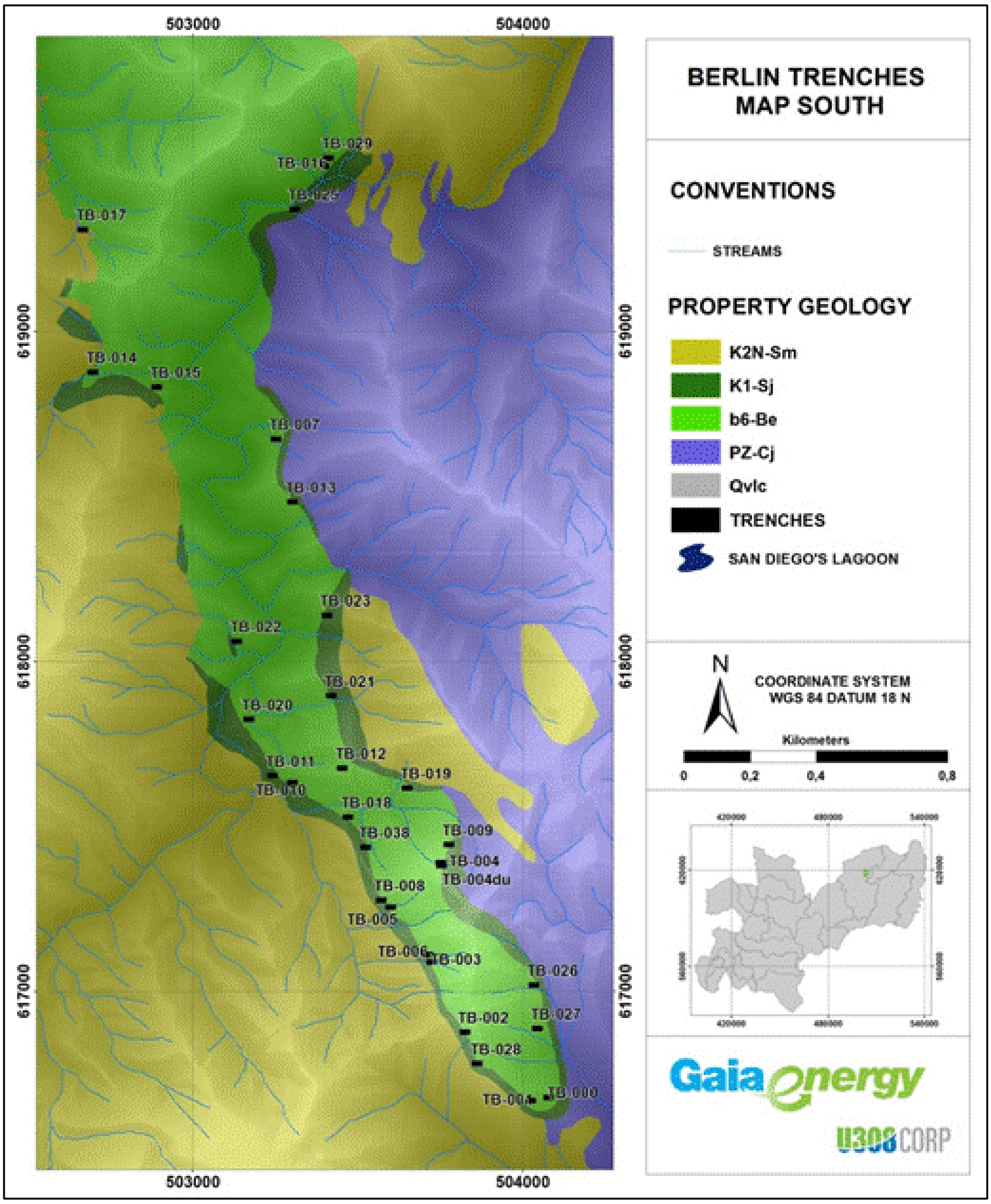


Figure 9-1: Geological map of the southern part of the Berlin syncline showing the location of trenches

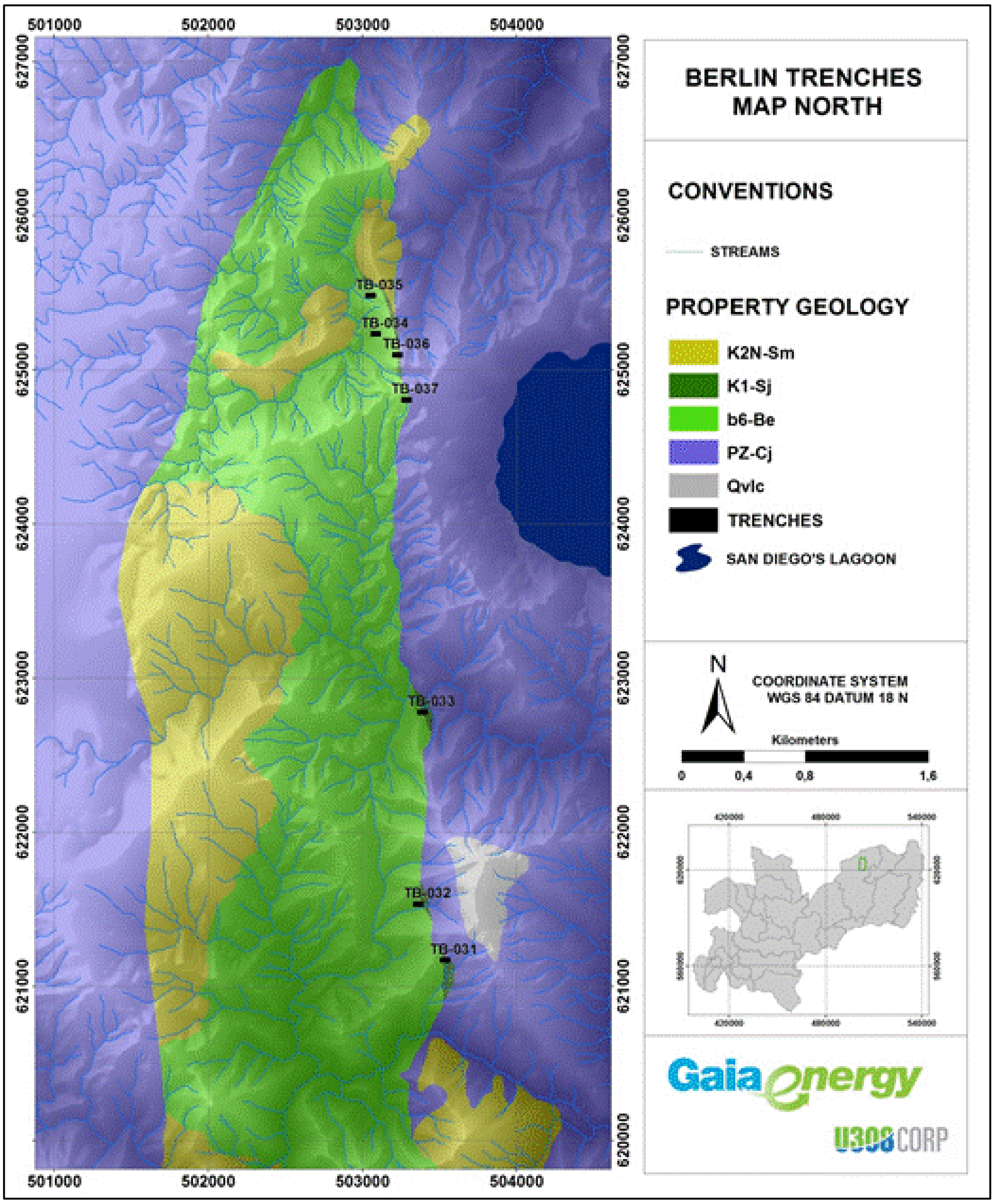


Figure 9-2: Geological map of northern part of the Berlin syncline showing the location of trenches

10 DRILLING

10.1 Drill Programs

Kluane Drilling Ltd. ('Kluane') of Whitehorse, Canada, has undertaken two drill campaigns at Berlin as follows:

- 2010-2011 program: One rig commenced drilling on October 15, 2010, and a second rig commenced on April 10, 2011. Both rigs left site on October 26, 2011. This was an exploration drill program followed immediately by infill drilling for resource estimation. The monthly drilling rates are summarised in Figure 10-1. A total of 82 bore holes were drilled for 18,523 m. The location of the platforms from which the holes were drilled is shown in Figure 10-2 and the list of holes drilled from each platform is shown in Table 10-1. A summary of assay data from the drill intercepts is shown in Table 10-2.

- The second campaign was between April and September, 2012, and focused on exploration of the area to the north of the resource area outlined in the 2010-2011 campaign (Figure 10-3) and a summary of assay data from the drill intercepts is listed in Table 10-3. Fifteen holes were drilled for a total of 6,511 m. Assay results for the 2012 phase of exploration drilling are summarised in Table 10-4.

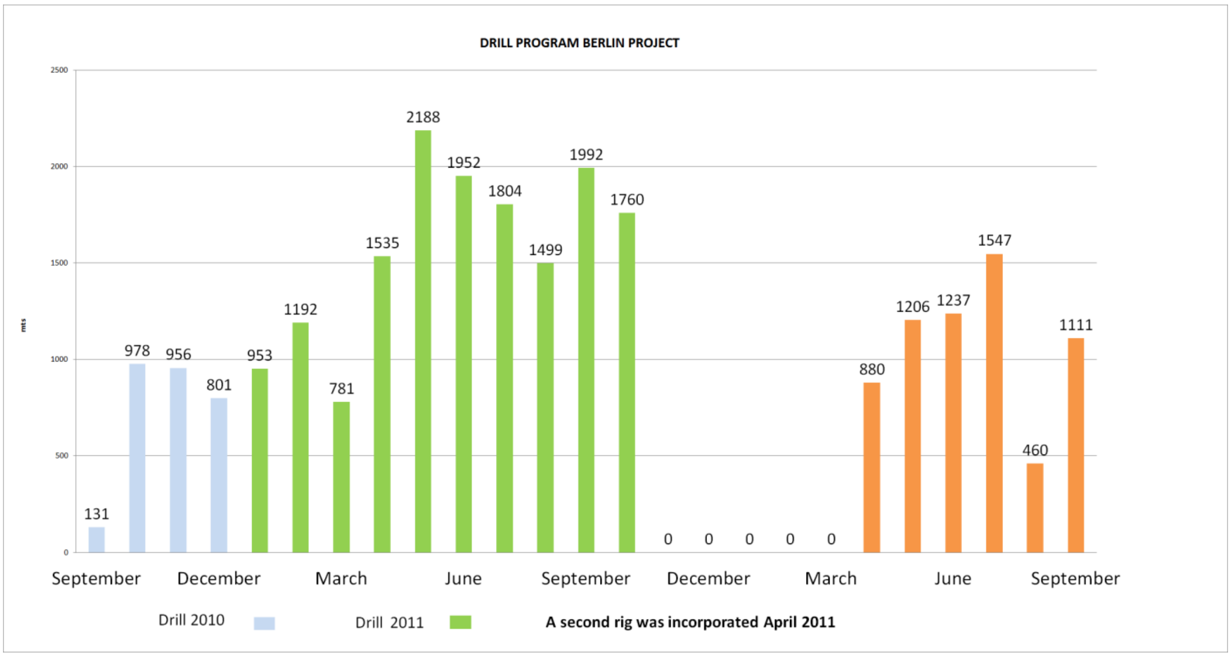


Figure 10-1: Chart showing metreage drilled by month in the two campaigns drilled at Berlin

Kluane used man-portable KDHT-1000, wireline rigs that the drill company has designed and manufactured in-house. Holes are collared with N-thinwall ("NTW") that has a core diameter of 57 mm, comparable with HQ at a 63.5 mm diameter and are typically reduced at depth to B-thinwall ("BTW") that has a core diameter of 42 mm, comparable with the 47.6 mm diameter of NQ core.

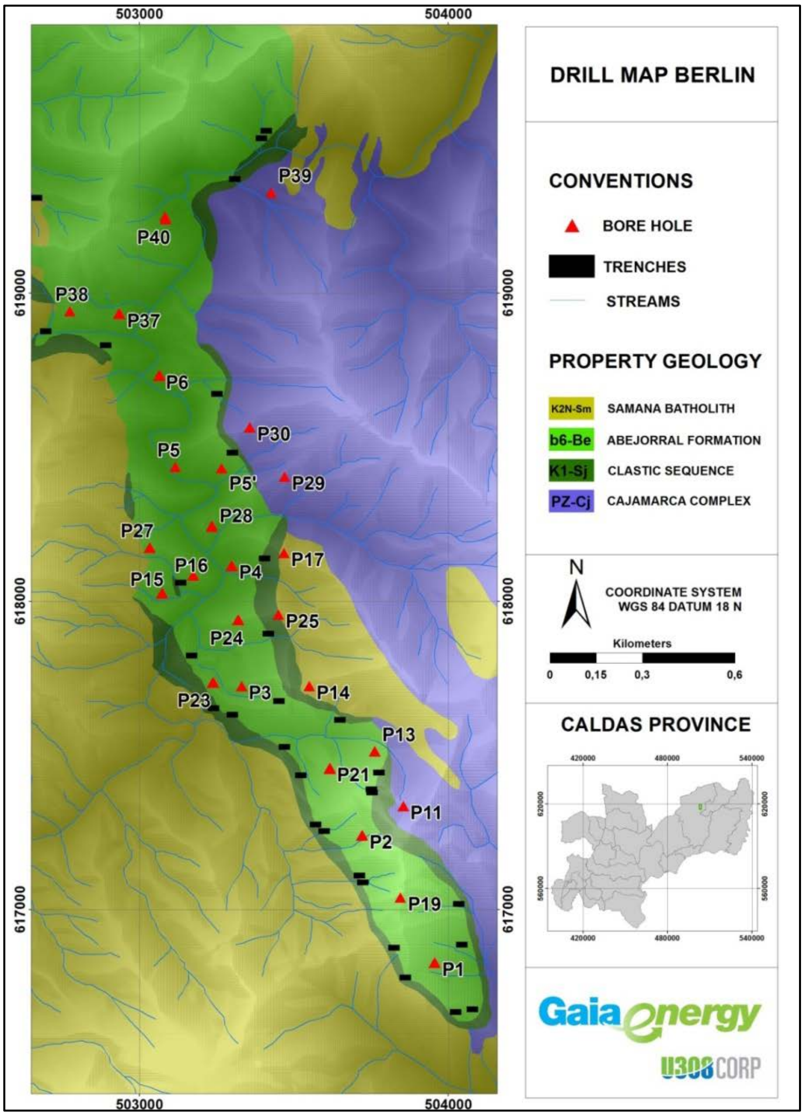


Figure 10-2: Map of the southern part of the Berlin syncline showing the location of platforms from which bore holes were collared

Table 10-1: Summary of bore hole length and drill platform number for bore holes drilled in the 2010-2011 campaign

Hole	Platform	Length	Hole	Platform	Length
DDB001	P1	131	DDB042	P16	176
DDB002	P1	101	DDB043	P25	178
DDB003	P1	133	DDB044	P15	88
DDB004	P2	282	DDB045	P15	83
DDB005	P2	163	DDB046	P25	138
DDB006	P2	300	DDB047	P15	91
DDB007	P3	238	DDB048	P24	213
DDB008	P3	131	DDB049	P24	225
DDB009	P3	151	DDB050	P39	255
DDB010	P4	271	DDB051	P24	275
DDB011	P4	166	DDB052	P40	191
DDB012	P4	179	DDB053	P24	215
DDB013	P5'	350	DDB054	P40	187
DDB014	P5'	271	DDB055	P24	197
DDB015	P5	194	DDB056	P40	167
DDB016	P5	223	DDB057	P23	82
DDB017	P6	286	DDB058	P40	600
DDB018	P6	250	DDB059	P23	131
DDB019	P6	280	DDB060	P23	115
DDB020	P6	323	DDB061	P23	150
DDB021	P30	131	DDB062	P14	113
DDB022	P30	151	DDB063	P14	90
DDB023	P30	304	DDB064	P14	108
DDB024	P30	196	DDB065	P40	265
DDB025	P28	207	DDB066	P14	268
DDB026	P28	192	DDB067	P21	201
DDB027	P28	186	DDB068	P40	462
DDB028	P28	230	DDB069	P21	197
DDB029	P28	305	DDB070	P21	285
DDB030	P29	144	DDB071	P37	325
DDB031	P11	331	DDB072	P21	268
DDB032	P29	180	DDB073	P37	382
DDB033	P29	402	DDB074	P21	247
DDB034	P11	346	DDB075	P37	332
DDB035	P11	355	DDB076	P13	152
DDB036	P27	253	DDB077	P13	297
DDB037	P17	173	DDB078	P37	431
DDB038	P27	155	DDB079	P19	304
DDB039	P17	331	DDB080	P38	156
DDB040	P27	155	DDB081	P19	375
DDB041	P17	197	DDB082	P19	186

Table 10-2: Assay results (at a 0.04% U_3O_8 cut-off grade) for intercepts from bore holes drilled in the 2010-2011 campaign

Plataform No	Bore Hole No	From (m)	To (m)	Estimated True Width (m)	U_3O_8 %	U_3O_8 lb/t	V_2O_5 %	P_2O_5 %	Mo ppm	Re ppm	Ag ppm	Ni ppm	Zn ppm	Y_2O_3 ppm	Nd_2O_3 ppm
P1	DDB001	109.7	111.2	1.5	0.079	17393	0.68	14.6	294	1.8	5.9	1140	2930	641	154
	DDB002	79.2	82.3	3.0	0.137	30185	0.76	9.1	360	5.1	5.9	940	171	700	173
	DDB003	80.8	83.8	3.1	0.124	27181	0.71	17.6	626	7.0	3.8	1809	6349	784	160
P2	DDB004	Bore Hole did not reach target depth													
	DDB005	138.7	146.3	7.6	0.152	33426	0.62	10.7	578	8.2	6.2	1462	365	662	177
	DDB006	199.7	201.2	1.5	0.046	10098	0.33	5.3	142	1.6	2.3	358	719	273	75
P3	DDB007	152.4	155.5	3.1	0.115	25316	0.45	8.9	632	7.3	2.2	1958	2922	386	85
	DDB008	91.4	93.0	1.5	0.068	15057	0.73	14.4	81	0.5	5.5	206	679	916	239
	DDB009	94.5	96.0	1.5	0.032	7035	1.07	8.1	63	0.4	8.0	802	2660	260	67
P04	DDB010	207.9	211.9	4.0	0.103	2.26	0.4	7.6	584	5	3	2660	3215	402	105
	DDB011	123.2	125.9	2.7	0.126	2.76	0.5	3.7	609	7	3	2197	3319	491	113
	DDB012	135.5	138.2	2.7	0.078	1.71	0.3	6.3	404	5	3	1712	2532	340	92
P05	DDB015	161.9	163.9	2.0	0.136	2.99	0.5	10.0	658	7	3	2773	3905	518	124
	DDB016	182.2	186.2	4.0	0.099	2.18	0.4	7.6	552	6	2	2178	2920	378	93
P05	DDB013	330.7	332.7	2.0	0.131	2.89	0.5	9.2	741	10	4	4768	4740	554	138
	DDB014	259.9	261.9	2.0	0.142	3.13	0.5	10.3	725	8	3	3515	4670	605	150
P06	DDB017	237.5	240.5	3.0	0.091	1.99	0.4	7.7	541	5	3	2580	3333	424	93
	DDB018	228.0	229.8	1.8	0.129	2.83	0.4	9.4	558	8	3	2406	3694	608	151
	DDB019	225.8	228.4	2.6	0.092	2.03	0.4	7.5	493	5	2	1944	2857	428	102
	DDB020	295.6	298.5	3.0	0.091	2.00	0.4	7.4	575	7	3	3240	3293	496	127
P11	DDB031	295.1	297.7	2.5	0.090	1.99	0.3	2.3	409	4	2	1594	2458	368	82
	DDB034	296.4	299.4	3.0	0.111	2.44	0.5	2.3	593	6	2	1977	2910	403	86
	DDB035	69.9	77.9	8.0	0.120	2.64	0.5	9.7	596	5	3	2434	3237	470	110
	DDB035	79.6	82.5	2.9	0.132	2.90	0.5	11.4	619	5	3	2224	3592	446	90
	DDB035	83.2	89.9	6.7	0.130	2.87	0.5	9.1	686	6	3	2972	3613	532	118
	DDB035	90.5	92.1	1.6	0.110	2.41	0.7	12.3	1007	5	4	3129	4249	384	73
	DDB035	130.7	134.6	3.9	0.162	3.56	0.6	13.0	747	6	4	3056	4230	641	135
	DDB035	301.9	304.0	2.1	0.093	2.04	0.4	8.6	671	6	2	2090	2465	290	51
	DDB035*	306.6	307.8	Low Recovery											
P13	DDB076	74.7	77.6	3.0	0.118	2.59	0.5	9.3	675	6	4	2928	3387	511	118
	DDB077	116.3	130.8	3.0	0.149	3.29	0.6	10.6	739	7	3	3120	3807	579	127
	DDB077*	244.7	245.4	Low Recovery											
P14	DDB062	82.3	83.1	0.8	0.044	0.96	0.3	1.8	122	0	3	738	1600	490	166
	DDB062	83.8	84.7	0.9	0.112	2.47	1.0	12.7	23	1	27	1145	3210	1918	508
	DDB063	Mineralized layer faulted out													
	DDB064	Mineralized layer faulted out													
P15	DDB047	39.6	40.7	1.1	0.145	3.19	1.2	14.2	1246	13	4	3411	4894	409	91
P16	DDB042	132.8	135.8	3.0	0.112	2.46	0.4	2.3	563	7	3	2249	3269	467	102
	DDB044	65.5	66.1	0.6	0.411	9.03	0.0	21.3	1	0	0	64	226	75	16
	DDB044	67.1	68.0	0.9	0.303	6.67	1.1	19.8	1913	19	9	6611	8054	1184	274
P17	DDB037	137.2	138.0	0.8	0.045	0.98	1.0	16.3	196	1	3	1290	2290	1113	312
	DDB039	286.5	287.5	1.0	0.042	0.93	0.3	5.3	358	2	2	985	1720	152	33
	DDB039	287.9	290.1	2.2	0.159	3.51	0.5	11.3	647	8	4	2806	4295	631	167
	DDB041	170.3	171.9	1.6	0.125	2.74	0.8	2.3	226	6	6	3857	3858	589	0
P19	DDB081	352.3	355.0	2.7	0.134	2.96	0.6	11.1	820	8	3	2871	4161	526	111
	DDB081	355.1	356.6	1.5	0.060	1.33	0.3	6.6	551	4	4	2406	3768	372	111
	DDB082	131.4	132.4	1.0	0.237	5.22	1.1	21.0	1440	21	6	2520	784	926	230
	DDB082	132.6	133.7	1.1	0.049	1.07	0.3	7.8	200	4	4	295	100	375	125

Table 10-2: Continued

Plataform No	Bore Hole No	From	To	Estimated True Width	U_3O_8		V_2O_5	P_2O_5	Mo	Re	Ag	Ni	Zn	Y_2O_3	Nd_2O_3
		(m)	(m)	(m)	%	lb/t	%	%	ppm	ppm	ppm	ppm	ppm	ppm	ppm
P21	DDB067	152.7	154.1	1.4	0.111	2.44	0.5	10.2	418	7	3	1800	3217	507	99
	DDB067	154.2	155.3	1.1	0.058	1.27	0.3	7.3	184	2	2	1075	2140	479	136
	DDB070	253.0	256.4	3.4	0.108	2.38	0.5	8.9	594	5	3	2540	3578	473	116
	DDB072	181.4	183.6	2.2	0.098	2.16	0.5	9.1	509	8	3	1999	3324	483	110
	DDB072	210.9	211.6	0.7	0.064	1.40	0.4	7.9	443	3	2	1680	1790	282	64
	DDB072	212.2	213.0	0.8	0.046	1.02	0.3	6.1	331	2	1	1310	1400	210	51
	DDB072	217.9	219.5	1.6	0.090	1.99	0.4	8.9	388	7	3	1485	2422	521	122
	DDB074	214.9	215.5	0.7	0.125	2.75	1.3	26.8	49	0	12	1055	2130	1467	297
P23	DDB057	46.3	47.1	0.8	0.089	1.95	1.0	23.5	154	0	2	271	525	1321	328
	DDB059	48.8	49.4	0.6	0.405	8.90	1.8	1.3	3850	47	12	6680	2290	411	411
	DDB060	76.6	77.6	1.0	0.051	1.13	1.1	11.8	138	0	6	392	1121	762	174
	DDB061	88.8	89.4	0.6	0.278	6.13	1.4	26.3	1990	20	5	4930	5280	884	143
	DDB061	89.9	90.9	1.0	0.231	5.09	0.9	18.0	1170	13	5	4980	4950	1116	250
P24	DDB048*	164.60	167.00	Low Recovery											
	DDB048	167.4	168.0	0.6	0.185	4.08	0.6	13.4	809	12	3	2410	4200	572	121
	DDB049*	190.5	192.0	Low Recovery											
	DDB051	241.4	244.0	2.6	0.118	2.59	0.5	8.8	581	7	3	2411	3210	420	103
	DDB053	157.0	159.8	2.8	0.084	1.85	0.3	6.7	485	4	2	1751	2541	362	83
P25	DDB046*	104.2	104.7	Low Recovery											
P27	DDB036	111.3	115.2	4.0	0.066	1.45	0.3	5.7	533	6	2	1496	2114	246	54
	DDB036	161.5	163.5	1.9	0.046	1.00	0.3	4.5	556	5	1	1520	1630	156	31
	DDB036	222.9	225.0	2.1	0.092	2.02	0.4	6.8	454	5	2	1643	2492	372	81
	DDB038	94.7	97.5	2.8	0.061	1.34	0.4	6.1	490	6	2	1471	1827	208	41
	DDB040	100.1	100.8	0.7	0.074	1.64	0.4	7.3	558	6	2	2010	2210	215	44
P28	DDB025	156.3	159.1	2.8	0.133	2.93	0.4	10.0	606	6	3	2743	3787	489	0
	DDB026	159.3	162.2	2.9	0.122	2.68	0.4	9.8	630	6	3	2495	3641	443	0
	DDB027	160.3	163.2	2.8	0.084	1.84	0.4	7.2	464	4	2	1767	2475	326	0
	DDB028	181.0	183.6	2.6	0.104	2.30	0.4	7.7	482	6	2	1990	2986	420	0
	DDB029	255.4	258.4	3.0	0.128	2.81	0.5	8.8	632	8	3	3125	3927	526	70
P29	DDB30	Mineralized layer faulted out													
	DDB32	Mineralized layer faulted out													
	DDB033	370.5	373.2	2.7	0.094	2.06	0.4	7.4	523	4	2	2562	3365	479	115
P30	DDB021	113.2	113.7	0.5	0.070	1.53	0.9	19.0	11	0	1	577	1360	1543	360
	DDB022	134.6	135.1	0.5	0.092	2.02	0.8	17.9	128	4	5	2060	2700	1233	324
	DDB023	Mineralized layer faulted out													
	DDB024	Mineralized layer faulted out													
P37	DDB071	299.1	300.6	1.5	0.140	3.09	0.5	10.4	902	9	3	4278	4451	632	150
	DDB073	351.5	353.4	1.9	0.145	3.18	0.6	10.1	1070	12	4	6221	5085	632	146
	DDB073	353.6	354.2	0.6	0.182	4.00	0.6	11.7	901	11	6	6280	6080	979	269
	DDB075	297.1	298.4	1.4	0.148	3.25	0.5	9.7	772	9	4	3533	4071	644	173
	DDB078	404.7	405.1	0.4	0.109	2.39	0.5	11.7	646	5	3	3150	3820	437	79
	DDB078	405.6	408.4	2.9	0.153	3.37	0.6	9.5	1079	12	4	6676	5329	699	166
	DDB078	410.9	411.3	0.5	0.177	3.89	0.7	11.5	1230	20	6	8810	6930	917	221
	DDB078	411.5	412.2	0.7	0.253	5.56	0.8	15.0	1235	14	7	9070	7770	1295	353
P40	DDB056	112.3	114.0	1.7	0.044	0.97	0.4	4.5	351	3	19	1870	2302	165	47
	DDB065	245.2	246.2	1.0	0.058	1.27	0.3	5.4	486	4	2	2290	2070	255	51
	DDB068	434.2	435.5	1.3	0.151	3.32	0.6	11.2	1270	12	3	6660	5210	635	126

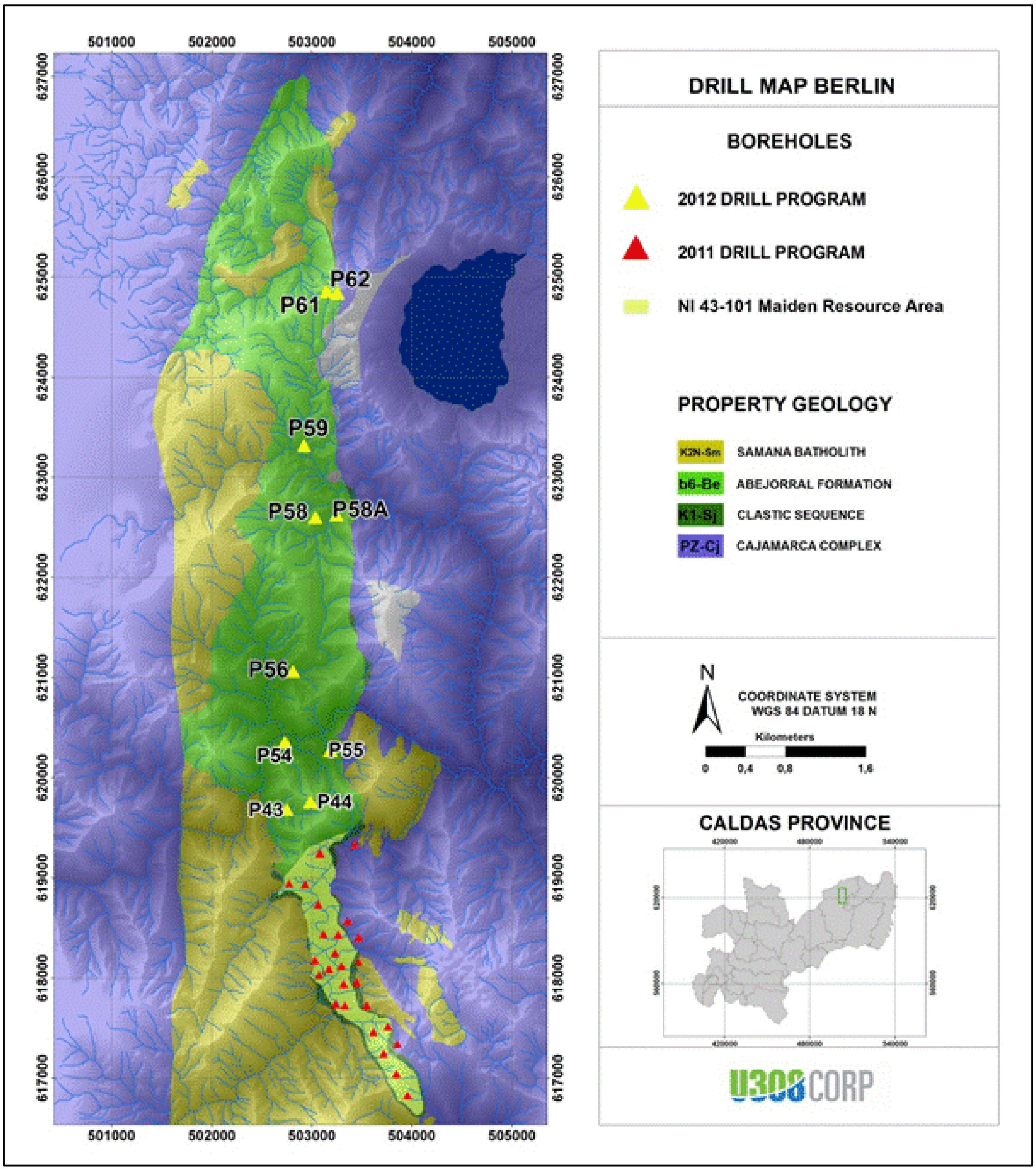


Figure 10-3: Map of the whole Berlin syncline showing the location of platforms from which the 2012 drill campaign was undertaken relative to the area in which the infill drilling was undertaken for the resource estimate in the 2010-2011 campaign

Table 10-3: Summary of bore hole length and the platform from which each hole was drilled in the 2012 campaign

Drillhole	Platform	Length (m)
DDB-083	P54	496.2
DDB-084	P44	384.0
DDB-085	P44	379.5
DDB-086	P54	450.2
DDB-087	P44	376.1
DDB-088	P54	698.0
DDB-089	P43	385.6
DDB-090	P55	153.9
DDB-091	P55	202.0
DDB-092	P58	608.0
DDB-093	P56	736.5
DDB-094	P58A	500.0
DDB-095	P59	700.0
DDB-096	P61	330.0
DDB-097	P62	111.3

Table 10-4: Assay results (at a 0.04% U_3O_8 cut-off grade) for intercepts from bore holes drilled in the 2012 campaign

	Bore Hole Info		Intercept (m)			Grade										
	Platform	Bore Hole Number	From	To	Estimated True Width	Uranium		Vanadium	Phosphate	Molybdenum	Rhenium	Rare Earths: Neodymium	Rare Earths: Yttrium	Nickel	Zinc
						U_3O_8		V_2O_5	P_2O_5	Mo	Re	Nd_2O_3	Y_2O_3	Ni	Zn
						%	lb/t	%	%	ppm	ppm	ppm	ppm	%	ppm
NARROW-WIDTH ZONE	P54	DDB-083	464.0	464.4	0.2	0.162	3.56	0.61	10.4	1,120	17.3	144	650	0.73	5,440
	P44	DDB-084	308.4	308.7	0.2	0.014	0.30	0.04	1.6	67	0.6	34	109	0.06	401
		DDB-087	328.3	328.9	0.4	0.063	1.39	0.30	4.9	428	5.8	65	266	0.33	2,248
	P54	DDB-088	658.7	659.1	0.3	0.139	3.06	0.59	11.9	713	8.8	115	519	0.37	4,260
	P43	DDB-089	353.7	354.3	0.4	0.031	0.68	0.17	3.1	233	1.7	30	138	0.11	1,226
EAST-CENTRAL AREA	P44	DDB-085	318.1	322.3	3.4	0.091	2.00	0.41	6.6	600	8.0	84	387	0.35	3,022
	P54	DDB-086	414.9	415.8	0.9	0.083	1.82	0.39	6.3	579	8.7	84	361	0.37	3,026
	P55	DDB-090	129.9	130.9	1.0	0.138	3.03	0.57	8.8	767	12.0	179	605	0.55	5,268
		DDB-091	179.8	182.3	2.2	0.105	2.30	0.46	7.3	614	8.5	103	395	0.39	3,656
	P58	DDB-092	Did not reach target depth												
	P56	DDB-093	713.2	716.6	2.4	0.078	1.71	0.33	6.3	491	7.8	84	357	0.35	2,875
		Including	714.7	716.3	1.3	0.119	2.61	0.45	8.3	700	12.6	134	563	0.53	4,349
	P58A	DDB-094	417.1	423.2	5.1	0.123	2.71	0.53	10.0	725	9.8	132	579	0.50	4,612
	P59	DDB-095	Did not reach target depth												
	P61	DDB-096	Sedimetary sequence in which mineralisation occurs was faulted out												
	P62	DDB-097	62.6	63.5	0.9	0.064	1.41	0.74	13.5	18	0.004	211	724	0.00	34

The geological observations made from the bore hole core are reported in Section 7. In order to minimise the environmental impact of drilling, multiple holes were drilled from most platforms. This required that some bore hole intersections were slightly oblique to mineralisation. In all cases, the angle of intersection of bedding observed in core was used to estimate the true thickness of the mineralised intersection. The nature of the mineralisation is defined in more detail in Section 14.

Down-hole radiometric analysis was done with a Mount Sopris probe manufactured by Mount Sopris Instruments and calibrated at that company's Grand Junction, Colorado facilities. On completion of each bore hole, the probe was lowered to the bottom of the hole on a cable and the radioactivity was measured at 10 cm intervals as the probe was winched up the hole. Data from the probe was downloaded at the field camp, analysed and stored in a database.

The friable and fractured nature of the mineralisation in some bore holes resulted in poor core recovery. Equivalent uranium grades were estimated from down-hole radiometric data in bore holes from which core recovery was low. As demonstrated in Section 14, there is a well-defined linear relationship between assay grade and uranium grade as estimated from the down-hole radiometric data.

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11 SAMPLE PREPARATION, ANALYSES AND SECURITY

Coffey Mining reviewed all aspects of sampling prior to calculating the mineral resource estimate. Coffey Mining considered that “sampling, preparation, assaying, drilling and storage procedures undertaken by U3O8 Corp. meet or exceed industry standard practice and are of high quality and suitable for use in Resource studies” (Coffey Mining, 2012). Procedures and protocols are being maintained in the ongoing exploration by U3O8 Corp.

11.1 Sampling Procedure

11.1.1 Exploration and Trench Samples

Trench locations and field sampling locations were determined on the basis of radioactivity and field mapping by U3O8 Corp’s geologists using standard industry practices. All personnel involved with the sampling wore dosimeters and masks if personnel were entering a deep trench or a confined space to do the sampling.

Samples typically consisted of 2-3 kg of material. A duplicate sample was taken from each sample site. All samples were bagged and numbered at the location at which they were taken in the field. Sealed samples were placed in backpacks and were carried by personnel or mules to U3O8 Corp’s vehicle at the end of the day. Samples were transported back to the U3O8 Corp’s local office in the Berlin village in a vehicle that is owned by the Company and was driven by one of its personnel. The samples were unloaded at the office, ordered, checked and stored in a locked, well ventilated store room. At approximately weekly intervals, the samples were transported in a U3O8 Corp. vehicle to a town called La Dorada, where the samples were delivered to a national courier company that provides transport to ALS Chemex’s preparation facility in Bogota. Duplicate trench samples were stored at the U3O8 Corp’s storage facility in the town of Ibague located 250 km from Berlin.

11.1.2 Drill Core Samples

Drill core was placed in aluminium or plastic core boxes by the drill contractor who also marked the depth at appropriate intervals on the core and on the core box. Rock quality data (“RQD”) was recorded by technicians at this early stage prior to the core being transported. A preliminary radiometric log was also done at this time with measurements being taken at 10 cm intervals with hand-held scintillometers.

The core boxes were sealed and carried by mule or aerial cableway to the nearest road where they were transferred to a waiting Company vehicle. A technician accompanied the core from the field to the vehicle. Detailed geological logging was undertaken on paper forms and the radioactivity of the core was checked and recorded. Sample intervals were marked by the geologists. Sample intervals were typically one metre, but varied according to lithology to ensure that sample intervals did not cross significant geological contacts. Radioactivity measured on the core was compared against, and checked for consistency with, a down-hole log of radioactivity.

The core was then cut with diamond saws in a well-ventilated area of the core storage facility. The core was cut in half and then one half was cut again. The ½ core with the two ¼ core segments was returned to the core box as each section was cut. Sampling then took place with ¼ core being put in polythene sample bags for assay and the duplicate section of ¼ core being placed in polythene bags for storage for later use for assay verification or metallurgical test work, for example. The core boxes were then stored in a locked, well ventilated store room.

Duplicate samples and sample blanks were inserted in the sample sequence at pre-determined intervals; they were numbered such that they were in sequence with mineralised material. Standards were also inserted at pre-determined intervals.

The sample bags containing ¼ core and QAQC samples for assay were weighed and packed in boxes for shipment by commercial road transport to ALS Chemex's sample preparation facility in Bogota.

11.2 Sample Preparation

On arrival at ALS Chemex's preparation facility, the samples were ordered and weighed, dried and jaw-crushed to 10 mesh ("#") – nominal 2 mm grain size. The 10# material was riffle-split and a sub-sample of 1 kg was pulverised to 75 µm, then split into a 150 gram ("g") sub-sample. This pulp was shipped by ALS Chemex to their assay facilities in Canada and Peru via commercial transport company. Remaining 10# material and excess pulp sample was returned to the Corporation for storage at its core storage facility in Ibague.

11.3 Sample Analysis

Different sample analysis methods were required to cover the suite of elements of potentially economic interest and these analytical procedures are described below under ALS Chemex's procedure codes.

11.3.1 ME-MS61U

A 0.25 g split of the sample pulp is digested with perchloric, nitric, hydrofluoric and hydrochloric acids (multi-acid digestion). The residue is topped up with dilute hydrochloric acid and analysed by Inductively Coupled Plasma - Atomic Emission Spectrometry ("ICP-AES"). Following this analysis, the results are reviewed for high concentrations of bismuth, mercury, molybdenum, silver and tungsten and such samples diluted accordingly. Samples are then analysed by inductively coupled plasma-mass spectrometry Inductively Coupled Plasma - Mass Spectrometry ("ICP-MS"). Results are corrected for spectral inter-element interferences. This method provides assay results for a 47 element suite, and uses an internal standard certified for uranium.

11.3.2 ME-M61

This uses the same method as ME-MS61U, yielding the same 47 element suite, but does not include the internal standard for uranium.

11.3.3 ME-MS81

The decomposition of the sample pulp is done using lithium metaborate fusion (code FUS-LI01) in which 0.2 g of sample pulp is added to 0.9 g of lithium metaborate flux and fused in a furnace at 1,000°C. The resulting melt is then cooled and dissolved in 100 millilitre ("mL") solution of 4% nitric acid and 2% hydrochloric acid. This solution is then analysed by ICP-MS. This assay method provides assay data for 38 elements, including uranium and the full suite of rare earth elements.

11.3.4 ME-XRF

This analytical method is applied specifically to samples that have phosphate content above the 10% upper detection limit of the ICP method. A calcined or ignited sample (0.9 g) is added to 9 g of lithium borate flux (50% $Li_2B_4O_7$- 50% $LiBO_2$), mixed well and fused in an auto fluxer at temperatures of between 1,050°C and 1,100°C. A flat molten glass disc is prepared from the resulting melt. This disc is then analysed by X-ray fluorescence spectrometry (“XRF”).

11.3.5 AA24

This analytical method is fire assay for samples that are periodically assayed for gold. Fire assay is of a 50 g aliquot and is finished with Atomic Absorption Spectroscopy (“AAS”).

11.4 Quality Assurance and Quality Control (“QAQC”)

All U3O8 Corp. sampling was supervised by qualified, experienced geologists and the actual samples were taken by appropriately trained geological technicians overseen by experienced geologists. All personnel involved with the sampling in the field and the chain of custody of the samples are employed by, or contracted by, U3O8 Corp.

All assay data were sent electronically by the analytical laboratory to the Berlin Project Manager in Colombia, the VP Exploration, whom resides in Argentina, and to the Company’s Technical Database Manager in Toronto, Canada. QAQC analysis was done by the Technical Database Manager who responded directly to the laboratory with queries related to the data and requested reanalysis or whatever remedial actions were required for QAQC purposes.

11.4.1 Laboratories Certificates

Sample preparation was done in ALS Chemex’s facility in Bogota, Colombia and fire assays were conducted at ALS Chemex’s facilities in Lima, Peru, while assay by other methods was done at ALS Chemex’s laboratory in Vancouver, Canada. ALS Chemex is a division of ALS Minerals which has been in operation for over 60 years and has over 60 analytical laboratories world-wide. ALS Minerals is certified to ISO 9001 (QC) standards and has an ISO/IEC17025 accreditation from the Standards Council of Canada.

Coffey Mining reviewed the sample preparation undertaken at the laboratory in Bogota and concluded that “the sample preparation is undertaken to a high industry standard” (Coffey Mining, 2012).

12 DATA VERIFICATION

Coffey Mining conducted a variety of data validation routines to verify the robustness of the database prior to calculating the resource estimate (Coffey Mining, 2012).

The verification checks did not highlight any material issues with the database and the resulting data was considered appropriate for use in mineral resource estimation.

A total of 12 independent samples were taken and were analysed using ICP and XRF by SGS laboratories in Perth, Western Australia. The samples gave results within acceptable limits of the original assays. Uranium values of the independent samples were within 2% of the values of the original samples (Coffey Mining, 2012).

Coffey Mining checked the veracity of the QAQC data for the Berlin Project as supplied by the Company, and considered it to meet or exceed industry standards (Coffey Mining, 2012). The analyses of certified standards, blanks, duplicates and laboratory standards were reviewed for both bore hole core and trenches.

In the opinion of Coffey Mining (Coffey Mining, 2012), the QAQC data available for the Berlin Project showed overall good accuracy and precision. The following comments and recommendations were made:

- The U3O8 Corp. assaying shows good levels of accuracy and precision, and the resulting assay database is suitable for use in resource estimation studies;
- Lower grade U standards should also be sourced (e.g. 200 ppm U, 500 ppm U, 800 ppm U) so as to test the accuracy of lower grade mineralised intervals;
- Additional umpire and coarse-crush (10#) duplicates are required to allow for an analysis of the coarse-crush precision levels; and
- It is recommended that standards, blanks, pulp duplicates and umpire pulp duplicates be sampled at a rate of 1:20.

The Company continues to apply the same QAQC procedures and independent data verification procedures in ongoing exploration in the Berlin Project.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Approach

Details of metallurgical test work undertaken on mineralised material from the Berlin Project were provided in the prior NI 43-101 written on the Berlin Project (Coffey Mining, 2012). This section provides a summary of that test work and incorporates information on subsequent test work.

Mineralisation at Berlin is confined to a specific limestone unit that weathers to a residue from which carbonate has been leached, leaving a rock that has the appearance of sandstone with silty intervals. Leach tests have been carried out on both fresh and weathered mineralised rock. Commodities of potential economic interest are: uranium, phosphate, vanadium, yttrium, neodymium, rhenium, molybdenum, nickel, silver and zinc and manganese.

The primary mineralised material at Berlin has a carbonate content of approximately 55%. In order to avoid high acid consumption, the initial leach tests used alkaline reagents. Uranium extraction was low to moderate, and was ineffective for a wide range of the commodities of potential economic interest. Alkaline leach tests were undertaken by SGS Lakefield and ANSTO.

Acid leach tests were conducted by SGS Lakefield, ANSTO and SGS Lakefield OreTest Pty ("SGS OreTest"). These tests showed improved rates of extraction from a wide suite of elements, including uranium, but with high acid consumption.

The most successful extraction method was found to be a two-step acidic ferric iron leach followed by an acid wash of the residue. This method proved to be effective for the extraction for the widest suite of elements. In the second step of the leach process, although a hydrochloric acid wash obtained higher extractions, the current process is based on a sulphuric acid wash so as to avoid the introduction of chlorine into the extraction process. All leach tests were conducted on raw mineralised material without beneficiation.

Reagent consumption in the acidic ferric iron leach tests is relatively high with either ferric iron or acid being required to drive the leach reaction. It is evident that within certain limits the proportion of ferric iron to acid is unimportant to the effectiveness of the leach process and the proportion used depends largely on the cost of these reagents. In practice, relatively low ferric iron concentrations are used to avoid fouling downstream extraction processes. Clearly, a means of removing carbonate from the mineralised material would reduce reagent consumption. Two means of beneficiating the mineralised material have been investigated:

- Flotation with initial test work having been undertaken by SGS Lakefield providing encouraging results. On advice from ANSTO, further testing on flotation was undertaken at Optimet in South Australia; and
- Pre-leach with acetic acid: acetic acid attacks carbonate minerals selectively without affecting other minerals such as phosphates. Test work was undertaken at SGS Lakefield in Canada and SGS OreTest in Western Australia.

13.2 Laboratories and Consultants

The metallurgical test work reported below was undertaken at the following laboratories:

- SGS OreTest in Perth, Western Australia. SGS OreTest was established as a metallurgical services company in 1993 as Lakefield OreTest Pty Limited and is now a subsidiary of the SGS Lakefield group, which has been offering mineral processing services to the mining industry since 1948;
- SGS Lakefield in Ontario, Canada, and predecessor companies, have been undertaking metallurgical test work for over 50 years and its Lakefield facility is ISO/IEC 17025 accredited;
- ANSTO was formed in 1987. It is a State agency within the portfolio of the Commonwealth Department of Innovation, Industry, Science and Research in New South Wales, Australia. ANSTO is responsible for delivering specialised advice, scientific services and products to government, industry, academia and other research organisations. It does so through the development of new knowledge, delivery of quality services and support for business opportunities; and
- Flotation test work is ongoing at Optimet in South Australia.

Metallurgical test work subsequent to that included in the prior NI 43-101 report on the Berlin Project (Coffey Mining, 2012) has been carried out under the guidance of Dr Paul Miller. Dr Paul Miller, a QP, has overseen the metallurgical test work carried out by SGS OreTest and at Optimet, and verified the technical information relating to the tests reported from those laboratories in this report. Dr Miller is a metallurgist who has specialised in hydrometallurgy and has over 30 years' experience in the commercial application of processes for the treatment of sulphide-bearing ore. Dr Miller has a doctorate in Chemical Engineering, is a member of the Institute of Mining and Metallurgy, London, and is also a Chartered Engineer. He is currently Managing Director of Sulphide Resource Processing Pty Ltd. Dr Miller is responsible for the design and interpretation of the ferric acid leach tests discussed in this Section 13.

Dr Miller has relied on information relating to the bore hole intersections used in the composite samples provided by Dr Richard Spencer, President and CEO of U3O8 Corp. and a QP.

13.3 Nature of Material

The metallurgical test work was carried out on ¼ core samples or sample reject material which is half-core that has been jaw crushed to ~2 mm grain size. Of the 82 bore holes drilled in the 2010-2011 drill campaign at Berlin, 74 intersected the mineralised horizon. Material from 25 (34%) of these intersections has been used in the various metallurgical tests (Table 13-1). The intersections used in the metallurgical test work are from bore holes from throughout the resource area on which the current PEA is based (Figure 13-1) and hence are considered to be representative of the mineralisation encountered in the Berlin deposit to date.

Mineralised material was shipped to the various laboratories where the samples were crushed to 10# (~2 mm diameter), blended together and homogenised to constitute the composite samples listed in Table 13-1

Table 13-1: Details of the nature of the material and bore hole intercepts from which the composite samples used in metallurgical test work were derived

Composite Sample	Lab where sample was prepared	Material	Rock Type	Bore Hole Number	From (m)	To (m)	Sample Type	Mass (kg)
Colombian Comp	SGS Lakefield	Oxidised, saprolite-like material	Sandstone & Siltstone	DDB-001	109.7	111.2	¼ core	3
				DDB-002	77.7	85.1		5
DDB5		Rock from drill core	Sandstone	DDB-005	138.65	144.76	¼ core	9
DDB7			Limestone	DDB-007	152.42	157.16		7
Berlin Comp			Sandstone & Limestone	2 kg of Comp DDB5 and 2 kg of DDB7 were mixed			¼ core	4
DDB10-15			Limestone	DDB-010	207.89	211.89	10# & ¼ core	42
				DDB-011	123.2	125.94		
				DDB-012	135.46	138.2		
				DDB-013	328.68	332.68		
				DDB-014	257.98	261.9		
				DDB-015	161.14	163.86		
UC Comp	SGS OreTest	Rock from drill core	Limestone	DDB-018	226.27	229.77	10# & ¼ core	11
				DDB-019	225.78	227.78		
				DDB-020	294.63	298.53		
				DDB-025	155.44	159.09		
				DDB-026	158.42	161.32		
BER-160611 or ANSTO 1	ANSTO	Rock from drill core	Limestone	DDB-016	181.45	186.95	¼ core	39
				DDB-017	236.86	240.94		
				DDB-018	226.27	229.77		
				DDB-019	225.22	228.35		
				DDB-020	294.63	298.53		
				DDB-025	155.44	159.09		
				DDB-026	158.42	162.22		
DDB27-38 or ANSTO 2				DDB-027	160.32	163.16	¼ or ½ core	49
				DDB-028	180.95	183.55		
				DDB-029	253.79	258.93		
				DDB-031	295.13	297.66		
				DDB-033	370.45	373.15		
				DDB-034	296.35	300.3		
				DDB-036	111.25	115.2		
					222.93	225.0		
				DDB-038	94.69	97.53		
					287.91	290.14		

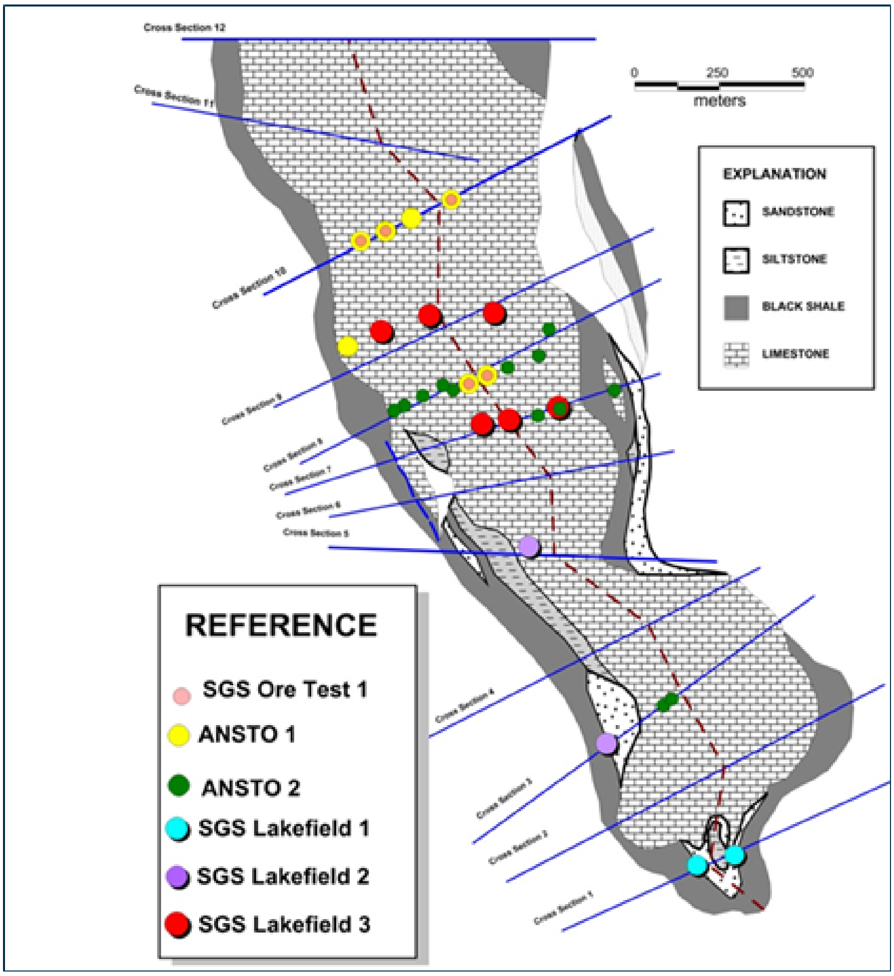

Figure 13-1: Map of the mineralised layer at Berlin unfolded into a horizontal plane showing the composition of the host rock and the location of bore hole intercepts that have been used in metallurgical test work undertaken to date

13.4 Head Grade & Composition of Composite Samples

Estimated mineral content of the ANSTO 1 composite sample is shown in Table 13-2. The principal acid consuming minerals are carbonate (calcite and dolomite) and phosphate. This sample is typical of the unoxidised mineralised material from the Berlin deposit.

The head grade of the various samples used in metallurgical test work is shown in Table 13-3. There are two main geological facies or mineralisation types at Berlin: one is a sandstone facies that has a relatively low carbonate content (composite samples Colombian Comp and DDB5), while over 90% of the uranium resource is hosted in carbonate facies (composite samples DDB7, DDB10-15, BER-160611 and UC Comp; Figure 13-1 and Table 13-1). One composite, sample Berlin Comp, is a mixture of material from DDB5 and DDB 7; and therefore, is a mixture between sandstone and carbonate mineralised material.

Table 13-2: Estimated mineral composition of the ANSTO 1 (BER-160611) composite sample of unoxidised mineralised material from the Berlin deposit (ANSTO, 2012)

Mineral	Chemical Formulae	Wt (%)
Calcite	$CaCO_3$	57.7
Fluorapatite	$Ca5(PO_4)_3F$	18.2
Quartz	SiO_2	15.8
Muscovite	$(K_{0.82}Na_{0.18})(Fe_{0.03}Al_{1.97})(AlSi_3)O_{10}(OH)_3$	2.6
Dolomite	$CaMg(CO_3)_2$	2.5
Pyrite	FeS_2	1.9
Chlorite	$(Mg,Fe)_5Al(Si_3Al)O_{10}(OH)_8$	0.9
Sphalerite	ZnS	0.4

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Table 13-3: Head grade of composite samples from the Berlin Project used in metallurgical test work

Labs		SGS Lakefield					ANSTO	SGS OreTest	
		Composite Sample Names							
Element	Units	Heads Colombian Comp	DDB7 Comp	DDB5 Comp	Berlin Comp	DDB10-15 Comp	ANSTO 1 or BER-160611	UC Comp	ANSTO 2 or DDB27-38
U_3O_8	%								
U	%	0.045	0.079	0.130	0.130	0.085	0.085		
Re	ppm	2.7	5.6	11.7	--	--		4.67	5.19
CO_2	%	1.69	25.60	0.99	9.82	26.90			
S	%	1.04	0.63	0.92	--	0.86	0.24	0.97	0.94
SiO_2	%	--	--	--	45.00	14.40	8.60	7.50	4.00
Ag	ppm	2.3	1.7	5.9	5.4	< 4		2.3	2.4
Al	%	2.92	0.66	3.09	2.22	0.67	0.74	0.68	0.70
As	ppm	80	101	212	97	169	130	205	145
Ba	ppm	1,050	1,080	1,900	1,310	957	850	880	1,729
Be	ppm	2.3	< 3	< 3	< 2	< 0.8		0.6	0.7
Bi	ppm	< 20	< 20	< 20	< 0.6	< 20		0.3	0.2
Ca	%	4.49	28.60	13.10	16.08	30.95	28.50	27.20	31.60
Cd	ppm	11	19	13	15	27		20.7	24
Co	ppm	8	5	10	6	< 10		9	6
Cr	ppm	317	287	774	547	274	700	868	507
Cu	ppm	150	80	184	131	85		157	103
Fe	%	2.85	0.47	1.17	1.43	0.50	0.53	0.81	0.73
K	%	0.74	0.22	0.78	0.56	0.21	0.24	0.24	0.26
Li	ppm	< 20	< 20	< 20	18	< 10		7	10
Mg	%	0.41	0.32	0.42	0.38	0.49	0.43	0.39	0.51
Mn	ppm	39	50	41	77	155		158	155
Mo	ppm	343	455	777	496	501		531	402
Na	%	0.04	0.03	0.05	0.04	0.07	<0.01	0.04	0.04
Ni	ppm	739	1,830	2,010	1,350	2,420		2,485	2,151
P	%	2.17	3.17	5.94	3.78	3.36	2.9	3.23	3.50
Pb	ppm	< 80	< 30	119.00	48.00	< 30		68	14
Sb	ppm	56	54	67	63	65		51	47
Se	ppm	192	202	601	350	285		232	180
Sn	ppm	< 20	< 20	< 20	2.00	< 20		0.9	0.4
Sr	ppm	486	618	924	663	736		838	805
Ti	%	0.22	0.05	0.14	0.13	0.04	0.038	428	489
Tl	ppm	< 60	< 30	< 30	15.40	< 30		7.5	6.6
V	%	0.30	0.21	0.50	0.34	0.25	0.19	0.23	0.25
Zn	%	0.06	0.21	0.06	0.11	0.28		0.26	0.25
Au	ppm	0.04	0.02	0.02	--	--			
Ce	ppm	85	59	154	119	66		73	110
Dy	ppm	17	17	48	31	22		21	26
Er	ppm	13	13	36	21	16		16	19
Eu	ppm	3	3	9	6	4		4	5
Gd	ppm	17	17	51	32	23		24	29
Ho	ppm	4	4	12	7	6		5	6
La	ppm	128	135	392	240	171		186	263
Lu	ppm	2	2	5	3	2		2	2
Nd	ppm	73	67	196	128	89		80	89
Pr	ppm	18	16	47	32	21		21	31
Sc	ppm	7	3	7	6	3		4	3
Sm	ppm	13	12	35	21	16		15	22
Tb	ppm	3	3	7	5	3		3	4
Th	ppm	9	3	9	9	3	<0.01	3	<.05
Tm	ppm	2	2	5	3	2		2	2
U	ppm	449	--	--	--	853		929	735
Y	ppm	225	230	664	415	326	320	363	340
Yb	ppm	11	10	29	17	13		12	13
Zr	ppm	0	0	0	0	0	400	11	6

13.5 Metallurgical Test Work on Unoxidised Ore

13.5.1 Alkaline Leach

ANSTO tests BER 3A and 4B were performed under aggressive alkaline leach conditions with a low slurry density of 2% solids and at temperatures of 90° and 250°C and reagent concentrations as shown in Table 13-4. Recoveries for uranium were not significantly different (67% and 68% extraction) under the very different temperature conditions (250°C and 90°C respectively). The recovery of vanadium, phosphate and nickel under alkaline conditions was poor. Molybdenum recovery was good at 91%, especially under the 90°C conditions.

Table 13-4: Summary of ANSTO alkaline leach tests

Test No	Conditions								Extractions %					
	Time Hrs	Grind p80 um	Temp °C	pH	Pulp Density (% Solids)	Na_2CO_3 Addn kg/t	$NaHCO_3$ Addn kg/t	Oxidant Addn kg/t	U	V	P	Y	Mo	Ni
BER 3 A	6	35	250	N/R	2	4,000	1,000	0	66.7	22.0	0.0	N/R	88.1	0.0
BER 4 B	48	12	90	10	2	2,000	500	16.2	68.2	27.2	3.4	N/R	91.1	11.3

SGS Oretest also performed two alkaline leach tests using material ground to below 20 µm but at pulp densities more typical of a commercial operation at 33% solids. Different reagent loadings were applied to the tests. The results from these tests are shown below in Table 13-5, showing that very poor extractions were achieved. Although it is accepted that the addition of an external oxidising agent may have resulted in improvements to extraction, the ANSTO results suggested only marginal benefits when an oxidising agent was added.

Table 13-5: Summary of SGS OreTest alkaline leach tests

Test No	Conditions								Extractions %					
	Time Hrs	Grind P80 µm	Temp °C	pH	Pulp Density (%solids)	Na2Co3 Addn kg/t	NaHCo3 Addn kg/t	Oxidant Addn kg/t	U	V	P	Y	Mo	Ni
1313	24	<20	70	10	33	100	40	0	33.6	0.9	0.1	0.0	48.0	0.2
1314	24	<20	70		33	70	40	0	36.0	0.7	0.0	0.0	53.2	0.1

13.5.2 Pressure Oxidation

Both ANSTO and SGS Oretest undertook pressure oxidation tests and a summary of the results obtained are shown below in Table 13-6.

Table 13-6: Conditions under which pressure oxidation tests were undertaken and metal extractions achieved

Test No	Conditions							Extractions %					
	Time Hrs	Grind P_{80} µm	Temp °C	Pulp Density (%solids)	Na2Co_3 Addn kg/t	$NaHCo_3$ Addn kg/t	Oxygen pressure Kpa	U	V	P	Y	Mo	Ni
01POX (SGS)	8	<20	150	38.8	40	10	1000	58.2	3	0.1	0.1	88.3	0
BER11A (ANSTO)	8	<12	220	37	40	10	1000	33.2					
BER12A	8	<65	220	40	40	10	1000	34.2					
BER13A	8	<65	170	40	40	10	1000	24.7					
BER14A	8	<65	220	40	40	10	1000	48.3					

The values of metal extraction from pressure oxidation tests are lower than anticipated even at a very fine particle size where a high liberation of uranium would be expected. It is accepted that while uranium locked in carbonates are unlikely to be attacked by pressure oxidation, the extractions obtained under a variety of conditions are still poor. It would be expected that the mineralised material would have a large consumption of oxidant due to the nature of the mineralogy and this would liberate significant amounts of carbon dioxide into the gas phase preventing further oxygen from being introduced for reaction and thereby hindering leaching. This is the only explanation that can be offered for the low extractions obtained.

13.5.3 Acid Leach

Baseline sulphuric and hydrochloric acid leach tests were performed to determine rates of extraction of uranium and other metals under various conditions as shown in Table 13-7. Recoveries for uranium were moderate to good, but with high to very high levels of acid consumption.

In all sulphuric acid leach tests, sodium chlorate was added to achieve a target Oxidation-Reduction Potential ("ORP") of 500 mV when measured using a platinum electrode against an Ag•AgCl saturated KCl reference electrode.

Table 13-7: Summary of conditions under which acid leach tests were performed on oxidised material, and metal extractions achieved, on composite samples from the Berlin Project (Coffey Mining, 2012)

Laboratory	Lab Test #	Sample	Grind P_{80} (µm)	T (°C)	Pulp density (%)	Leach time (Hours)	Average pH	Acid			Metal Recovery (%)						
								Type	Acid added (kg/t)	Net Acid Consumption (kg/t)	U	V	P	Y	Mo	Ni	Re
SGS	13	DDB7	46	65	50	48	2.54	H_2SO_4	712	652	78.0	72	52	34	32		12
	32	DDB10-15	100	20	2	3.5	5.1	Acetic	385		4.7	1	<1				
ANSTO	BER 1A	BER-160611	35	60	2	24	1.5	H_2SO_4	477	468	76.8	24.6	28.1		43	40	
	BER 1B		35	60	2	24	1.5	H_2SO_4	144	139	68.3	29.5	14.9		66	75	
SGS OteTest	AJ1033	UC Comp	100	25	10	48	2	H_2SO_4	558	550	19	10.9	18.7	17	11	30	

13.5.4 Acidic ferric iron leach

13.5.4.1 Background

Test work has demonstrated that relatively mild leaching conditions can be applied to the treatment of mineralised material from Berlin for extraction of a broad variety of elements. Test work was conducted on two composite samples at SGS OreTest in Australia and another two at SGS Lakefield in Canada (Table 13-8). All tests were undertaken at atmospheric pressure. Leaching was conducted in two steps:

- Step 1 is a leach with sulphuric acid and ferric iron at a temperature of 65°C for 24 to 48 hours; and
- Step 2 – the residue from the Step 1 is leached using 10% solutions of sulphuric or hydrochloric acid at 40°C for 12 to 24 hours.

The average extractions of elements from the 15 batch tests, together with the extractions obtained for a suite of metals are shown in Table 13-8. Good mass balances were obtained from the tests conducted.

Table 13-8: Summary of results of acidic ferric iron leach tests conducted on Berlin composite samples of unoxidised material

Lab	Composite Sample #	Leach Test Number	Maximum Grain Size (µm)	Ferric Leach (Stage1)				Releach / Wash (Stage 2)			Elemental Extraction %								
				Concentration (g/L)	Pulp density (%)	Temp. (°C)	Time (hrs)	Acid type & conc.	Pulp Density (%)	Temp. (°C)	Uranium	Vanadium	Phosphate	Yttrium	Neodymium	Zinc	Nickel	Molybdenum	Rhenium
SGS OreTest	UC Comp	1035/1071	106	50	10	65	48	10%HCL	10	40	98.4	73.2	99.2	94.8	51.0	99.4	61.7	48.0	33.8
		1079/1118	75	50	5			10%HCL	10		97.2	79.6	93.6	96.1	89.5	64.3	61.1	58.9	49.6
		1078/1117	38	25	5			10%HCL	10		98.8	82.0	99.5	95.7	94.7	98.1	59.7	47.7	29.7
		1082/1119	38	50	10			10%HCL	10		96.6	74.6	99.1	94.6	86.4	99.9	50.1	43.4	58.0
Average extraction from Composite Sample UC Comp with hydrochloric acid wash											**97.8**	**77.4**	**97.9**	**95.3**	**80.4**	**90.4**	**58.2**	**49.5**	**42.8**
SGS OreTest	ANSTO 2	1084/1110	106	50	5	65	48	10%HCL	10	40	97.2	80.2	99.5	95.8	82.8	98.5	62.7	56.8	27.1
		1085/1112	106	50	10			10%HCL	10		96.4	78.9	92.5	95.7	84.6	98.6	77.4	61.1	71.2
Average extraction from Composite Sample ANSTO 2 with hydrochloric acid wash											**96.8**	**79.6**	**96.0**	**95.8**	**83.7**	**98.5**	**70.1**	**59.0**	**49.2**
SGS Lakefield	DDB10-15	39/51	75	50	10	65	48	10%HCL	10	40	98.2								
		40/52	75	50	5			10%HCL	10		98.7								
Average extrcation from Composite Sample DDB10-15 with hydrochloric acid wash											**98.5**								
Average extraction from Composite Samples with hydrochloric acid wash											**97.7**	**78.5**	**96.9**	**95.5**	**82.0**	**94.5**	**64.1**	**54.2**	**46.0**
SGS OreTest	UC Comp	1035/1072	106	50	10	65	48	10%H_2SO_4	10	40	93.1	56.0	97.9	79.6	48.9	93.7	58.6	44.8	22.8
	ANSTO 2	1084/1111	106	50	5			10%H_2SO_4	10		97.3	69.4	99.4	89.8	63.6	97.0	62.7	53.8	11.1
		1085/1113	106	50	10			10%H_2SO_4	10		98.0	73.3	99.4	89.0	66.4	97.0	76.4	55.7	64.6
Average extraction from Composite Samples UC Comp & ANSTO 2 with sulphuric acid wash											**96.1**	**66.3**	**98.9**	**86.1**	**59.6**	**95.9**	**65.9**	**51.4**	**32.8**
SGS Lakefield	DDB10-15 Comp	39/43	75	50	10	65	24	10%H_2SO_4	10	40	97.9								
		39/47	75	50	10		48	10%H_2SO_4	10		86.7								
		40/44	75	50	5		24	10%H_2SO_4	10		96.3								
		40/48	75	50	5		48	10%H_2SO_4	10		95.5								
Average extraction from Composite Sample DDB10-15 with sulphuric acid wash											**94.1**								
Average extraction from 48 hour ferric leach tests with sulphuric acid wash											**94.1**	**66.3**	**98.9**	**86.1**	**59.6**	**95.9**	**65.9**	**51.4**	**32.8**
Average extraction from 24 hour ferric leach tests with sulphuric acid wash											**97.1**								

13.5.4.2 The Role of Ferric Iron

One of the most significant features of the ferric iron leach tests is that, for these tests which were conducted at a low pulp density, no acid additions were required to control pH. The acid required for this first stage of leaching can be provided by means of the natural acidity of the ferric sulphate solution (pH 1.5 to 2.5) combined with the generation of sulphuric acid by the natural precipitation of ferric iron as hydroxide. This is illustrated in Reaction 1 below in which each mole of ferric sulphate generates between two and three moles of acid.

$$Fe_2(SO_4)_3 \quad + \quad 6H_2O \quad \leftrightarrow \quad 2Fe(OH)_3 \quad + \quad 3H_2SO_4$$

Ferric sulphate Water Ferric iron hydroxide Sulphuric acid

Reaction 1: Ferric Hydroxide Formation

The second stage re-leach used only a relatively mild acid strength of 10% sulphuric or hydrochloric acid. Although the tests were conducted at a low pulp density, the principal role of the acid supplied is to act as a carrier for the ferric iron which is the dominant reactant for leaching. This is well illustrated by comparison of metal recoveries from the acid leach tests (Table 13-7) with those of the acidic ferric iron leach (Table 13-8), which shows higher rates of extraction for uranium as well as a broad suite of elements in the acidic ferric iron leach tests.

The fate of the ferric lixiviant used at 50 gram per litre ("g/L") can be estimated based on the reactions most likely to occur for carbonate neutralisation and phosphate release. Theoretically there are four possible iron species or end products which could form under the test conditions used:

- Ferric iron is hydrolysed to form the solid ferric hydroxide $Fe(OH)_3$ (this constitutes the major component in current tests);
- Ferric iron is hydrolysed to form the solid goethite (this would constitute a very minor component at the temperature of 65°C used in the current tests);
- Ferric iron is reduced to ferrous sulphate ($FeSO_4$) which is soluble; and
- Ferric iron is un-reacted and remains in solution as ferric sulphate $Fe_2(SO4)_3$.

Using sensible reaction assumptions the fate of ferric lixiviant from Test 39 (10% pulp density, 75 µm grain size, 65°C temperature and a concentration of 50 g/L ferric iron) appears to be as follows (Table 13-9):

- Up to 68% of the ferric iron appears to have formed ferric hydroxide solid which can be recovered in a re-leach step and re-used. Dissolution of the hydroxide is necessary in order to remobilise uranium and other elements that co-precipitated with the ferric hydroxide;
- Up to 19% of the ferric iron appears to have formed goethite solid which will be lost as it is largely resistant to dissolution with sulphuric acid although it could be dissolved with hydrochloric acid. However, goethite does not appear to capture soluble uranium and therefore, its dissolution is not necessary;
- Approximately 9% of the ferric iron has been converted to ferrous sulphate in solution and is available to be reconverted to ferric iron and recycled; and
- Approximately 4% of the ferric iron remains un-reacted and is still available for reaction or recycling in the liquor.

The above inventory suggests that up to 19% of the iron has been permanently lost from the system as insoluble goethite-like precipitates. This presumes that re-leaching of the residue to remobilise ferric iron is carried out together with re-conversion of ferrous to ferric iron. This also makes the assumption that acid is available for the re-leach and re-conversion stages. If insufficient acid is available for reconversion then further iron is lost from the system. In the current test scenario, the ferric make-up requirements would be 85 kg of ferric per tonne of ore due to the iron lost as goethite (presuming sulphuric acid and not HCl is used for re-leaching). This goethite has contributed 241 kilogram per tonne ("kg/t") of acid to the overall acid needs.

No external acid additions were made in Test 39 (the acidic ferric iron leach step) and the reaction of ferric iron to give the different iron end products created a minimum of 740 kg/t of acid for reaction. This value is in reasonable agreement with the theoretical requirement of approximately 766 kg/t of acid for neutralisation of carbonates and release of phosphate from apatite and other phosphate minerals.

In the sulphuric acid leaching step, approximately 80.7% of the uranium was extracted and remained in the ferric liquor. On acid washing of the acidic ferric iron leach residue (Test 43), the uranium extraction was increased to an overall value of 97.8% by solubilising uranium which had co-precipitated with iron. The measured amount of acid consumed in the acid washing step was 125 kg/t of ore equivalent and all the iron remobilised was present in the ferric form. Allowing for the need to convert the soluble ferrous iron formed from acidic ferric iron leaching back to ferric iron for re-use, a further 34 kg/t acid would be required. Therefore empirically the total acid consumption would be 160 kg/t with a ferric consumption of 85 kg/t.

These tests were conducted at a relatively low pulp density of either 5% or 10% solids. A commercial operation would be expected to operate at a higher pulp density in order to decrease reactor volume and to also increase the grade of the Pregnant Liquor Solution (“PLS”) for processing. At higher pulp densities, acid addition to the first stage leach becomes a requirement, rather than depending solely on ferric sulphate to generate acid by precipitation. This is due to practical constraints imposed by issues such as mixing when very large amounts of precipitates are formed.

Further work is currently underway to define more precisely the mechanisms of the reactions in order to optimise the leach process. Specifically, the balance between the precipitation of iron that generates acid versus addition of acid to allow more ferric iron to be recirculated into the leach solution requires thorough definition.

Integration of the leach extraction step into the complete flow sheet for metal recovery also requires consideration in terms of ensuring a PLS of sufficient grade is produced, which can be treated to selectively recovery the elements of value. This is generally achieved by operating at the highest pulp density which can be tolerated while still achieving high extraction. The sulphate medium used in the leach process also lends itself readily to conventional precipitation, IX and SX operations while minimising the leaching of unwanted accessory elements which might interfere with subsequent liquor processing.

Table 13-9: Tabulation of ferric iron and sulphuric acid make-up required for the two step acidic ferric iron leach process and uranium extraction achieved with each test on mineralised material from the Berlin Project

Test ID	Sample	Target P80, µm	Feed Density	Temp (ºC)	Acid Target, FA	Kg/t Ferric Lixiviant Make-up required	Total Overall kg/t Acid Make-up for releach + ferrous conversion	% Uranium extracted
FERRIC LEACH AND RELEACH TESTS DDB10-15 Comp 10% solids								
Test 39	DDB10-15 Comp	75	10	65	50 g/L Fe^{3+} added as ferric sulphate	84	159	80.7%
Test 43	Releach Test 39 RES 24 h		10	40	10% Sulphuric for Re-leach			89.5%
							%U Total =	98.0%
Test 39	DDB10-15 Comp	75	10	65	50 g/L Fe^{3+} added as ferric sulphate	294	34	80.7%
Test 47	Releach Test 39 RES 48 h		10	40	10% Sulphuric for Re-leach			31.2%
							%U Total =	86.8%
Test 39	DDB10-15 Comp	75	10	65	50 g/L Fe^{3+} added as ferric sulphate	114	156	80.7%
Test 51	Releach Test 39 RES 48 h		10	40	10% Hydrochloric for Re-leach			90.8%
							%U Total =	98.2%
FERRIC LEACH AND RELEACH TESTS DDB10-15 Comp 5% solids								
Test 40	DDB10-15 Comp	75	5	65	50 g/L Fe^{3+} added as ferric sulphate	8	718	87.8%
Test 44	Releach Test 40 RES 24 h		10	40	10% Sulphuric for Re-leach			79.6%
							%U Total =	97.5%
Test 40	DDB10-15 Comp	75	5	65	50 g/L Fe^{3+} added as ferric sulphate	80	858	87.8%
Test 48	Releach Test 40 RES 48 h		10	40	10% Sulphuric for Re-leach			62.8%
							%U Total =	95.5%
Test 40	DDB10-15 Comp	75	5	65	50 g/L Fe^{3+} added as ferric sulphate	22	246	87.8%
Test 52	Releach Test 40 RES 48 h		10	40	10% Hydrochloric for Re-leach			89.6%
							%U Total =	98.7%
FERRIC LEACH DDB5 Comp 10% solids								
Test 41	DDB5 Comp	75	10	65	50 g/L Fe^{3+} added as ferric sulphate	5	197	71.8%
Test 45	Releach Test 41 RES 24 h		10	40	10% Sulphuric for Re-leach			66.8%
							%U Total =	90.6%
Test 41	DDB5 Comp	75	10	65	50 g/L Fe^{3+} added as ferric sulphate	4	268	71.8%
Test 49	Releach Test 41 RES 48 h		10	40	10% Sulphuric for Re-leach			78.8%
							%U Total =	94.0%
Test 41	DDB5 Comp	75	10	65	50 g/L Fe^{3+} added as ferric sulphate	3	169	71.8%
Test 53	Releach Test 41 RES 48 h		10	40	10% Hydrochloric for Re-leach			76.0%
							%U Total =	93.2%
FERRIC LEACH DDB5 Comp 5% solids								
Test 42	DDB5 Comp	75	5	65	50 g/L Fe^{3+} added as ferric sulphate	20	N/A	85.4%
Test 46	Releach Test 42 RES 24 h		10	40	10% Sulphuric for Re-leach			96.3%
							%U Total =	99.5%
Test 42	DDB5 Comp	75	5	65	50 g/L Fe^{3+} added as ferric sulphate	3	217	85.4%
Test 50	Releach Test 42 RES 48 h		10	40	10% Sulphuric for Re-leach			70.6%
							%U Total =	95.7%
Test 42	DDB5 Comp	75	5	65	50 g/L Fe^{3+} added as ferric sulphate	3	112	85.4%
Test 54	Rleach Test 42 RES 48 h		10	40	10% Hydrochloric for Re-leach			71.5%
							%U Total =	95.8%

13.5.4.3 Particle Size

Three different particle sizes were investigated in the acidic ferric iron leach tests (106 µm, 75 µm, 38 µm) with very little difference in metal extraction. In fact, some of the superior results were obtained from those tests using the larger size range of 106 µm (Table 13-8). Although more extensive tests are required to verify the influence of particle size, these results are promising in suggesting that an acceptable grind size associated with normal milling operations may be sufficient to liberate elements and expose them sufficiently for leaching.

13.5.4.4 Temperature

There is evidence that Step 1 of the acidic ferric iron leach process is temperature-sensitive with higher extraction obtained with higher temperature. Significantly higher rates of extraction were obtained at higher temperature for uranium, phosphate, yttrium, neodymium and nickel (Table 13-10). Zinc extraction dropped notably at higher leach temperature.

Table 13-10: Comparison of extraction data for Step 1 of the ferric iron leach process for selected elements at two different temperatures. Other leach conditions were similar with ferric iron concentration of 50 g/L, particle size 38 µm, pulp density of 5% and 48 hour test duration

Leach Test Number	Temp (°C)	Extraction										
		U	V	P	Y	Nd	Zn	Ni	Mo	Re	Ag	Mn
1034	40	51%	5%	5%	10%	7%	92%	42%	9%	6%	nd	97%
1035	65	68%	7%	17%	26%	17%	69%	52%	8%	nd	nd	97%

13.5.4.5 Pressure

All tests were conducted at atmospheric pressure and no noxious gases or fumes were generated.

13.5.4.6 Leach Time

Test work shows that optional uranium extractions are obtained after a short leach time of 24 hours (Table 13-8).

13.5.4.7 Leach Stages

Although the test work consisted of two stages of leaching, the results suggest that the major influence of the second stage was to improve recovery by making soluble those elements which had been leached in the first stage but had co-precipitated with iron hydroxide species. This suggests that a commercial operation could consist of a single train or reactors with the operating conditions for the stages of reactors being appropriately controlled with different reagent additions to represent the two stages of leaching.

13.5.4.8 Optimal Leach Conditions

Optimal conditions, based on the results of test work done to date, for the initial acidic ferric iron leach (Step 1) and acidic leach of the iron-rich residue (Step 2) for the acidic ferric iron leach process are summarised in Table 13-11.

Table 13-11: Summary of optimal leach conditions for Step 1 and Step 2 of the acidic ferric iron leach process

Condition	Step 1 – acidic ferric iron leach	Step 2 – acid releach
Pressure	Atmospheric	
Temperature	65°C	25°C
Leach Time	8 - 12 hours	8 - 12 hours
Pulp Density	15%-35%	30%-40%
Ferric Iron Concentration	10 – 50 g/L	-
Maintained Sulphuric Acid Concentration	50 g/L	10%

13.5.4.9 Reagent Consumption

Results from test work on mineralised material from Berlin shows that good rates of extraction are achieved for a wide range of elements irrespective of the proportion of sulphuric acid to ferric iron used. Table 13-9 shows the calculated ferric iron and acid requirement for the two-step acidic ferric iron leach process for various samples of mineralised material from Berlin with percentage extraction shown for uranium. On one end of the spectrum, leach tests use ferric iron only without external acid addition in Step 1 of the acidic ferric iron leach process. Under these conditions, and at a low pulp density, all of the acid required for the consumption of carbonate is generated by the ferric iron. Table 13-9 also shows the quantity of acid required per tonne of mineralised material for the regeneration of ferric iron from ferrous iron as well as the quantity of acid required for Step 2 of the acidic ferric iron leach process in which the iron-rich precipitate is leached with sulphuric acid.

On the other end of the spectrum, tests using mainly sulphuric acid with a small amount of ferric iron, achieve similar extraction for the suite of elements (Table 13-12). This is demonstrated by comparing the results of acid-only leach tests with results of a test done under similar conditions in which ferric iron is added with a similar amount of sulphuric acid. Test conditions were: ferric iron of 0 and 50 kg/t, with similar amounts of sulphuric acid addition (558 and 512 kg/t respectively in Tests 1022 and 1180, Table 13-12). These results indicate that high rates of extraction are achieved across a wide range of ratios of acid to ferric iron. Extraction, therefore, is dependent on the presence of ferric iron rather than on a specific proportion of ferric iron to sulphuric acid.

Temperature is also an important factor in acid leach with uranium extraction rising from just 19% at ambient temperature to 77.5% at 65°C (Tests 1033 and 1180, Table 13-12).

Table 13-12: Sulphuric acid and ferric iron leach test results for mineralised material from Berlin showing sensitivity of the efficiency of acid leach to temperature and the increase in extraction rate with the addition of ferric iron under similar acid conditions at constant temperature

Test No	Conditions	Ferric iron concentration (kg/t)	Acid usage kg/t	Metal Recovery (%)					
				U	V	P	Y	Mo	Ni
1033	Sulphuric acid ambient temp 25°C	0	652	19	10.9	18.7	17	11	30
1022	Sulphuric acid temp 65°C	0	558	78	72	52	34	32	n/a
1180 /1190	Sulphuric acid and 50 kg/t ferric temp 65°C	50	512	98	70	99	89	38	53

Results from tests in which different ratios of ferric iron and sulphuric acid are used allows the calculation of a practical trend line that shows the proportion of these reagents required to achieve efficient leaching of elements (Figure 13-2). The fact that mineralised material from Berlin is amenable to leaching under such wide reagent ratios is beneficial to the economics of the deposit because the ratio can be adjusted according to reagent price. If sulphuric acid prices were to rise, for instance, a higher proportion of ferric iron could be used so that less sulphuric acid is required.

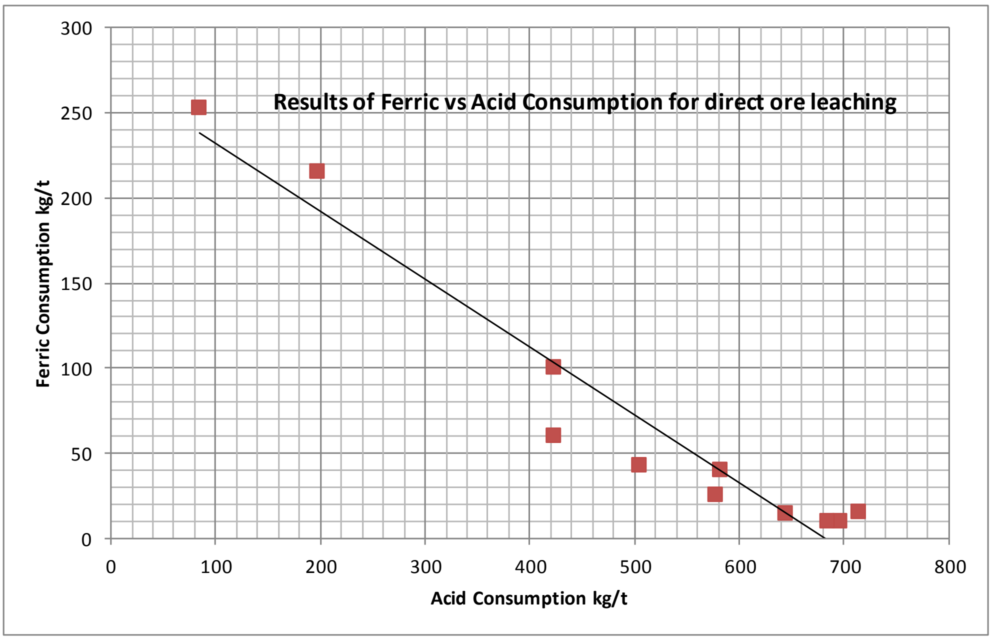

Figure 13-2: Summary of sulphuric acid and ferric iron consumption in acidic ferric iron leach tests of mineralised material

By comparison to the practical trend line developed above, Figure 13-3 below shows the theoretical trend line for the ratio of ferric iron to sulphuric acid required for optimal leaching of elements for raw, unbeneficiated mineralised material as well as for material beneficiated by removal of carbonates with acetic acid.

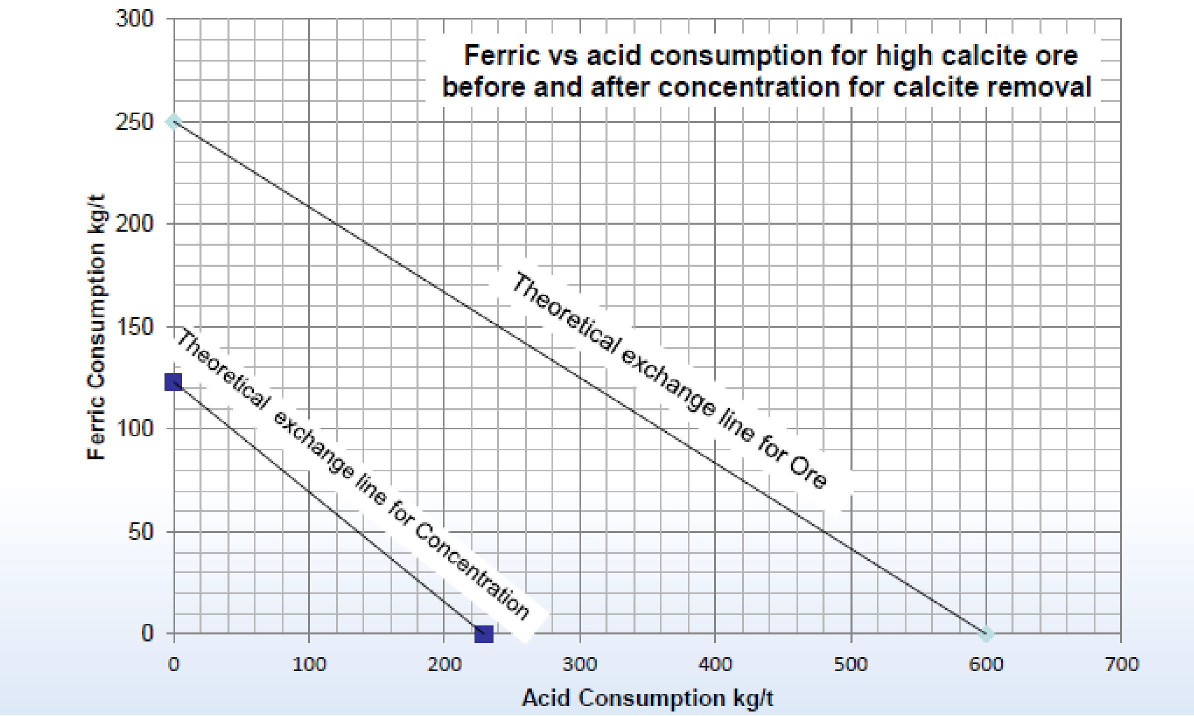

Figure 13-3: Theoretical exchange line that shows the ratio of ferric iron and sulphuric acid required for optimal element leaching from mineralised material from Berlin. Exchange lines are shown for unbeneficiated material and for material that has been beneficiated by acetic acid leach

13.5.5 Beneficiation by Acetic Acid Pre-leach

13.5.5.1 Background

Gharabaghi *et al.* (2010) demonstrated the potential of organic acids to selectively dissolve carbonate without dissolving phosphate. Experimental work by these authors had shown that, under certain leach conditions, organic acids selectively attack calcite without affecting apatite. Acetic acid reacts with calcite to form calcium acetate as per the following reaction:

$$2CH_3COOH + CaCO_3 \Rightarrow Ca(CH_3COO)_2 + CO_2 + H_2O(1)$$

Acetic acid Calcite Calcium acetate Carbon dioxide Water

Reaction 2: Calcium Acetate Formation

Initial test work at SGS Lakefield confirmed that acetic acid attacked carbonate selectively relative to apatite. In Test 35, 100 µm material from Composite DDB10-15 was leached for 3.5 hours with 20% acetic acid (Coffey Mining, 2012). Approximately 35% of the carbonate gangue was removed by acetic acid and resulted in a 29% mass loss between the feed and residue. Analysis of the solution showed that it contained only 4.7% of the uranium and very little phosphate (8 ppm P). These results show that acetic acid dissolves carbonate and that very little of the uranium contained in the mineralised material goes into solution. This means that the majority of the uranium and essentially all of the phosphate are concentrated at a higher grade in the smaller mass of residue.

Given the initial success of the acetic acid leach test, additional test work was undertaken by SGS OreTest. Four leach tests were undertaken on composite sample ANSTO Comp by SGS OreTest. The objective of these tests was to reduce the amount of acid consuming gangue reporting to the subsequent acidic ferric iron leach process.

A sample of mineralised material ground to 106 µm was subjected to acetic acid leach using a dilute acid strength of either 0.89 or 1.44 moles per litre at ambient temperature for either 30 or 60 minutes. Test results showed excellent selective leaching of calcite and dolomite together with a substantial mass reduction of between 53% and 60% between the original feed and the sample residue remaining after acetic acid leaching (Table 13-13). This decrease in mass resulted in a significant upgrading of all the elements of value in preparation for their subsequent extraction by acidic ferric iron leaching.

Between 5% and 7% of the uranium contained in the mineralised material was leached into solution by the acetic acid. A shorter leach time and a lower acetic acid strength resulted in less uranium being leached while still eliminating the majority of the calcite (Table 13-13). As it is recognised that approximately 5% of the uranium is contained in calcite, this would give an explanation for this amount of uranium being leached in the acetic acid process. Other elements with a tendency to leach with acetic acid are nickel (between 12% and 18%) and zinc (between 8% and 11%). Leaching of vanadium was limited to between 4% and 6% and phosphate to below 0.1%.

Table 13-13: Summary of conditions under which acetic acid leach tests were undertaken with percentage extraction of selected elements by acetic acid

Test No.	Sample	Size - P_{100}, µm	Percent Solids (w/w)	Acetic Acid Conc (moles per litre)	Leach Time (minutes)	Mass Loss, %	Final Extraction, %								
							U	V	P	Y	Nd	Zn	Ni	Mo	Re
1200	ANSTO Comp	106	7.04	0.89	30	53	5.5	3.2	0.0	3.4	2.1	7.3	12.2	14.1	9.4
1201		106	6.81	1.44	30	60	7.4	5.9	0.1	3.4	2.5	10.0	18.2	11.6	10.0
1202		106	9.88	1.44	30	53	5.2	4.2	0.1	3.1	2.3	8.3	14.1	10.6	8.6
1203		106	6.81	1.44	60	61	7.1	6.0	0.1	3.2	2.7	9.4	18.1	12.0	9.9

After reacting with calcite to form calcium acetate, acetic acid can be regenerated using less costly sulphuric acid as per Reaction 3.

$$Ca(CH_3COO)_2 + H_2SO_4 + H_2O \Rightarrow CaSO_4.2H_2O \downarrow + 2CH_3COOH \quad (2)$$

Calcium acetate   Sulphuric acid   Water   Gypsum   Acetic acid

Reaction 3: Acetic Acid Regeneration

The regeneration of the acetic acid by reacting the calcium acetate with sulphuric acid (Reaction 3) results in the precipitation of gypsum. Approximately 580 kg of sulphuric acid is required per tonne of mineralised material to regenerate the acetic acid, producing approximately 776 kg of gypsum dehydrate per tonne of ore. This acid consumption is essentially the same as the amount of sulphuric acid that would be required to leach the carbonate directly. Nevertheless, there are advantages to using an acetic acid pre-leach:

- The generation of gypsum – a marketable by-product of high purity which could partially offset the cost of the sulphuric acid; and

- Consumption of carbonate by acetic acid reduces the volume of low-carbonate feedstock for subsequent acidic ferric iron leach by 53-60%, resulting in an approximate doubling of the metal and phosphate grade in the low-carbonate feedstock. This increase in grade is beneficial to downstream metal recovery by substantially increasing the grade of elements reporting to the PLS. This reduces equipment sizing and makes processing of the PLS more cost effective with conventional recovery routines of IX, SX and selective precipitation.

Realistic process modelling suggests that regeneration of acetic acid by reacting calcium acetate with sulphuric acid is not 100% efficient and that up to 56 kg of acetic acid may be required as a top-up, per tonne of mineralised material, to maintain the acetic acid concentration. Further modelling work is required to evaluate this more fully and to propose methods for reducing this loss, which is mainly in solid-liquid separation of the gypsum and washing routines. This is important as it constitutes a major operating expense for the acetic acid leach operation.

SGS OreTest performed a regeneration test on the calcium acetate solution which showed that the small amount of soluble uranium remained in solution and did not report to the final gypsum product. It should therefore be possible to selectively recover this uranium from solution. An ICP scan of the final gypsum product shows the gypsum to be of high purity (Table 13-14). Further test work can be undertaken to improve the calcium content of the final product.

Table 13-14: ICP analysis of the final gypsum product generated by reconstitution of acetic acid

Element	Gypsum Product ppm	Element	Gypsum Product ppm
Ag	<0.5	Mo	<0.5
Al	920	Na	<500
As	<5	Nb	<0.5
Ba	83	Nd	3
Be	<0.5	Ni	<10
Bi	<0.5	P	<100
Ca	215,000	Pb	<5
Cd	<0.5	Pr	1
Ce	9.3	Rb	<0.25
Co	<0.5	S	183,000
Cr	<50	Sb	0.5
Cs	<0.25	Sc	<1
Cu	<10	Se	<10
Dy	1	Sn	<1.5
Er	<0.5	Sr	348
Eu	<0.25	Ta	<0.25
Fe	140	Tb	<0.25
Ga	<1	Te	<0.5
Gd	1	Th	0.30
Hf	<0.25	Ti	7
Ho	<0.25	Tl	<0.5
In	<0.1	Tm	<0.25
K	390	U	0.70
La	12.1	V	<5
Li	<0.5	W	<0.5
Lu	0	Y	5.5
Mg	<100	Yb	<0.5
Mn	4.0	Zn	<25
		Zr	<2.5

13.5.6 Beneficiation by Flotation

13.5.6.1 Objective and Background

Flotation is being investigated as an alternative means of beneficiation to acetic acid pre-leach. Although acetic acid leach has proved to be an extremely efficient means of removing carbonate from the mineralised material, flotation may provide a more cost-effective alternative. Alternatively, both flotation and acetic acid leaching may be used in tandem as a means of beneficiating mineralised material from Berlin.

There have been two areas of focus for flotation test work based on the following characteristics of the mineralised material:

- Uranium in mineralised material from Berlin is closely associated with organic carbon, and since the carbon is amenable to flotation, the carbon with contained uranium can be separated through a primary flotation step known as "pre-flotation". Sulphide minerals that contain nickel, zinc and possibly molybdenum and rhenium also concentrate efficiently into a flotation concentrate. The test work findings also indicate a tendency for apatite to report to the flotation concentrate preferentially to calcite under conditions which promote sulphide flotation; and

- The carbon-poor flotation tailings contain a high proportion of calcite that is difficult to separate from apatite, the principal phosphate mineral, due to their similar surface properties. Not only is phosphate an important contributor to the economics of the project as a source of phosphoric acid, but valuable rare earth elements, principally yttrium and neodymium, also occur principally as phosphate minerals. Hence, the second component of the flotation work is focused on selective flotation for the recovery of apatite while rejecting calcite, as a means of reducing acid requirements in the acidic ferric iron leach extraction step. The two objectives are to some extent interdependent. For example, the early removal of organic carbon as a pre-float concentrate would also eliminate its potential interference in selective flotation for the separation of apatite from calcite. Investigations for the selective separation of apatite from calcite have also shown success, but further work is needed to reduce the loss of uranium and other commodities including apatite, reporting to the carbonate-rich flotation tailings.

13.5.6.2 Grind Size

SGS Lakefield Tests DDB-F1 and DDB-F2 were designed to investigate the effect of grind size on flotation results. Grind sizes of 104 µm and 58 µm were used. Results suggest a slight improvement in carbon flotation for the finer particle size and, as a result of this observation, all subsequent tests undertaken by SGS Lakefield were performed at the finer grain size. Although a greater amount of carbon was collected (12.5%) at a finer size for the same mass pull, the amount of uranium and calcite/apatite reporting to the pre-float concentrate remained constant. Test work at Optimet was undertaken at two grind sizes of 45 µm and 25 µm and results show no appreciable difference in selectivity of calcite and apatite. However this work did suggest a slight improvement in sulphide recovery during pre-flotation at the finer grind size and, as both sulphide and carbon values are collected in this stage, it provides support to the SGS Lakefield findings that a finer grind size is of benefit in pre-flotation.

13.5.6.3 Carbon Pre-float

SGS Lakefield undertook two tests that included a pre-float of carbon before conducting flotation tests specific to the separation of apatite from calcite. This protocol was adopted based on the reasoning that the graphite appeared to inhibit the separation of calcite and apatite by flotation and, thus, the objective of the pre-float was to remove the organic carbon to avoid such interference.

Results for the carbon pre-float tests (SGS Tests DDB-F1 to DDB-F4) are illustrated in Figure 13-4. The results show that carbon reports to the pre-float concentrate much more readily than the other minerals of interest and that the proportion of total organic carbon ("TOC") concentrated in the pre-float material increases with increasing mass pull. There is also a slight preference for apatite collection into the pre-float concentrate over calcite at higher mass pull. Mass pull refers to the mass of material that is concentrated in each step of the flotation process. Although not shown by this data set, sulphides also have a tendency to report to the concentrate even when pre-float conditions are specific for TOC flotation.

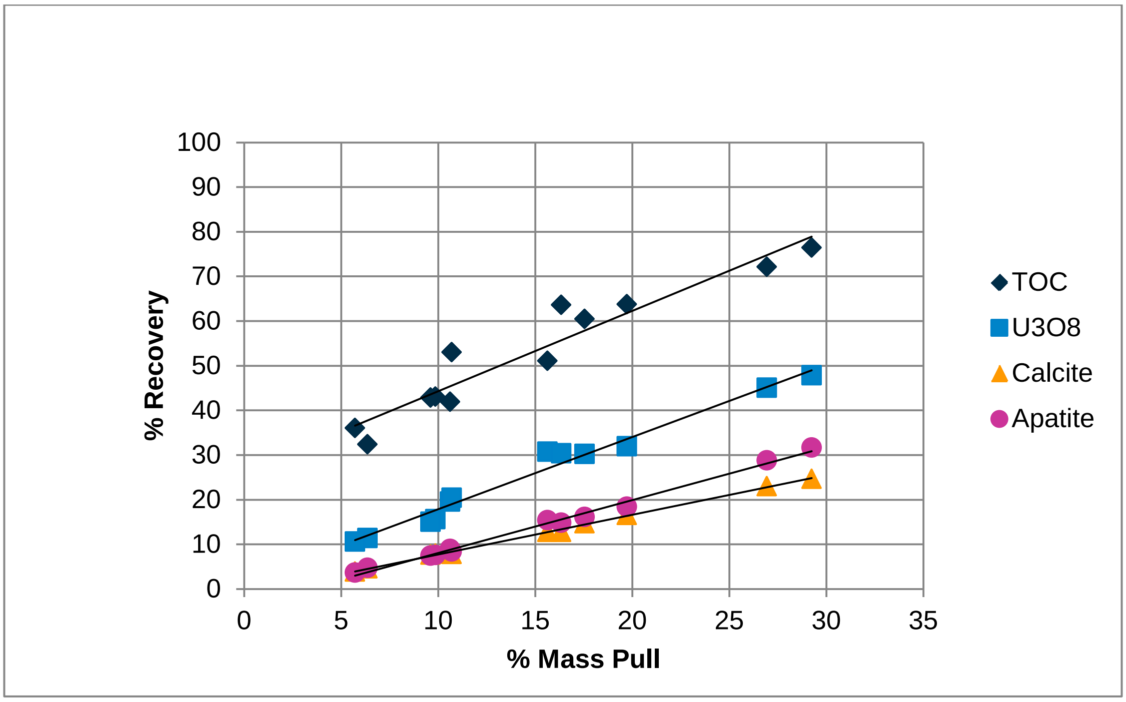

Figure 13-4: Recovery as a function of mass pull into the carbon pre-float in SGS Lakefield Tests DDB-F1 to DDB-F4

Figure 13-5 below shows the recovery of uranium against organic carbon for the SGS Lakefield pre-float tests. A first order magnitude relationship with the available data gives the best correlation ratio of 0.914 suggesting that complete recovery of the organic carbon might result in a uranium recovery of approximately 65%.

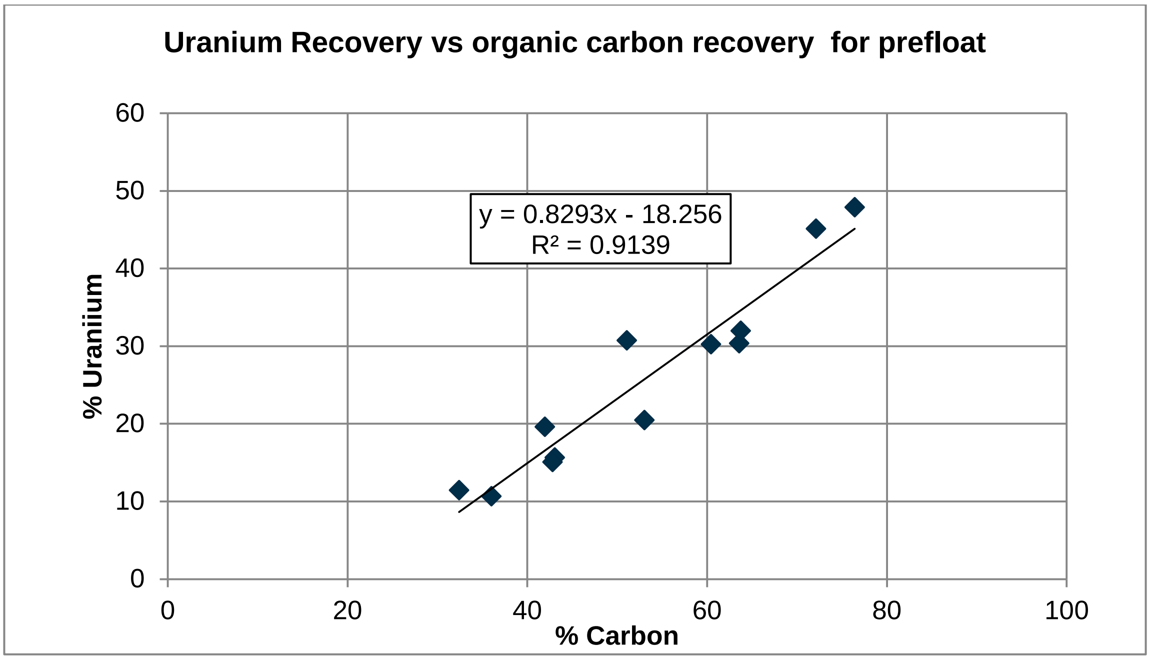

Figure 13-5: Uranium recovery versus organic carbon recovery in pre-float material from SGS Lakefield test work

13.5.6.4 Sulphide Flotation after Carbon Pre-float

The introduction of sulphide flotation after the carbon pre-float in SGS Lakefield Tests DDB-F3 and DDB-F4 resulted in a significant improvement in separation of minerals. The only difference in protocol between these two tests was the use of Calgon as a dispersant in Test DDB-F4.

The use of a carbon pre-float followed by sulphide flotation resulted in 71.3% of the uranium and 57.4% of the apatite being recovered with only 40.9% of the calcite into 47% of the mass in Test DDB-F3 (Figure 13-6). Over 86% of the carbon and 84% of the sulphide were recovered into the concentrate.

A significant enhancement of the separation of apatite from calcite coincides with the start of sulphide flotation (26% mass pull, Figure 13-6). It is presumed that this is largely associated with the use of PAX flotation reagent which would have a natural tendency to suppress calcite relative to sulphides. The sulphide flotation stage resulted in a near doubling of the sulphide content of the sulphide concentrate from 48.7% to 84% and an increase in carbon recovery from 79.3% to 86.5%. Uranium recovery improved from 57.8% to 71.3% and vanadium recovery to the concentrate was also high at 76.1%.

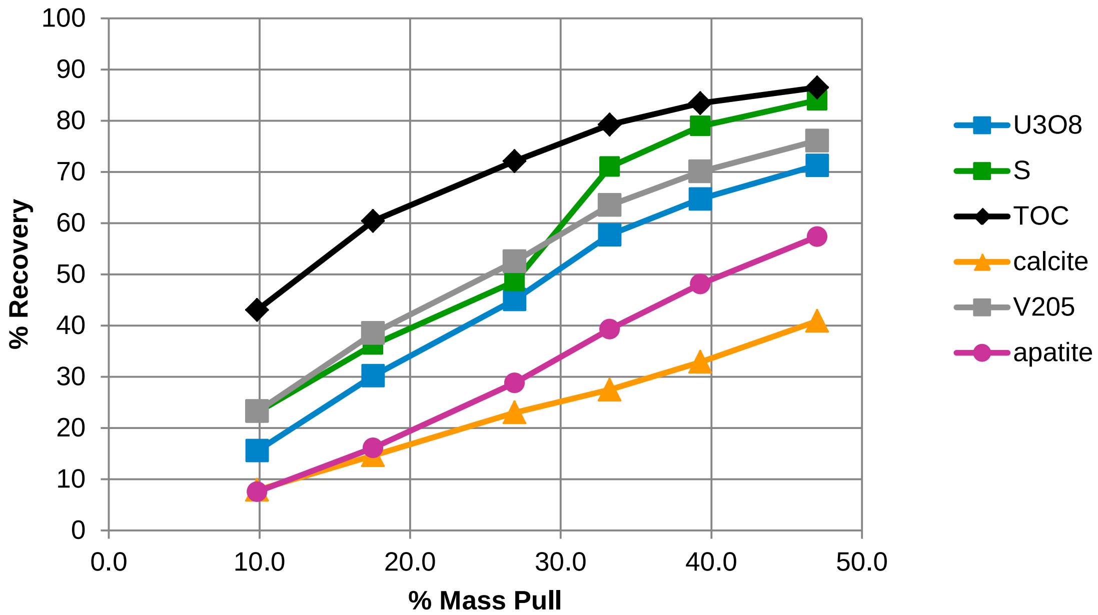

Figure 13-6: Percent recovery of elements and minerals versus mass pull for carbon pre-float and sulphide flotation for SGS Lakefield Test DDB-F3. (S is sulphur, TOC is total organic carbon)

Organic carbon assays were not performed in the extensive test work undertaken at Optimet. However, from the good correlation between organic carbon and sulphur recovery from the SGS Lakefield work (Figure 13-6), it is reasonable to assume that sulphur recovery can be used as a rough proxy for carbon recovery in the evaluation of test work undertaken at Optimet. This point is illustrated in a comparison of uranium against carbon recovery in the pre-float and sulphide flotation stages in the SGS Lakefield tests with uranium against sulphur assays in the pre-float and sulphide flotation tests done at Optimet (Figure 13-7).

Time is an important factor in selectivity with sulphide flotation:

- Sulphide flotation time of 10 minutes resulted in a mass pull of 7% or less, with approximately 35% of the sulphides, approximately 10% of the uranium, only 3% of both phosphate and calcite reporting to the concentrate; and

- An increase of the sulphide flotation time of 20 minutes resulted in a mass pull of approximately 25% with over 80% of the sulphides, approximately 50% of the uranium, 35% of the phosphate and 17% of the calcite reporting to the concentrate.

These results confirm the observation from the SGS Lakefield results that the conditions used for carbon pre-float and sulphide flotation are favourable to the separation of apatite from calcite; proportionately more apatite reports to the concentrate than calcite with longer flotation times.

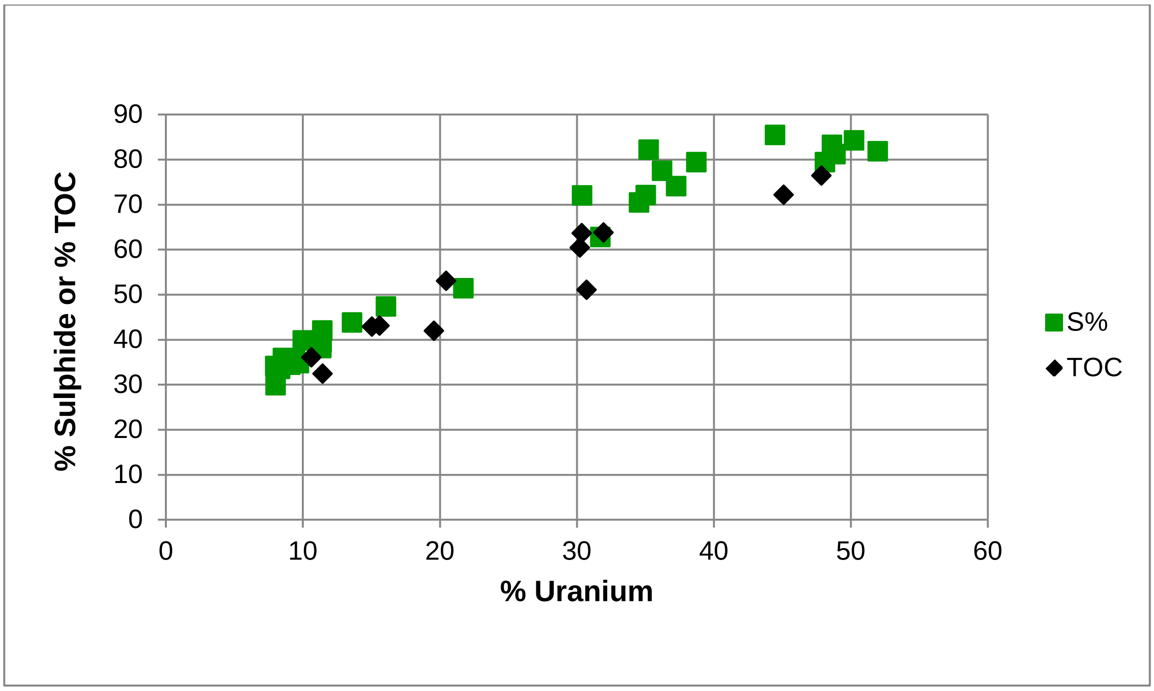

Figure 13-7: Plot of sulphur content from Optimet test work and total organic carbon content (TOC) from SGS Lakefield test work against uranium content of pre-float and sulphide flotation concentrate.

The linear relationship in the Optimet test work between sulphide and uranium content of the combined carbon pre-float and sulphide concentrate suggests that if conditions were set such that 100% of the sulphide was collected into the concentrate, then approximately 59% of the uranium would also report to the concentrate (Figure 13-8). This result is comparable with the SGS Lakefield work that indicated that if 100% of the organic carbon were to be recovered then approximately 65% of the uranium would also report to the concentrate. Therefore, both sets of tests give similar theoretical expectations for uranium recovery into a pre-float carbon and sulphide concentrate. In fact, SGS Lakefield Test DDB-F3 achieved a higher uranium recovery of- 71% from a mass pull of 47%. This result suggests that further efficiencies can be expected from flotation.

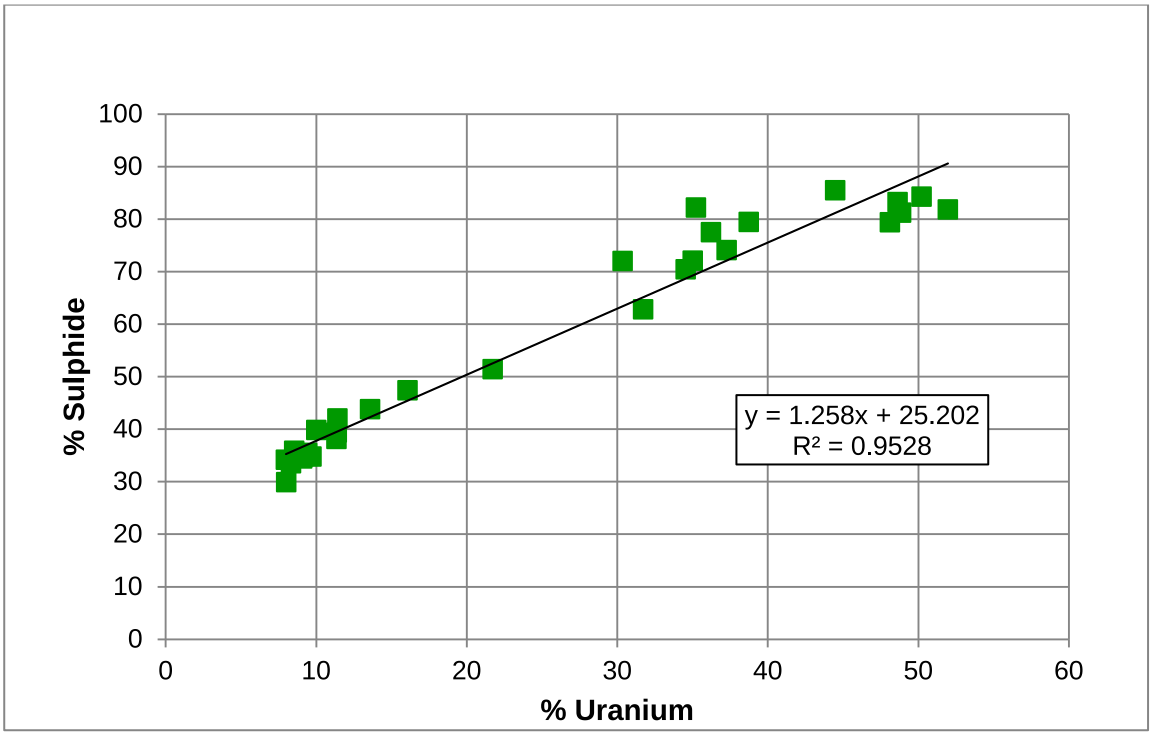

Figure 13-8: Percentage sulphide plotted against percentage uranium content of carbon pre-float and sulphide flotation concentrate for test work undertaken at Optimet

Figure 13-9 below shows the recovery of uranium, apatite and calcite against the mass of the material captured in the carbon pre-float and sulphide concentrate in the various tests undertaken at Optimet. These data show a steady improvement in that a greater proportion of apatite than calcite is captured in the concentrate as further rougher stages are added to the carbon pre-float together with longer flotation times. This progress resulted in the proportion of apatite to calcite reporting to the pre-float and sulphide concentrate increasing to a value of approximately 2.0. The addition of multiple cleaning stages for treating pre-rougher concentrates led to the improvement of this ratio to approximately 2.6. These data again demonstrate that flotation conditions that promote sulphide and carbon collection are also conducive to the concentration of apatite into the concentrate and provide a possible means of further increasing the extent of separation of apatite from calcite.

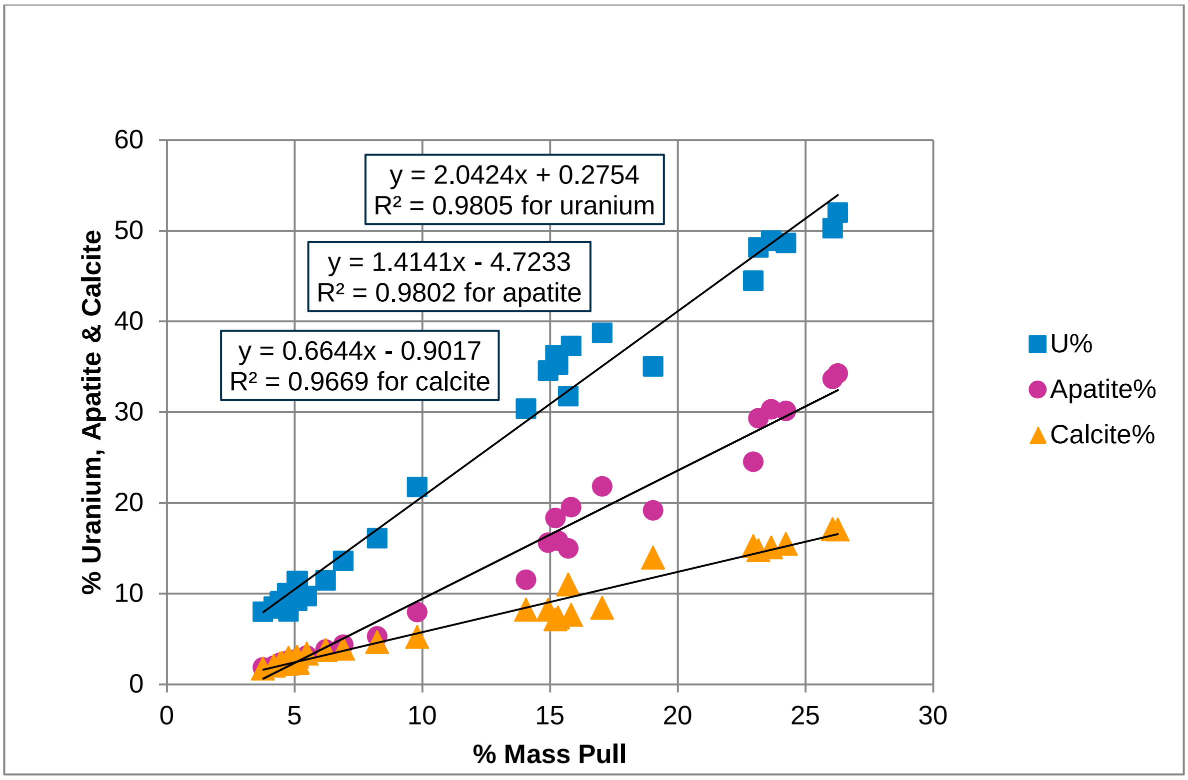

Figure 13-9: Percentage uranium, apatite and calcite reporting to the pre-float carbon and sulphide concentrate against percentage mass pull for tests undertaken at Optimet

13.5.6.5 Specific Separation of Calcite from Apatite by Flotation after Carbon Sulphide Flotation

Of critical importance to operating costs is the removal of as much acid-consuming calcite from the mineralised material as possible to reduce acid consumption. Extensive flotation test work has been undertaken at Optimet on the separation of calcite and apatite after extraction of the carbon pre-float and sulphide concentrate. The majority of these tests examined different reagent suites and identified certain modifiers that achieve superior separation of apatite from calcite. Figure 13-10 shows the percent uranium recovery achieved in the apatite/calcite separation flotation stage following the pre-float stage and the respective mass pull to the apatite concentrate. Uranium recovery to the concentrate is quite variable and is dependent upon the quantity of uranium recovered in the previous carbon/sulphide pre-float stage.

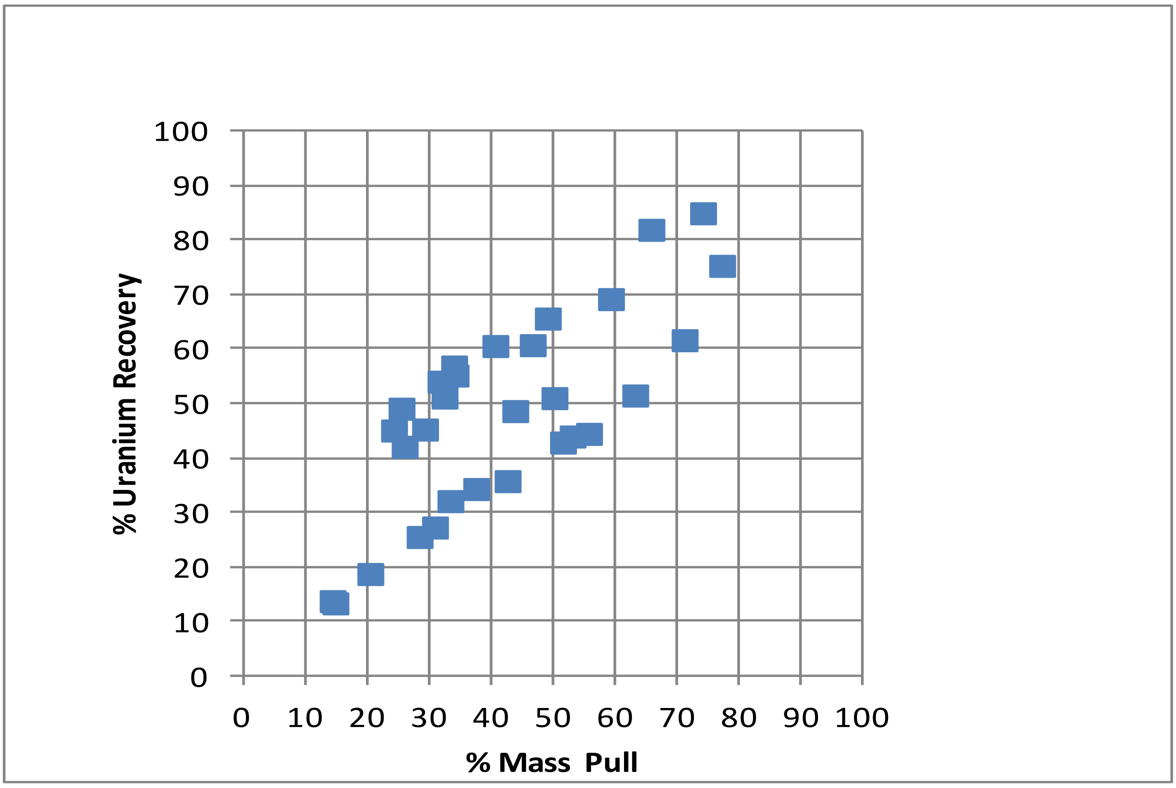

Figure 13-10: Percent uranium extraction versus percent mass pull to the apatite concentrate following pre-float for selected tests done at Optimet

Calcite rejection from the concentrate is generally poorer than was achieved in the pre-float carbon and sulphide concentrates (Figure 13-11). This figure shows the advances made in the test work with higher apatite to calcite ratios being achieved in the later tests for both concentrate types. The best results were obtained in Optimet Tests 30 and 32 in which, at a ratio of just less than 2.0, the percentage of apatite reporting to the concentrate was almost double the percentage of calcite reporting to it. The use of greater volumes of collector in the cleaning stages appears to have been responsible for the superior results.

These results are encouraging in that they provide evidence that more efficient separation of apatite from calcite is possible.

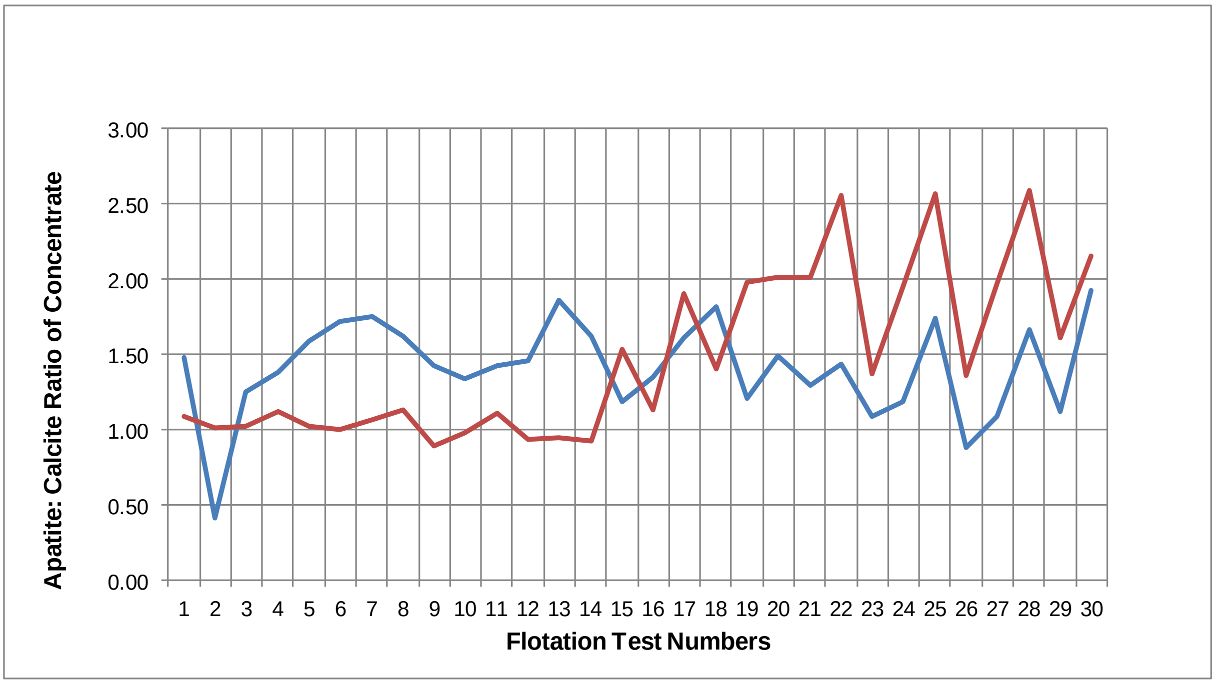

Figure 13-11: Comparison of the ratio of apatite to calcite reporting to carbon pre-float and sulphide concentrate (red line) compared with the same ratio for apatite flotation (blue line) in selected tests undertaken at Optimet

13.5.6.6 Results for Combined Pre-float, Sulphide Flotation and Apatite Flotation

Table 13-15 provides a summary of the results of the most successful flotation tests undertaken to date. Optimet Tests 32 and 33 show that a high mass pull of 82%-86% achieves recoveries to the combined rougher concentrates of 95%-96% of the uranium, 97-98% of the sulphide, 94%-96% of the apatite with 73-79% of the calcite. Elimination of 21% to 27% of the calcite from the concentrate, while maintaining recoveries in excess of 94% of the uranium, sulphide and apatite, is likely to result in significant cost savings from reduced sulphuric and acetic acid consumption in subsequent mineral processing. Separate cleaning of the rougher pre-float and selective flotation concentrates gave good rejection of calcite to give a final combined cleaner concentrate of 48% by weight containing only 33% of the calcite. However, uranium and phosphate losses were notable such that the final uranium and phosphate recovery to the combined cleaner concentrate was 66%-67% and 63%-65% respectively.

The best overall results are given by Tests 30, 32 and 33, which were due largely to a high mass pull in the pre-float giving high recovery of carbon/sulphides while maximising selective rejection of calcite over apatite, followed by careful cleaning steps to reduce the final mass of concentrate. Cleaning steps for both concentrates proved more successful than previous tests as a result of the addition of greater volumes of collector. Future work will continue in this direction in order to improve calcite rejection in cleaning while reducing uranium and phosphate losses.

Table 13-15: Summary of flotation test conditions for selected tests undertaken on mineralised material from Berlin at Optimet

Flotation Test Conditions and Conclusions										
Test no	Conditions	Recovery to Concentrates								Conclusion
		conc .type	stages	wt %	U%	S%	Si%	Apatite%	Calcite%	
29	45µm grind, similar to test 27 but with modifier 8 instead of 5 in all roughers	Prefloat rghr	4	19.04	35.03	72.06	19.37	19.15	13.94	Modifier 8 used was not as effective as modifier 5 for apatite calcite selectivty but improved selectivty against iron oxides
		Selective rghr	5	71.3	61.68	24.86	78.62	77.86	71.53	
		Combined	9	90.34	96.71	96.92	97.99	97.01	85.47	
30	45µm grind, similar to test 28 roughers but with addition collectors in respective cleaners to reduce uranium loss	Prefloat rghr	4	24.25	48.64	83.21	26.28	30.13	15.44	Additional collector used in cleaning stages was effective at improving the overall uranium recovery into combined cleaner conc without loss of apatite/calcite selectivity
		Prefloat clnr	4	15.22	36.24	77.46	18.3	18.32	7.14	
		Selective rghr	5	51.65	42.88	12.82	67.4	59.68	50.24	
		Selective clnr	4	28.49	25.53	7.35	55.5	38.38	22	
		Combine rghr	9	75.9	91.52	96.03	93.68	89.81	65.68	
		Combine clnr	8	43.71	61.77	84.81	73.8	56.7	29.14	
32	45µm grind, similar to test 30 roughers but with higher addition of collectors in respective cleaners	Prefloat rghr	4	26.07	50.24	84.25	27.83	33.66	17.06	Higher addition of collector in prefloat gave reduction in uranium loss to cleaner tail but no similar effect observed for selective rougher calcite/apatite float . Test gave best overall perfromance for uranium recovery and apatite/calcite separation
		Prefloat clnr	4	17.05	38.74	79.42	20.04	21.8	8.41	
		Selective rghr	5	55.85	44.47	12.64	67.68	60.76	55.85	
		Selective clnr	4	30.97	27.27	6.86	55.87	40.91	24.52	
		Combine rghr	9	81.92	94.71	96.89	95.51	94.42	72.91	
		Combine clnr	8	48.02	66.01	86.28	75.91	62.71	32.93	
33	25µm grind, otherwise similar to test 32	Prefloat rghr	4	22.97	44.48	85.43	27.01	24.53	15.22	finer grind gave similar response as 45µm test
		Prefloat clnr	4	15.32	35.24	82.15	20.13	15.81	7.34	
		Selective rghr	5	63.32	51.52	12.33	70.57	71.8	63.98	
		Selective clnr	4	33.46	32.12	6.85	59.46	48.95	25.46	
		Combine rghr	9	86.29	96	97.76	97.58	96.33	79.2	
		Combine clnr	8	48.78	67.36	89	79.59	64.76	32.8	

13.6 Metallurgical Test Work on Oxidised Mineralised Material

Composite sample Colombian Comp is weathered saprolite-like, clayey material from bore holes DDB-001 and DDB-002, while sample DDB-5 is sandstone material from bore hole DDB-005 and Berlin Comp is a mixture of sandstone and limestone from core from holes DDB-005 and DDB-007 (Table 13-1). Evidence is presented in Section 7 that the sandstone is weathered limestone from which the carbonate has been leached by organic acids in the oxidised part of the Berlin deposit.

Acid leach tests conducted on the oxidised material at temperatures of 50°C, 65°C and 80°C resulted in uranium extractions ranging from 33% at the lowest temperature to 71% at the highest temperature (Table 13-16). Vanadium, yttrium and rhenium also showed increasing extraction with increasing temperature. In contrast, maximum phosphate extraction of 62% was achieved at 65°C at a pulp density of 25% at a leach time of only four hours. Maximum molybdenum extraction of 29% was achieved at 65°C.

Acidic ferric iron leach tests undertaken at SGS Lakefield achieved high rates of extraction for uranium (Table 13-17). The average uranium extraction from the four 24 hour leach tests with the sulphuric acid wash was 97% decreasing to 94% in the 48 hour tests. Clearly the shorter leach time appears advantageous in order to maximise uranium extraction.

Table 13-16: Summary of conditions under which acid leach tests were done on oxidised material, and metal extractions achieved, on composite samples from the Berlin Project (Coffey Mining, 2012)

Laboratory	Lab Test #	Sample	Grind P_{80} (µm)	T (°C)	Pulp density (%)	Leach time (Hours)	Average pH	Acid			Metal Recovery (%)						
								Type	Acid added (kg/t)	Net Acid Consumption (kg/t)	U	V	P	Y	Mo	Ni	Re
SGS	11	Colombian Comp	41	80	50	48	0.94	H_2SO_4	194	158	70.8	51	31	67	15		19
SGS	14	DDB5	38	65	50	48	1.61	H_2SO_4	225	183	59.8	45	49	20	29		6
SGS	16	DDB5	15	50	25	4	0.95	HCl	175	104	45.5	9	62	50	17		
SGS	17	Berlin Comp	70	50	25	4	0.95	HCl	214	193	33.3	5	25	27	10		

Table 13-17: Summary of acidic ferric iron leach tests conducted on Berlin composite samples of oxidised material

Lab	Composite Sample #	Leach Test Number	Maximum Grain Size (µm)	Ferric Leach (Stage1)				Releach / Wash (Stage 2)			Elemental Extraction %								
				Concentration (g/L)	Pulp density (%)	Temp. (°C)	Time (hrs)	Acid type & conc.	Pulp Density (%)	Temp. (°C)	Uranium	Vanadium	Phosphate	Yttrium	Neodymium	Zinc	Nickel	Molybdenum	Rhenium
SGS Lakefield	DDB5 Comp	41/53	75	50	10	65	48	10%HCL	10	40	93.2								
		42/54	75	50	5			10%HCL	10		95.8								
Average extraction from Composite Samples with hydrochloric acid wash											94.5								
SGS Lakefield	DDB5 Comp	41/45	75	50	10	65	24	10%H_2SO_4	10	40	95.2								
		41/49	75	50	10		48	10%H_2SO_4	10		94.0								
		42/46	75	50	5		24	10%H_2SO_4	10		98.9								
		42/50	75	50	5		48	10%H_2SO_4	10		95.7								
Average extraction from Composite Sample DDB5 with sulphuric acid wash											96.0								
Average extraction from 48 hour ferric leach tests with sulphuric acid wash											94.9								
Average extraction from 24 hour ferric leach tests with sulphuric acid wash											97.1								

13.7 Preliminary Observations on the Nature of Tailings

Work on the nature of tailings is preliminary until the last details of the flow sheet and process have been established in a feasibility study. However, early indications are that the iron oxide and iron hydroxide species that are insoluble to the wash in the second step of the ferric iron leach process provide an effective chemical sink for many metals that may occur in the tailings from mineralised material at Berlin. Indications are that arsenic and antimony are likely to precipitate as an insoluble ferric arsenate, a naturally occurring mineral. An advantage of avoiding the use of hydrochloric acid in the extraction process is that, in the absence of chlorine, mercury would remain in the inert, solid state.

Further work is required on determining the concentration of radionuclides in the tailings. However, radionuclide levels should remain roughly at the current natural levels of the mineralised material and the current mine plan envisages that the tailings will be stored underground where the radionuclides naturally occurred in the first place. The concept presented in the mining scenario is that tailings will be mixed with cement and pumped back underground where they would provide structural support through the backfilling of mined-out areas.

The envisaged ferric iron leach and subsequent metal and phosphate extraction by IX, SX and direct precipitation would create neutral water streams and most of the plant's water would be recycled.

13.8 Process Confidence

13.8.1 Beneficiation

Two principal avenues are being investigated for the removal of carbonate from the mineralised material from the Berlin deposit. Carbonates, mainly calcite with minor dolomite, react with acid which is required for the extraction of metals and phosphate. The lower the carbonate content of the feed after beneficiation, the lower the acid consumption, which impacts operating costs.

The current PEA is based on using acetic acid to consume calcite. The use of acetic acid as a means of beneficiating carbonate-rich ores is a new concept. This technology has been proven on a laboratory scale by the original authors (Gharabaghi *et al*., 2010) and has been demonstrated to be effective on mineralised material from the Berlin deposit by SGS OreTest.

The second method of beneficiation that is being investigated for use at Berlin is flotation. This work requires further development but is using standard, proven technology and reagents. This work has been done at SGS Lakefield in Ontario, Canada, and at Optimet in South Australia.

13.8.2 Extraction Process

Extensive test work on 34% of the mineral intersections drilled in the resource area at Berlin has shown acidic ferric iron leach to be particularly effective in extracting phosphate and an extensive suite of metals from the mineralised material at Berlin.

Acidic ferric iron leach was first used commercially for the extraction of uranium and yttrium in the Elliot Lake mining camp in Ontario, Canada, in the 1960s and 1970s. It is currently used for uranium extraction at Rossing Uranium mine in Namibia and Buffelsfontein mine in South Africa. It is used for the extraction of a suite of metals, including uranium, from the multi-commodity Talvivaara mine in Finland. Certain aspects of the acidic ferric iron leach process have been developed and modified to optimise recoveries from the mineralised material from Berlin. The main addition to conventional acidic ferric iron leach is the inclusion of a dilute acid wash of the initial iron-rich residue that precipitates after the acidic ferric iron leach step.

The addition of the acid wash to the iron-rich residue generated by the original acidic ferric iron leach process and its application to a broad spectrum of metals, including rare earth elements and phosphate, led to U3O8 Corp. applying for a patent for the revisions to the original ferric iron leach process.

The ability to "exchange" reagents in terms of the provision of acid requirements for leaching either through ferric precipitation or by external acid addition is also considered to be a useful principle of the proposed process. However for the purposes of the current study a high acid addition, low ferric method of leach extraction has been adopted. This gives considerable confidence that a PLS would be produced which can be readily integrated into conventional recovery routines. However, it is intended that further metallurgical studies be undertaken to evaluate various PLS's of different final iron/acid compositions and their compatibility with metal recovery routines in order to improve the economics of the overall process.

13.8.3 Recovery Technology

Recovery of metals and phosphate from the PLS uses well known technology including SX, IX and direct precipitation using reagents that are commonly used in the extractive industry. Some of the characteristics of the ferric iron extraction process are designed to simplify downstream recovery from the PLS. For example, the use of sulphuric rather than hydrochloric acid in the releach step simplifies recovery of metals from the PLS.

Recovery methods envisaged are similar to those that were used in the Elliot Lake camp and are currently used in processing facilities at Rossing, Buffelsfontein and Talvivaara.

Although the recovery processes utilise well known technology, the steps in the conceptual flow sheet will have to be proved by a comprehensive test work program in the further development of the project.

13.9 Summary

- Aggressive alkaline leach achieved uranium extractions of 67%-68%, but with poor rates of extraction for associated metals and phosphate, with the exception of molybdenum, which achieved recoveries of 88%-91%.

- Sulphuric acid leach tests on unbeneficiated oxidised and unoxidised mineralised material yielded similar uranium extractions. Extractions ranged from 19% to 78% with higher extractions being achieved at higher temperatures of 80°C. Acid consumption ranged from approximately 100 kg to 200 kg/t of mineralised material in the oxidised material and up to 650 kg/t of mineralised material for the unoxidised material.

- A two-step acidic ferric iron leach followed by a wash of the precipitate with 10% strength sulphuric acid has proved to be the most efficient process for the extraction of metals and phosphate from mineralised material from Berlin. Principal aspects of the acidic ferric iron leach are as follows:

  - Acidic ferric iron leach tests have been carried out on composite samples from 34% of the mineralised intersections that were used for the initial resource estimate undertaken by Coffey Mining (2012) and on which the current PEA is being undertaken. Intercepts used in the test work are from throughout the resource area. In addition, acidic ferric iron leach test work has been done at two independent laboratories, one in Canada and another in Australia. These factors lead to the conclusion that the metallurgical test work is representative of the resource area.

- Acidic ferric iron leach yields excellent extractions for a wide range of elements of potential economic interest in mineralised material from Berlin. The acidic ferric iron leach does not show significant improvements in efficiency with grind size; a relatively standard 100 µm grain size is adequate for excellent metal and phosphate extraction. The first step of the acidic ferric iron leach was done at a relatively low temperature of 65°C at atmospheric pressure. The dilute acid wash was done at a temperature of 40°C. No noxious fumes were emitted.

- Slightly higher rates of extraction were achieved with a dilute hydrochloric acid wash in the second step of the acidic ferric iron leach process relative to those from a sulphuric acid wash. However, in order to avoid downstream processing complications through the introduction of chlorine into the system, it is recommended that the sulphuric acid wash be used in the extraction of metals and phosphate from mineralised material from Berlin.

- Uranium extractions from ferric iron leach followed by a dilute sulphuric acid wash ranged from 93%-98% with an average of 96% for both oxidised and unoxidised material (24 hour leach time).

- Ferric iron leach with dilute sulphuric acid wash yielded the following average percentage extraction from other elements of economic interest: vanadium, 66%; phosphate, 99%; yttrium, 86%; neodymium, 60%, zinc, 96%; nickel, 66%; molybdenum, 51% and rhenium, 33%.

- Optimal leach conditions for Step 1 of the leach process are 65°C temperature, pulp density of 15%-30% with a leach time of 24 hours at an acid concentration of 50 g/L and a ferric iron concentration of 25 g/L to 50 g/L.

- Optimal leach conditions for Step 2, the acid wash, are 40°C temperature, pulp density of 30%-40% with a leach time of 12-24 hours with dilute (10%) sulphuric acid.

- Within certain limits the ferric iron leach process is insensitive to the ratio of sulphuric acid and ferric iron used in the leach process and this can be adjusted to optimise reagent costs. Acid is important for the releach step to minimise co-precipitation of elements, and acid addition to the leach is required when operating at high pulp densities. The presence of ferric iron is important, to act as an oxidising agent. Ferric iron consumption ranges between 50 kg and 100 kg/t of unoxidised mineralised material. Sulphuric acid consumption ranges between 100 kg and 350 kg/t of unoxidised mineralised material. Current estimates of consumption on unoxidised material are 85 kg of ferric iron and 160 kg of sulphuric acid per tonne of mineralised material. Oxidised material will have significantly lower reagent consumption.

- A pre-leach with acetic acid provides an effective means of extracting carbonate from the mineralised material.

  - Acetic acid reduces the mass of the mineralised material by between 53% and 60% through the dissolution of carbonate. Less than 1% of the apatite is dissolved. Between 5% and 7% of the uranium contained in the mineralised material is extracted into the liquid and it appears that this will be available for extraction by downstream processing so that minimal uranium is lost from the system.

  - The high cost of acetic acid requires that it is regenerated through the use of sulphuric acid. The reaction of acetic acid with calcite generates calcium acetate in solution, which reacts with sulphuric acid to form acetic acid and a gypsum precipitate. Assay of the gypsum produced from the acetic acid process applied to mineralised material from Berlin shows the gypsum to be of very high purity, with the potential to be a saleable by-product that would partially offset the cost of the sulphuric acid used to recycle the acetic acid.

  - The extraction of calcite by the acetic acid pre-leach results in a decrease of reagent consumption in subsequent ferric iron leach. Ferric iron consumption decreases to 20 kg to 50 kg/t of mineralised material and sulphuric acid consumption decreases to 50-100 kg/t of mineralised material.

- Flotation provides an alternative means of beneficiating mineralised material from the Berlin deposit. Test work to date shows that:

  - There is some evidence that finer grind sizes contribute marginally to flotation efficiency. Grind sizes of 25 µm, 45 µm, 58 µm and 104 µm have been tested.

  - Approximately 50% of the uranium can be extracted into a carbon pre-float in a mass pull of approximately 30%.

  - The addition of a sulphide flotation step increases uranium extraction to approximately 71% with up to 87% of the carbon and 84% of the sulphide, 76% of the vanadium and 57% of the apatite and only 41% of the calcite.

  - A possible advantage of producing a sulphide concentrate is that in addition to nickel, zinc and pyrite, it may provide a means of concentrating molybdenite. The very high correlation coefficient of molybdenum and rhenium assays suggests that rhenium occurs in molybdenite, which is its typical habit. Extraction of nickel, molybdenum and rhenium may prove to be significantly higher from a sulphide concentrate than they are through ferric iron leach. Extraction of these elements is relatively low through ferric iron leach: approximately 66% for nickel, 51% for molybdenum and 33% for rhenium. Zinc is the exception with extraction of 96% with ferric iron leach.

  - Progress is being made with flotation tests designed to separate apatite from calcite after removal of most of the carbon and sulphide in a pre-flotation step. Longer flotation times of 20 minutes and higher concentrations of flotation reagents are yielding promising results.

  - The best result achieved from the whole flotation process including carbon pre-float, sulphide flotation and subsequent apatite flotation are with a high mass pull of 82%-86%. The resulting concentrate contains 95%-96% of the uranium, 97-98% of the sulphide, 94%-96% of the apatite with 73-79% of the calcite. Elimination of 21% to 27% of the calcite from the concentrate, while maintaining recoveries in excess of 94% of the uranium, sulphide and apatite is likely to result in significant cost savings from reduced requirements for sulphuric and acetic acid in subsequent mineral processing.

- Preliminary work on the nature of flotation tailings indicates that a benign tailing is likely to be produced from the metallurgical process that has been established for mineralised material from Berlin. It is envisaged that radionuclides in tailings will be pumped back underground where they occurred naturally in the first place.

The results achieved in the metallurgical test work undertaken to date have identified key areas for further work and optimisation as follows:

- Continue promising developments in the use of flotation to remove acid-consuming carbonates from the mineralised material. Specifically, the use of modifiers that have potential to make the separation of apatite and calcite more efficient, need to be investigated further. Also, the deportment and composition of the sulphide concentrate needs to be studied since there is potential to increase metal extraction from sulphide relative to the moderate extractions achieved for nickel, molybdenum and rhenium with acidic ferric iron leach.

- Further studies are to be conducted in acidic ferric iron leaching with the view of evaluating operating parameters of a laboratory scale bench unit on a continuous basis and integration of the acidic ferric iron leach step and acid wash step into a single train of reactors.

- As progress is made on finalizing the most suitable means of beneficiating the mineralised material, further investigation is required into SX, IX and direct precipitation processes for the recovery of uranium, phosphate and associated metals.

- Tailings and effluent treatment tests should also be undertaken to obtain knowledge of the behaviour of the final leach residue and barren solutions.

- Although the recovery processes utilise well known technology, the steps in the conceptual flow sheet will have to be proved by a comprehensive test work program in the further development of the project.

14 MINERAL RESOURCE ESTIMATES

A NI 43-101 non-compliant resource estimate was undertaken in 1981 and is reported under historic exploration by Coffey Mining (2012). This PEA is based on a NI 43-101 compliant mineral resource estimate undertaken for the Company by Coffey Mining with an effective date of January 17, 2012 and reported in a NI 43-101 technical report dated March 2, 2012 (Coffey Mining, 2012). The resource estimate was undertaken on the southern 3 km of a 10.5 km trend (Figure 14-1) that is known to be mineralised through trenching reported by Coffey Mining (2012).

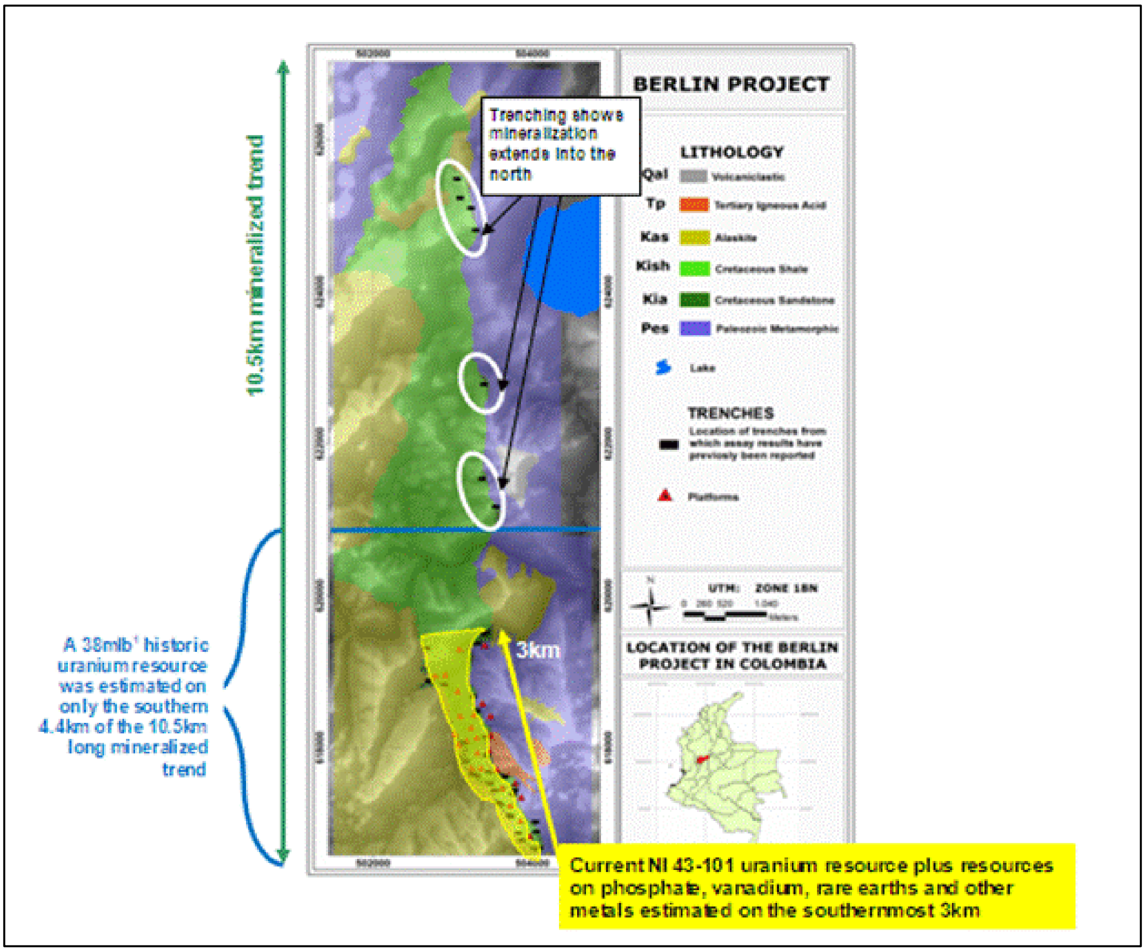

Figure 14-1: Geological map of the Berlin Project draped on an image of topography showing the distribution of the Cretaceous sequence (green) that contains the mineralisation. The yellow-shaded area shows the area in which the resource estimate was undertaken. Red triangles mark bore hole collar positions and black rectangles show the location of trenches excavated where the mineralised layer comes to surface

Coffey Mining updated the mineral resource estimate undertaken in 2012 (Coffey Mining, 2012) with the addition of estimates for calcium, magnesium and zinc at U3O8 Corp's request. The mineral resource estimate for these elements is detailed in this Section 14. The updated resource estimate is based on the same model as was used for the 2012 estimate for uranium, phosphate and other elements.

14.1 Uranium Mineral Resource Estimate

Resource size and grade are tabulated for various U_3O_8 cut-off grades in Table 14-1. It is evident that the uranium resource is insensitive to cut-off grade: for example, a doubling of the cut-off grade to 0.08% U_3O_8 makes no significant difference to the Indicated resource and decreases the Inferred resource by only ~4%. The recommended cut-off grade for NI 43-101 resource reporting is 0.04% U_3O_8.

Table 14-1: Tabulation of tonnage, grade and contained U_3O_8 for the southern 3 km of the Berlin trend (Coffey Mining, 2012)

U_3O_8 Cut-off Grade		Ore Tonnes (Millions)	Average Grade U_3O_8	Contained Uranium (U_3O_8) (mlbs)
(%)	(ppm)		(%)	
Indicated Resources				
0.00	0	0.6	0.11	1.5
0.01	100	0.6	0.11	1.5
0.02	200	0.6	0.11	1.5
0.03	300	0.6	0.11	1.5
0.04	400	0.6	0.11	1.5
0.05	500	0.6	0.11	1.5
0.06	600	0.6	0.11	1.5
0.07	700	0.6	0.11	1.5
0.08	800	0.6	0.11	1.5
0.09	900	0.6	0.11	1.5
0.10	1,000	0.5	0.11	1.2
Inferred Resources				
0.00	0	8.1	0.11	19.9
0.01	100	8.1	0.11	19.9
0.02	200	8.1	0.11	19.9
0.03	300	8.1	0.11	19.9
0.04	400	8.1	0.11	19.9
0.05	500	8.0	0.11	19.7
0.06	600	8.0	0.11	19.7
0.07	700	7.9	0.11	19.5
0.08	800	7.7	0.11	19.2
0.09	900	6.8	0.12	17.5
0.10	1,000	5.6	0.12	15

14.2 Mineral Resource Estimate for Phosphate, Vanadium, Yttrium, Neodymium, Nickel, Molybdenum, Rhenium and Silver

Mineral resource estimates were undertaken for phosphate, vanadium, yttrium, neodymium, nickel, molybdenum, rhenium and silver contained within the 0.04% U_3O_8 mineralised shell (Coffey Mining, 2012). A summary of the Indicated and Inferred resource estimate for other commodities in the southern 3 km of the 10.5 km mineralised trend in the Berlin Project at a cut-off grade of 0.04% (400 ppm) U_3O_8 is shown in Table 14-2.

Table 14-2: Mineral resource estimate for various commodities reported above various U_3O_8 lower cut-off grades using a bulk density of 2.72 t/m³ for fresh material and 2.0 t/m³ for weathered material. Ordinary Kriged Estimate for U_3O_8, multi-element data estimated using Inverse Distance to the power of 2, using 0.8 m assay composite data. Parent Block of 50 m (Y) 4 m (X) by 40 m (Z). *Preferred Reporting Cut-off – 0.04% U_3O_8.* (Coffey Mining, 2012)

U_3O_8 Cut-off Grade (%)	Ore Tonnes (Millions)	Other Commoditiies															
		Grade								Contained Metal							
		P_2O_5 (%)	V_2O_5 (%)	Y_2O_3 (ppm)	Mo (ppm)	Ni (%)	Ag (ppm)	Re (ppm)	Nd_2O_3 (ppm)	P_2O_5 (Million Tonnes)	V_2O_5 (mlbs)	Y_2O_3 (Tonnes)	Mo (mlbs)	Ni (mlbs)	Ag (Million Ounces)	Re (Tonnes)	Nd_2O_3 (Tonnes)
Indicated Resources																	
0.00	0.6	8.4	0.4	460	570	0.2	2.8	6.1	110	0.05	5.9	290	0.8	3.1	0.06	3.9	70
0.01	0.6	8.4	0.4	460	570	0.2	2.8	6.1	110	0.05	5.9	290	0.8	3.1	0.06	3.9	70
0.02	0.6	8.4	0.4	460	570	0.2	2.8	6.1	110	0.05	5.9	290	0.8	3.1	0.06	3.9	70
0.03	0.6	8.4	0.4	460	570	0.2	2.8	6.1	110	0.05	5.9	290	0.8	3.1	0.06	3.9	70
0.04	0.6	8.4	0.4	460	570	0.2	2.8	6.1	110	0.05	5.9	290	0.8	3.1	0.06	3.9	70
0.05	0.6	8.4	0.4	460	570	0.2	2.8	6.1	110	0.05	5.9	290	0.8	3.1	0.06	3.9	70
0.06	0.6	8.4	0.4	460	570	0.2	2.8	6.1	110	0.05	5.9	290	0.8	3.1	0.06	3.9	70
0.07	0.6	8.4	0.4	460	570	0.2	2.8	6.1	110	0.05	5.9	290	0.8	3.1	0.06	3.9	70
0.08	0.6	8.4	0.4	460	570	0.2	2.8	6.1	110	0.05	5.9	290	0.8	3.1	0.06	3.9	70
0.09	0.6	8.4	0.4	460	570	0.2	2.9	6.1	110	0.05	5.8	290	0.8	3.1	0.06	3.8	70
0.10	0.5	8.6	0.4	460	570	0.2	2.9	6.3	110	0.04	4.8	240	0.7	2.6	0.05	3.2	60
Inferred Resources																	
0.00	8.1	9.4	0.5	500	620	0.2	3.4	6.8	100	0.76	90.8	4,100	11.1	42.1	0.89	55.3	810
0.01	8.1	9.4	0.5	500	620	0.2	3.4	6.8	100	0.76	90.8	4,100	11.1	42.1	0.89	55.3	810
0.02	8.1	9.4	0.5	500	620	0.2	3.4	6.8	100	0.76	90.8	4,100	11.1	42.1	0.89	55.3	810
0.03	8.1	9.4	0.5	500	620	0.2	3.4	6.8	100	0.76	90.8	4,100	11.1	42.1	0.89	55.3	810
0.04	8.1	9.4	0.5	500	620	0.2	3.4	6.8	100	0.76	90.8	4,100	11.1	42.1	0.89	55.3	810
0.05	8.0	9.4	0.5	500	620	0.2	3.3	6.8	100	0.75	89.8	4,000	11.0	41.7	0.85	54.6	810
0.06	8.0	9.4	0.5	500	620	0.2	3.3	6.8	100	0.75	89.7	4,000	11.0	41.6	0.85	54.5	810
0.07	7.9	9.5	0.5	510	620	0.2	3.3	6.8	100	0.75	88.5	4,000	10.8	41.1	0.84	53.5	810
0.08	7.7	9.5	0.5	510	630	0.2	3.3	6.9	100	0.73	86.5	3,900	10.6	40.1	0.81	52.8	790
0.09	6.8	9.7	0.5	520	630	0.2	3.4	7	110	0.66	77.9	3,600	9.4	36.1	0.74	47.4	730
0.10	5.6	10	0.5	540	640	0.2	3.5	7.2	110	0.56	65.8	3,000	8.0	30.7	0.63	40.3	620

May not add up due to rounding

14.3 Resource Estimate for Calcite and Zinc

As a result of metallurgical test work (Section 13), it became apparent that resource estimates were required on some elements that were not included in the initial resource estimate completed by Coffey Mining in 2012. Metallurgical test work showed that zinc is efficiently extracted into solution by an acidic ferric iron leach and can easily be recovered from solution at little additional cost, and hence, although the grade of zinc in the mineralised material is not high, it provides a modest, positive contribution to modelled revenue.

In addition, a pre-leach with acetic acid generates calcium acetate from reaction with calcite as described in Section 13.5.5. Acetic acid is regenerated by treating the calcium acetate with sulphuric acid, which generates gypsum, a potential by-product of significant value. In order to include gypsum revenues in the PEA, an estimate of calcite resource was required. Hence, Coffey Mining was requested to undertake a resource estimate for zinc and calcite using the same resource model applied in the 2012 resource estimate (Coffey Mining, 2012).

The estimation of the calcite resource is based on the assignment of calcium between the principal calcium-bearing minerals in the Berlin deposit namely, calcite, apatite and dolomite. The assignment of the appropriate amount of calcium to apatite and dolomite was dependent on resource estimates of magnesium and phosphate. Phosphate resources were estimated previously (Coffey Mining, 2012), while the mineral resource estimate for calcium and magnesium, and the calcite resource derived therefrom, are presented with that for zinc in Table 14-3.

Table 14-3: Mineral resource estimate for various commodities reported above various U_3O_8 lower cut-off grades using a bulk density of 2.72 t/m³ for fresh material and 2.0 t/m³ for weathered material. Ordinary Kriged Estimate for U_3O_8, multi-element data estimated using Inverse Distance to the power of 2, using 0.8 m assay composite data, calcite%* calculated from the estimated block grades for P_2O_5, MgO and CaO. Parent Block of 50 m (Y) 4 m (X) by 40 m (Z). *Preferred Reporting Cut-off – 0.04% U_3O_8*. Percentage calcite was estimated using the following formula: Calcite% = { CaO% - (MgO% / 0.2186 * 0.3041) – (P_2O_5% / 0.4222 * 0.556) } / 0.5603.

Lower Cutoff (% U_3O_8)	Mt	CaO (%)	MgO (%)	Calcite (%)	Zinc (%)
Indicated					
0.04	**0.6**	**39.4**	**0.7**	**48.8**	**0.3**
0.05	0.6	39.4	0.7	48.8	0.3
0.06	0.6	39.4	0.7	48.8	0.3
0.07	0.6	39.4	0.7	48.8	0.3
0.08	0.6	39.4	0.7	48.8	0.3
0.09	0.6	39.3	0.7	48.7	0.3
0.1	0.5	39.2	0.8	47.9	0.3
Inferred					
0.04	**8.1**	**29.9**	**0.5**	**36.5**	**0.3**
0.05	8	30.1	0.5	36.7	0.3
0.06	8	30.1	0.5	36.7	0.3
0.07	7.9	29.9	0.5	36.5	0.3
0.08	7.7	30.0	0.5	36.5	0.3
0.09	6.8	29.0	0.5	35.1	0.3
0.1	5.6	28.1	0.5	33.6	0.3

14.4 Mineral Resource Estimate Methodology and Assumptions

The bore hole database in the vicinity of the estimation consists of 82 diamond drill holes totalling 18,551 m and 30 surface trenches totalling 100 m.

Drill hole spacing ranges from 60 m x 100 m to 100 m x 200 m. Both the diamond drill holes and the trenches were used to model the mineralisation geometry, however only the assay data from the diamond drill holes were used in the grade estimate.

A total of 317 mineralised intervals were used in the resource model. Of these, 69% were chemically assayed utilizing values using ICP-MS, supported by XRF analysis for samples with elevated phosphate. Samples were prepared by ALS Chemex in its facility in Bogota, Colombia, and the samples were shipped to ALS Chemex's assay laboratory in Vancouver, Canada, and Lima, Peru, for analysis. Radiometric data were used for 99 intervals. The assay data was composited to 0.8 m intervals for the resource estimation. Variography and search neighbourhood analysis were also conducted as an input into the U_3O_8 grade estimation.

The drill holes were typically drilled along east- to northeast-trending section lines, with dip ranging from 60° to 90°. The drilling angle and orientation was designed to intersect the mineralisation so as to produce a true-thickness intercept. However, a small number of drill holes intersected the mineralisation at acute angles where the shape of the folded mineralised layer is different in detail from the simple fold shape targeted. In cases where the intersection was oblique to measured bedding in the bore hole core, an estimate of true thickness was calculated.

The drill core was halved using a diamond saw, then character-sampled based upon estimated grade from measured radioactivity and lithology to a minimum nominal length of 20 cm, and a maximum length of 1.6 m. The average sample length was 80 cm.

Gamma ray measurements, recorded with a down-hole Mount Sopris probe, calibrated in Grand Junction, Colorado, were used to estimate the equivalent uranium grade ("eU_3O_8") where chemical assays were not available due to poor core recovery. eU_3O_8 values were used for approximately 31% of the assay intervals. Gamma ray data were measured for most of the bore holes drilled, providing a comparison between chemical assays and eU_3O_8 data. Assay and estimated eU_3O_8 values showed a very good correlation (Figure 14-2).

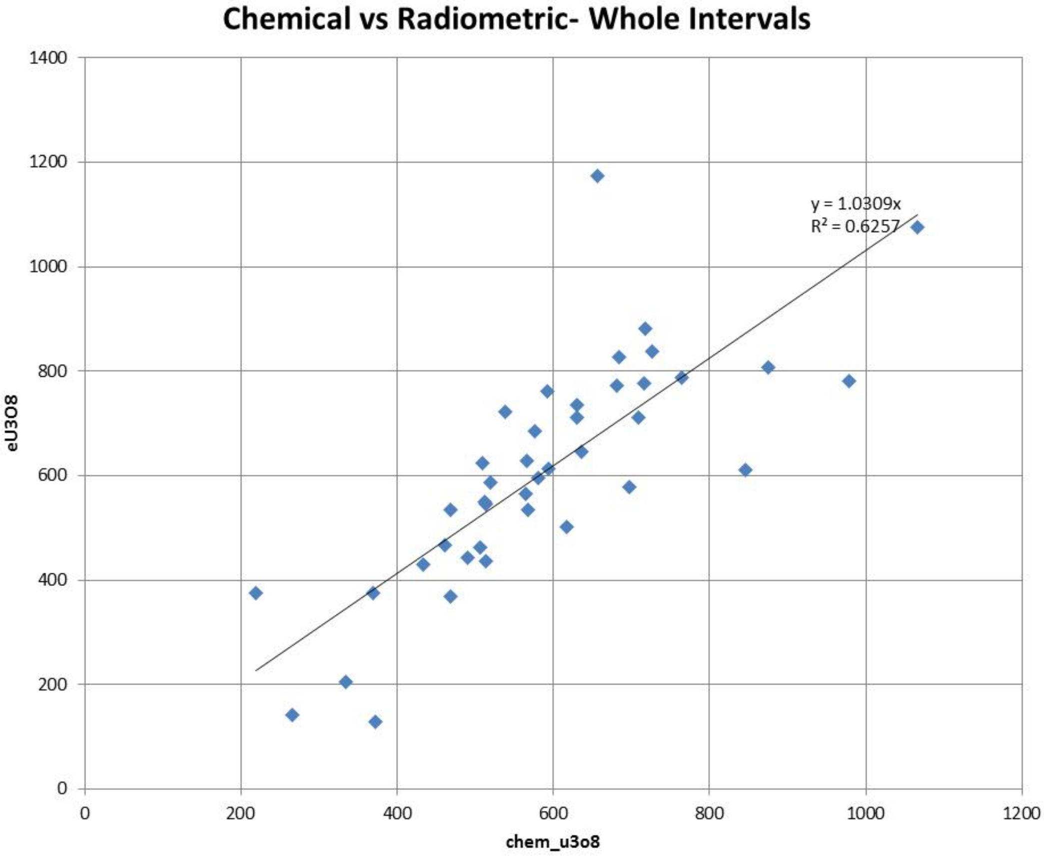

Figure 14-2: Scatter plot of chemical versus radiometric U_3O_8 data from bore holes included in the mineral resource estimate undertaken by Coffey Mining (2012).

Density for the resource was based upon 27 measurements taken from within the mineralisation using the water immersion method.

The mineralisation outline was based upon a nominal 0.04% U_3O_8 halo, with consideration given to the mineralised limestone lithology. Based upon the lithology modelling, a nominal weathering surface was defined as being 10 m below the topographic surface. The topographic surface was based upon a digital elevation model ("DEM") supplied surface obtained from ground-controlled precision satellite photographs using high-definition satellite elevation.

Ordinary Kriging was utilized for the U_3O_8 grade estimate and inverse distance to the power of two was used to estimate the multi-element data (including the extra elements used to estimate the calcite percentage (CaO% and MgO%) and Zn%). Top-cutting of the assay data was not considered necessary for the estimates.

The estimate was classified as Indicated and Inferred based upon the demonstrated geological and grade continuity, and the drill hole spacing. A map of the resource classification is shown in Figure 14-3.

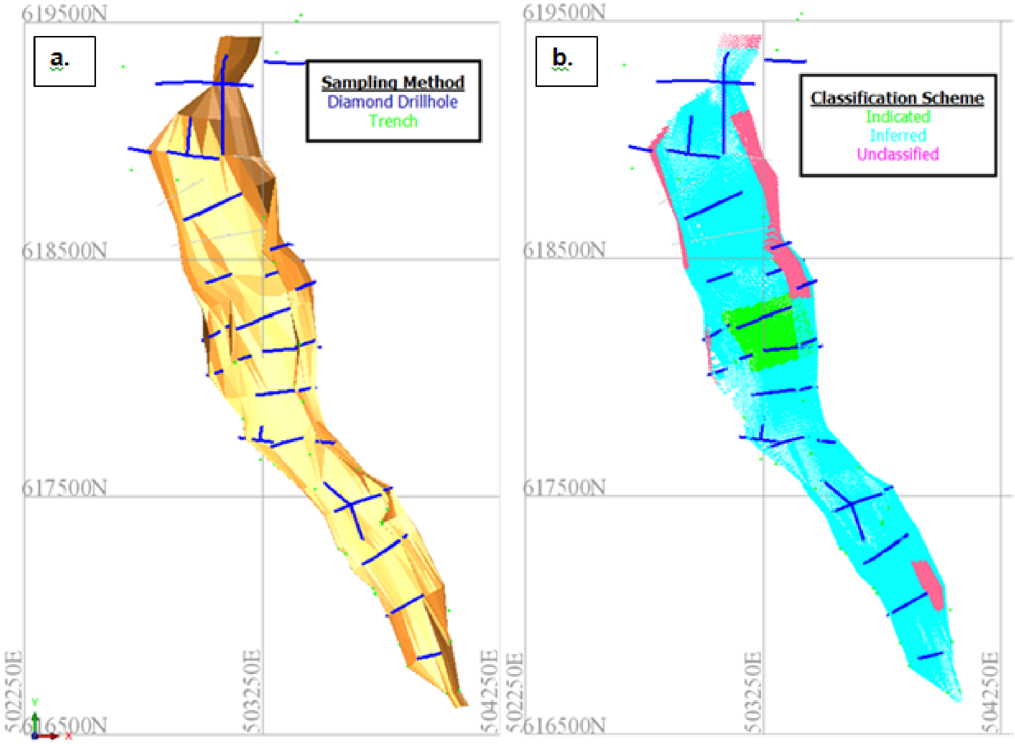

Figure 14-3: Map of the southernmost part of the Berlin syncline showing: a) the location of bore hole sections, trenches and continuity of mineralization used in the resource estimate; and b) resource classification

When estimating the calcite, the leached portion of the deposit (Figure 14-4), was set as absent, to provide a conservative calcite estimate for the deposit.

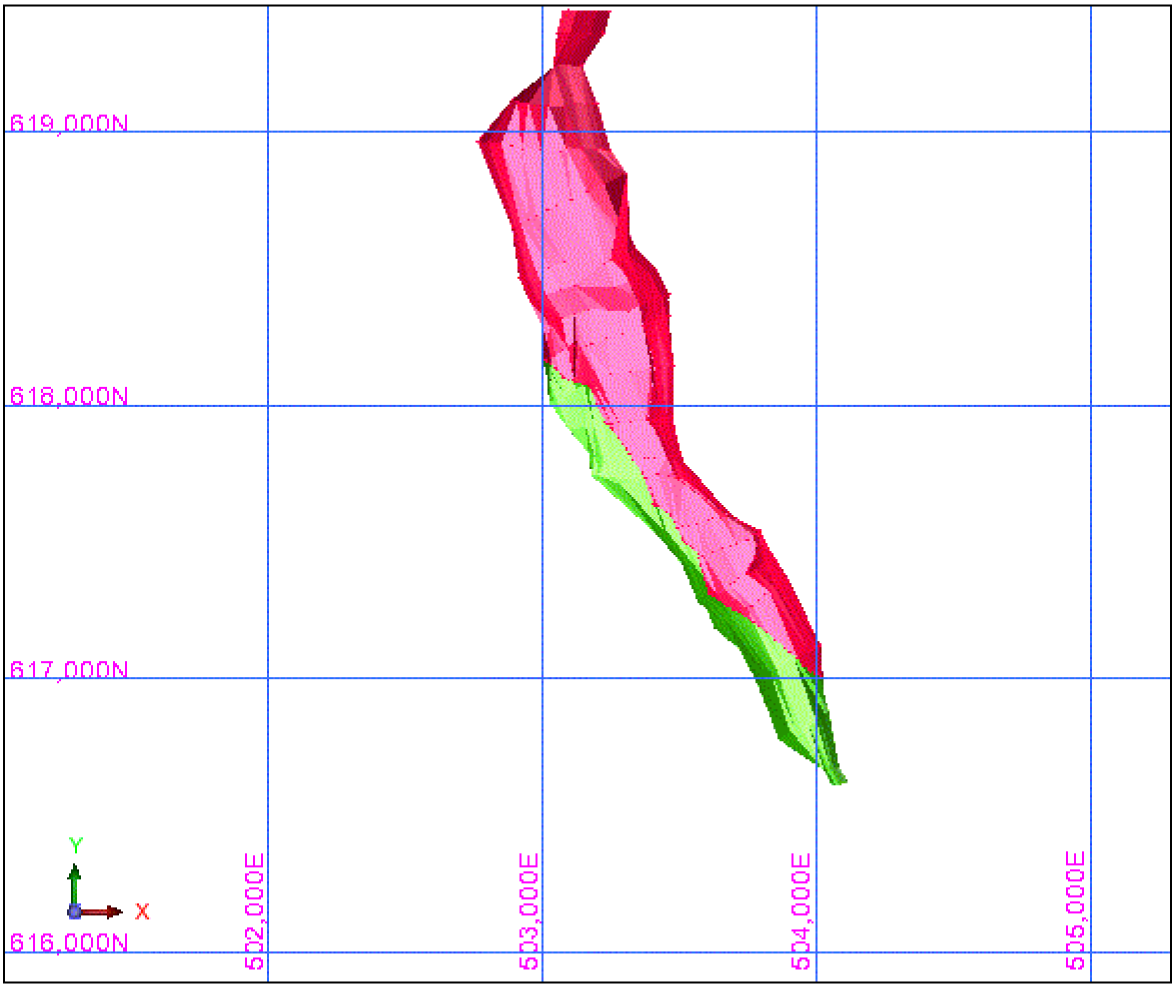

Figure 14-4: Map of the mineralised layer in the southernmost part of the Berlin syncline. Unoxidised mineralisation is shown in pink and oxidised mineralisation in green

The estimate of calcite percentage in the resource is comparable with estimates made from analyses and petrographic observation (ANSTO, 2011) (Section 13.4). CaO%, MgO% were estimated into the 2012 mineral resource model (Coffey Mining, 2012), the calculation of the mineral distribution used is as follows:

- Firstly, assigned the MgO% to dolomite using 21.9% as the MgO% content of dolomite;
- Secondly, assigned the P_2O_5% to apatite using 42.2% as the P_2O_5% content of apatite; and
- Finally, calculated the CaO% associated with dolomite (30.4%) and apatite (55.6%) and deducted that from the total CaO and assigned remaining CaO to calcite using 56.0% as the CaO% content of calcite.

14.5 Summary

- The initial resource estimated on the Berlin deposit is from drilling and trenching in a 3 km sector of a 10.5 km mineralised trend. The mineral resource estimate, at a cut-off grade of 0.04% U_3O_8, is 0.6 Mt of mineralised material at a grade of 0.11% U_3O_8 for 1.5 Mlb of contained uranium in the Indicated resource category and an Inferred resource of 8.1 Mt at a grade of and 0.11% U_3O_8 containing 19.9 Mlb of uranium.

- A resource estimate was also made, at a cut-off grade of 0.04% U_3O_8, for commodities that occur with the uranium at Berlin, and are in Table 14-4 (Coffey Mining, 2012).

Table 14-4: Summary of mineral resource estimates for commodities in the Berlin deposit (Coffey Mining, 2012 and this report). (Abbreviations: million tonnes (Mt), tonnes (t), million pounds (Mlb), million ounces (Moz)).

Commodity	Tonnage of Mineralised Material	Average grade	Contained metal or phosphate	Tonnage of Mineralised Material	Average grade	Contained metal or phosphate
	Indicated Resource			Inferred Resource		
Phosphate	0.6 Mt	8.4%	0.05 Mt	8.1 Mt	9.4%	0.76 Mt
Vanadium		0.4%	5.9 Mlb		0.5%	90 Mlb
Yttrium		460 ppm	290 t		500 ppm	4,100 t
Molybdenum		570 ppm	0.8 Mlb		620 ppm	11 Mlb
Nickel		0.2%	3.1 Mlb		0.2%	42 Mlb
Silver		2.8 ppm	0.06 Moz		3.4 ppm	0.89 Moz
Rhenium		6.1 ppm	3.9 t		6.8 ppm	55 t
Neodymium		110 ppm	70 t		100 ppm	810 t
Zinc		0.3%	4.4 Mlb		0.3%	45 Mlb

Figures may not add up due to rounding.

- In order for revenue from potential by-product gypsum to be incorporated in the cash flow model for this PEA, a resource estimate was required for calcite in the Berlin deposit. The logic is that mineralised material from Berlin can be beneficiated by the use of acetic acid which reacts with calcite, thereby removing a principal consumer of sulphuric acid that is used in the extraction of metals and phosphate from the mineralised material. Calcite reacts with acetic acid to form calcium acetate, and acetic acid can be regenerated from the acetate by reaction with less expensive sulphuric acid. The latter reaction forms gypsum, which constitutes a potentially saleable product that contributes to the modelled cash flow for one of the scenarios presented in the PEA. The calcite resource, at a cut-off grade of 0.04% U_3O_8, is estimated to be: Indicated – 0.6 Mt at a grade of 48.8% calcite for 0.29 Mt of contained calcite and Inferred – 8.1 Mt at a grade of 36.5% calcite for 3.0 Mt of contained calcite.

14.6 Further Work

- Further density data is required to characterise the density of the deposit, particularly in the weathered portion of the deposit.
- Further work is required to refine the base of weathering.
- Although the overall geometry and trend of the mineralisation is well defined, infill drilling is required to adequately define the areas of complex folding and structure. Further investigations are also required to determine if faulting is affecting the mineralisation, particularly for portions considered to be suitable for underground mining.

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15 MINERAL RESERVE ESTIMATES

There has been no reserve estimate stated for the Berlin Project.

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16 MINING METHODS

16.1 Characteristics of the Mineralised Body

16.1.1 Deposit Geometry and Mineralisation

Mineralisation at Berlin is confined to a specific layer that has the shape of the hull of a canoe that lists to the west – this is a doubly-plunging synclinal fold. The 10.5 km long axis of this canoe-like form trends northwards. The resource area is located in the southern part of the area, where the axis of the canoe-like fold gradually deepens from outcrop in the south to a depth of 400 m over a distance of 2.4 km to the north (Figure 16-1).

The west limb of the fold dips to the east. At depth, it flattens through the axis of the fold before gradually steepening into the east limb (Figure 16-2). The east limb varies in orientation from an eastward dip to steep to vertical orientations.

The mineralised layer averages 3 m thick and is composed of carbonate rock at depth and sandstone-siltstone in the weathered, oxidised zone near surface. The mineralised layer has a footwall of competent sandstone and conglomerate that forms a unit approximately 20 m thick. The sandstone-conglomerate is underlain by metamorphic schists or extremely competent granitoid rocks. The hanging wall consists of black shale that has moderate to poor rock strength.

The mineralised unit is locally disrupted by intrusive bodies and faults that dip to the east and have the effect of eliminating the mineralised layer, leaving windows from which the mineralised layer has been removed. Fault zones are characterised by extensively fractured ground.

The mineralised unit demonstrates remarkable continuity of grade throughout the resource area. The footwall cut-off for uranium and associated mineralisation is sharp and is discussed in more detail in Section 7.6. The upper contact of the mineralisation is likewise sharp for uranium, defining a tabular mineralised unit. Grades of some elements of value do extend a few metres further into the black shale hanging wall. Despite this, the cut-off grade for the resource estimate and for conceptual mine modelling is defined to be 0.04% U_3O_8, irrespective of the size of the halo of other elements that may extend into the hanging wall beyond the uranium mineralisation.

Radiation emitted from the rock mass is relatively low at Berlin and does not approach levels at which remote-controlled mining would be recommended.

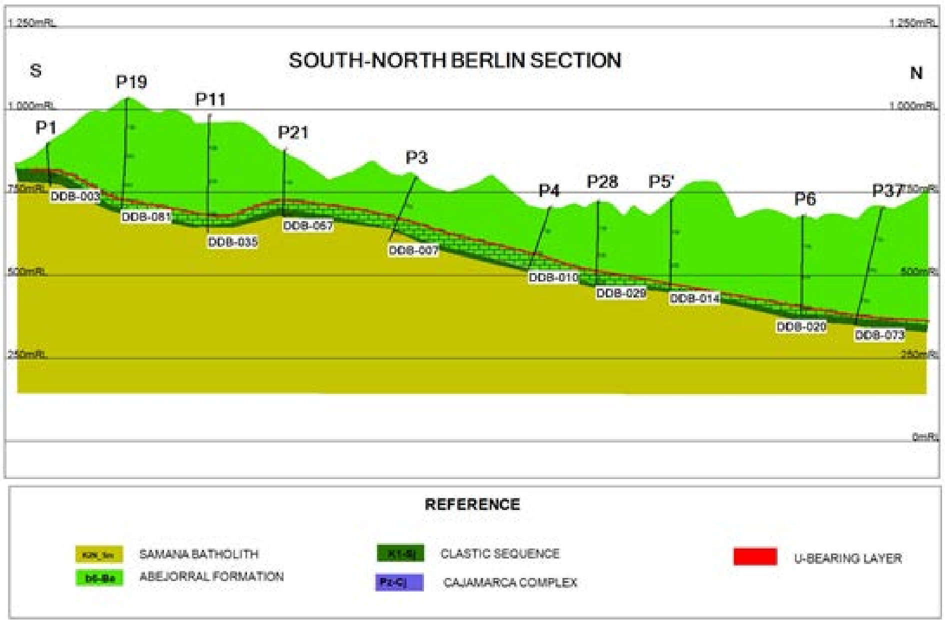

Figure 16-1: North-south long section shows axis of the Berlin syncline gradually deepening to the north

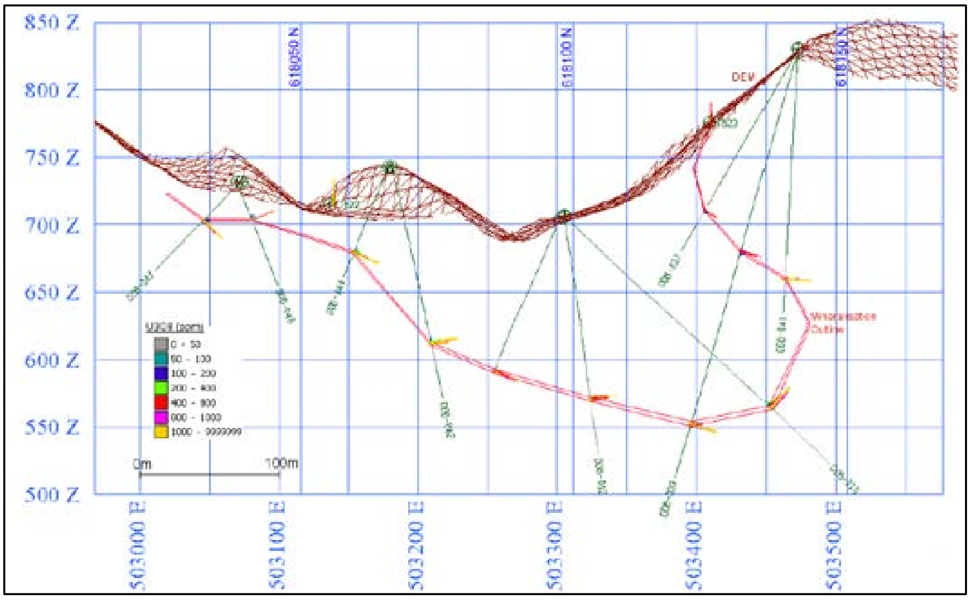

Figure 16-2: East-west cross section shows shape of the mineralised horizon (red) showing the eastward dip of the west limb and the variable orientation of the steeper east limb. Surface topography is shown in brown

16.1.2 Geotechnical Characteristics

Although no dedicated geotechnical drilling and analysis has been done to date, preliminary geotechnical characterisation of the mineralised zone and surrounding rock mass has been carried out through the logging of core from the 82 bore holes drilled in the resource area, and by review of associated geological models made from the data. A geotechnical model of the project site has been developed using these data and other characterisation features as outlined below.

16.1.2.1 Rock Mass Classification

Rock mass classification was undertaken on the basis of Rock Quality Designation ("RQD") measurements taken on whole core, visual evaluation of bore hole core as well as data collected during the consultant's visit to the site and core shed. Rock Mass Rating ("RMR") was used to define the principal characteristics of the rock within the resource area. RMR was done by logging the drill core, measuring strength, joint frequency and condition, RQD, water content, and strike and dip of the defined zones of structural weakness such as faults.

The immediate hanging wall to the mineralised zone at Berlin typically has a lower range in RMR values of 25 to 40. Weathered, oxidised rock varies from competent sandstone-textured material to clayey material. The weathered zone is somewhat irregular extending from surface to anywhere between 2 m and 20 m depth.

Fault zones are characterised by corridors of broken rock with lower RMR values. The preliminary mine concept does not attempt to specifically avoid zones of structural weakness in its design, but rather accounts for these corridors of weakness by generally applying conservative ground control systems. Ground conditions will be taken into account in future mine design as more detailed technical information becomes available.

16.1.2.2 Rock Quality

For stope stability purposes, estimated rock quality within 3 m - 5 m in the hanging wall and footwall of the mineralised zone was averaged for each bore hole.

16.1.2.3 Rock Strength

Mineralised rock strength is variable and can be described as moderately weak to weak and the rock is moderately to highly fractured. The immediate footwall rock has generally been qualitatively estimated as moderately strong.

16.1.2.4 Hydrogeology as Applied to Ground Control

Observations from volumes of water detected while drilling suggest that sub-surface waters are largely confined to shear zones and fault corridors at depths of less than 500 m below surface at Berlin. Pump tests will need to be performed on various rock intervals to define the hydrogeological characteristics of the resource area.

16.2 Mining Method Selection

The principal factors taken into account in the selection of mining methods were: the thin, tabular nature of the host-rock, the predictable and regular geometry of the mineralisation, its inclined "U"-shape in cross section, the extension of the mineralised layer from surface to considerable depth, its relatively low radioactivity and its relatively weak hanging wall.

These factors dictate that underground mining methods are appropriate for the Berlin resource area, and furthermore, that two mining methods are used:

- The steep-dipping east limb would be mined by cut and fill techniques; and
- The more shallowly dipping west limb would be mined with room and pillar methods where the dip of the mineralised zone is less than 45°.

Both cut and fill and room and pillar are conventional mining methods that have been used extensively in many mines, and for which appropriate experience is readily available in the mining industry.

16.3 Mining Method Description

16.3.1 Cut and Fill Method

The cut and fill mining method is typically used where the mineralised body has a steep dip. A typical development sequence for a cut and fill mining block is as follows (Figure 16-3 and Figure 16-4):

- A ramp is cut from the main access level towards the midpoint of the area to be mined. The ramp typically has as a grade of -15%. From the point at which the ramp or decline intersects the mineralised block, mining commences by drift development along the mineralised body on both sides of the decline. Once a block, typically 5 m to 10 m high, has been mined, the horizontal mined-out cavity is backfilled.
- The access ramp is then developed at a shallower angle to reach the part of the mineralised body that lies immediately above the recently-placed backfill. Mining equipment uses the recent backfill as the floor from which it operates.
- This procedure is repeated until the last access ramp has a grade of +15%, and mining of the section of mineralised material that can be reached from that ramp has been completed and the cavity backfilled. This concludes the mining of one section of a sub-level, each of which is typically 20 m to 50 m high.
- Mining then commences from a ramp cut towards the middle of the next mineralised block to be mined.

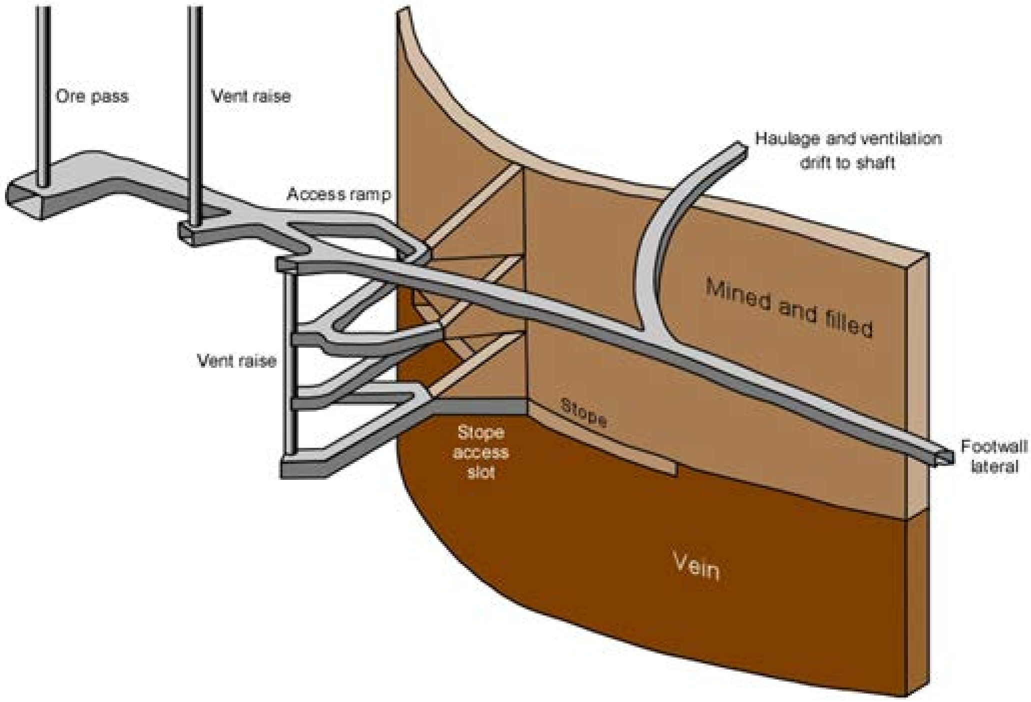

Figure 16-3: A schematic of the cut and fill mining method (OMSHR, 2012)

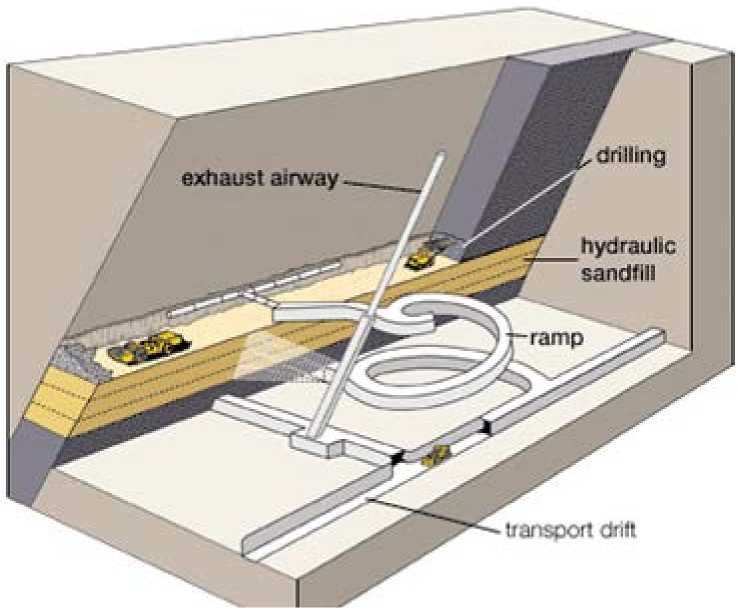

Figure 16-4: Schematic of the cut and fill mining method. Note that the upper part of the zone to be mined (above the level of the mining equipment) has been cut away to simplify the diagram reference (Atlas Copco, 1997)

16.3.2 Room and Pillar Method

Room and pillar mining is applicable to deposits that are tabular and flat to moderately inclined, and of relatively constant thickness. Mining of the mineralised body typically creates large open stopes where trackless machines travel on a flat or shallowly inclined floor (Figure 16-5). A minimum of development work is required to prepare for room and pillar mining. Excavation of roadways can be combined with production of the mineralised material, while mined-out stopes serve as transport routes.

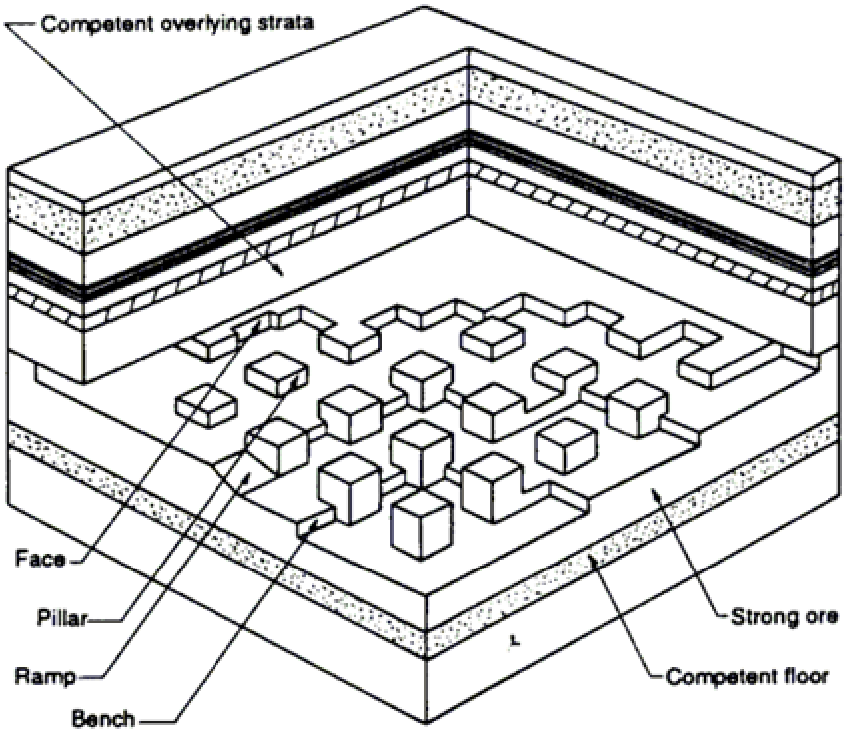

Figure 16-5: Schematic of the room and pillar mining method (Camm, 1991)

16.4 Ground Support & Rock Mechanics

16.4.1 Rock Stress Conditions

In situ vertical loading for Berlin operations is predicted to reach a maximum of 10 MPa at the deepest part of the resource area at some 400 m below surface. In addition, the 2008 version of the World Stress Map (Heidbach *et al*., 2008), indicates the presence of excess horizontal stress in the Berlin area and this will need confirmation and quantification as the project advances.

Preliminary modelling, with Rock Science Examine 2D software, of a typical cut and fill mining section for Berlin with 2.9 m stope width shows 6 MPa Sigma 1 (vertical stress) and 1.5 MPa Sigma 3 (horizontal stress). Stresses related to room and pillar mining are likely to be different due to the shallow dip of the west limb. Estimates of stresses in the west limb are 60 MPa for Sigma 1 and 15-20 MPa for Sigma 3.

16.4.2 Stability Analysis

Rock quality parameters were estimated on the basis of measured RMR Index data from core and input into Examine 2.0 software to provide a first-pass estimation of average stable stope size for steep-dipping and shallow-dipping domains.

A 20 m sublevel spacing was used as the base case for cut and fill mining methods at Berlin. A stope length of 30 m proved viable for average rock quality conditions. Modelling showed that stope lengths of 10 m were appropriate in areas of low rock quality, and stope lengths of up to 40 m could be used in areas of relatively good rock quality.

RMR data recorded from bore hole core from the resource area yielded values of 25 to 40 for areas of poor rock strength. Using the method of Bieniawski (1989), these values show that unsupported spans of 3 m x 3 m would be appropriate for room and pillar mining in the weaker parts of the shallow-dipping west limb of the Berlin resource (Figure 16-6) for a minimum of 10 hour to 100 hour operations. A wider span would require support. Room and pillar stope support would be with roof bolts and steel arcs as required as part of the standard operation. Initial pillar feasibility was designed using empirical methods. The most fundamental design factors in empirical pillar formulas are the degree of reduction taken for *in situ* compressive strength versus laboratory-measured strength, and a strength modification for height to width ratio to account for narrowness.

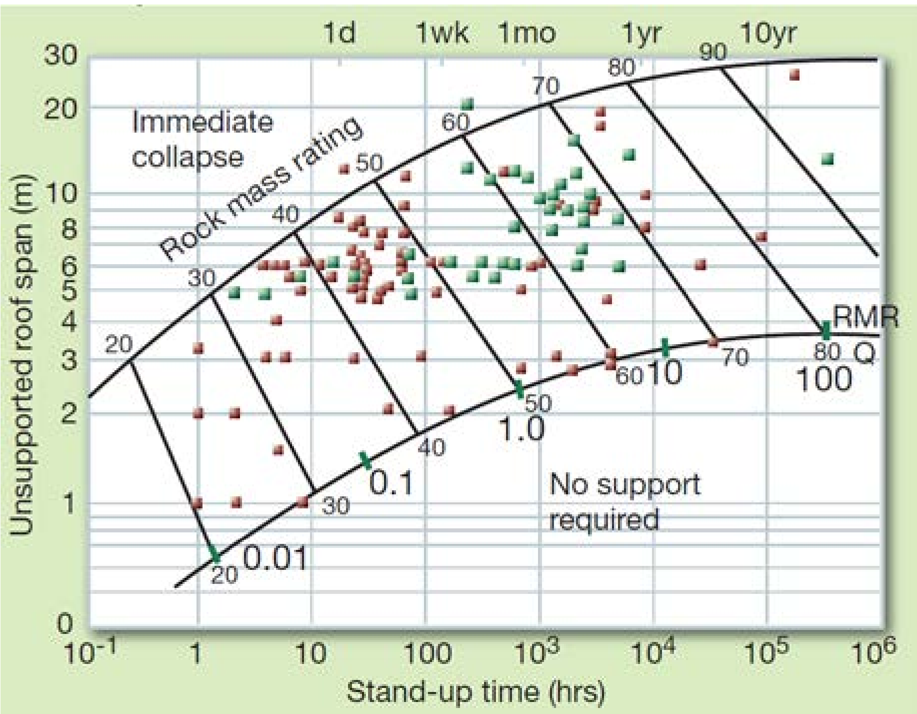

Figure 16-6: Stand up time data versus RMR from case histories (Bieniawski, 1989)

16.4.3 Ground Support

Ground support requirements were estimated using a combination of empirical and kinematic analyses. Separate ground support types were developed for what are considered long-term excavations including the main ramps, and short-term excavations including the mineralised zone, access drifts and production drifts.

The following are recommended ground support elements:

- Resined rock bolts;
- Swellex or Split Set (may be replaced with cable bolts of equivalent capacity);
- Mesh – 100 mm x 100 mm welded wire mesh; and
- Shotcrete with a minimum 8 hour strength of 10 MPa and a minimum 28 day strength of 30 MPa.

16.4.4 RQD/blasting characteristics

The rock mass strength and RQD would be used to establish the blasting characteristics of the mineralised material. Preliminary investigations suggest that rock drilling and blasting would be used as the principal production method. The fractured nature of the mineralised zone may make it amenable to alternative mining techniques such as bore hole mining where high pressure water is used to excavate soft or fractured ore.

16.4.5 Paste Backfill Design

The need for structural fill for mine ground control and to minimise resource sterilisation in pillars can be met with the use of paste backfill. Testing of various paste backfill compositions and process tailings will be required. A design target of 3.5 MPa, established by benchmarking worldwide, is assumed for backfill strength in the conceptual mine plan for the Berlin resource. Strength tests will be necessary and trial work in small drifts with equipment similar in weight to that proposed for the Berlin Project will be necessary. The average cement content for cut and fill backfill paste is 7% to 9% to ensure sufficient strength for mucking and load haul dump ("LHD")-bearing capacity.

16.5 Mineral Resource Parameters

The mineral resource on which the mine plan is based contains approximately 0.6 Mt in the Indicated category and 8.13 Mt of Inferred category, both of which have an average U_3O_8 grade of 0.11%. Hence, approximately 7% of the resource is in the Indicated category and 93% in Inferred. Although mineral resources have been estimated for multiple elements, the cut-off grade is based only on a uranium grade of 0.04% U_3O_8. As the project advances, the cut-off grade may evolve to a total mineral value cut-off based on the metal content of the mineralised material.

The use of uranium grade only for the cut-off grade is, however, practical from a mining point of view because grade control would be accomplished largely with a spectrometer in real time.

The mining model takes into account that approximately 20% of the resource will be left in structural pillars and hence would not be mined. Dilution estimates vary from approximately 10% to 20% for an average thickness of 3 m of mineralised material.

16.6 Conceptual Mine Design

The conceptual underground mine design was undertaken with Surpac software.

16.6.1 Exploration Decline

An exploration decline is proposed for several purposes including the collection of geotechnical data, testing backfill make-up and cement content, and for collecting a large tonnage of mineralised material for pilot plant studies in the feasibility stage of the project. The proposed exploration decline would be approximately 750 m long, 4.6 m wide and 5 m high inclined 15%, and located within the mineralised zone. The exact location of the exploration decline would be defined after additional infill drilling has been completed so that its position can be optimised with other planned mine development. There is a possibility that some of the infill drilling of the steep east limb of the resource area could be undertaken from the exploration ramp in order to cut mineralisation at a near-perpendicular angle to mineralisation and this would also save on drill metreage.

16.6.2 Mine Access

Principal underground access is envisaged to be by a 760 m long decline at -15% inclination from a portal located at an elevation of 805 m amsl (Figure 16-7). This decline would intersect a production ramp in the footwall of the deposit at an elevation of 690 m amsl. From this point of intersection with the principal decline, the production ramp would rise to the southeast at a grade of 10% to reach surface at an elevation of 744 m amsl. A second ramp would extend 2,100 m to the northwest to connect with the main pump station located at an elevation of 483 m amsl. This second ramp would provide access to mineral resources in the west limb of the Berlin syncline. Drifts would be driven from the ramp intersection at 690 m amsl to access mineralisation both above and below this elevation.

The proposed design of two mine developments, one to the north and the other to the south of the principal access ramp has two principal benefits:

- It decreases the hauling distance from the mine face to the underground crusher; and
- It allows the required 500,000 tpa to be mined from a single cut and fill sublevel of approximately 20 m height each year, thereby simplifying shift routines, and most importantly, contributing to safer working conditions.

The main access decline would be 4.6 m wide and 4.8 m wide.

16.6.3 Shafts and Raises

Two shafts would be required for the mine plan for ventilation and to provide secondary escape routes. One shaft would be located midway along the access decline and would have a total length of 180 m. The purpose of this shaft is primarily to provide an additional fresh air intake for ventilation during the development phase. The second shaft is an exhaust shaft located in the northeast of the resource area and has a total length of 250 m.

The planned diameter of the shafts is 3.7 m and construction is proposed to be with a mine-owned raise-boring machine. Each shaft would be equipped with an emergency escape hoist. The ventilation plan also requires raises to distribute fresh air through the mine workings. The estimated total length of ventilation raises for the mine plan is 2,400 m of 2.2 m to 2.5 m diameter raises.

Preliminary analysis suggests that the acquisition of a raise-boring machine is more cost effective than working with a contractor throughout the mine life. Having a raise-borer on site would allow all vertical or sub vertical openings to be cut efficiently at any time. A raise-borer would be used principally for shafts, ventilation raises, ore passes and emergency exits from ventilation openings.

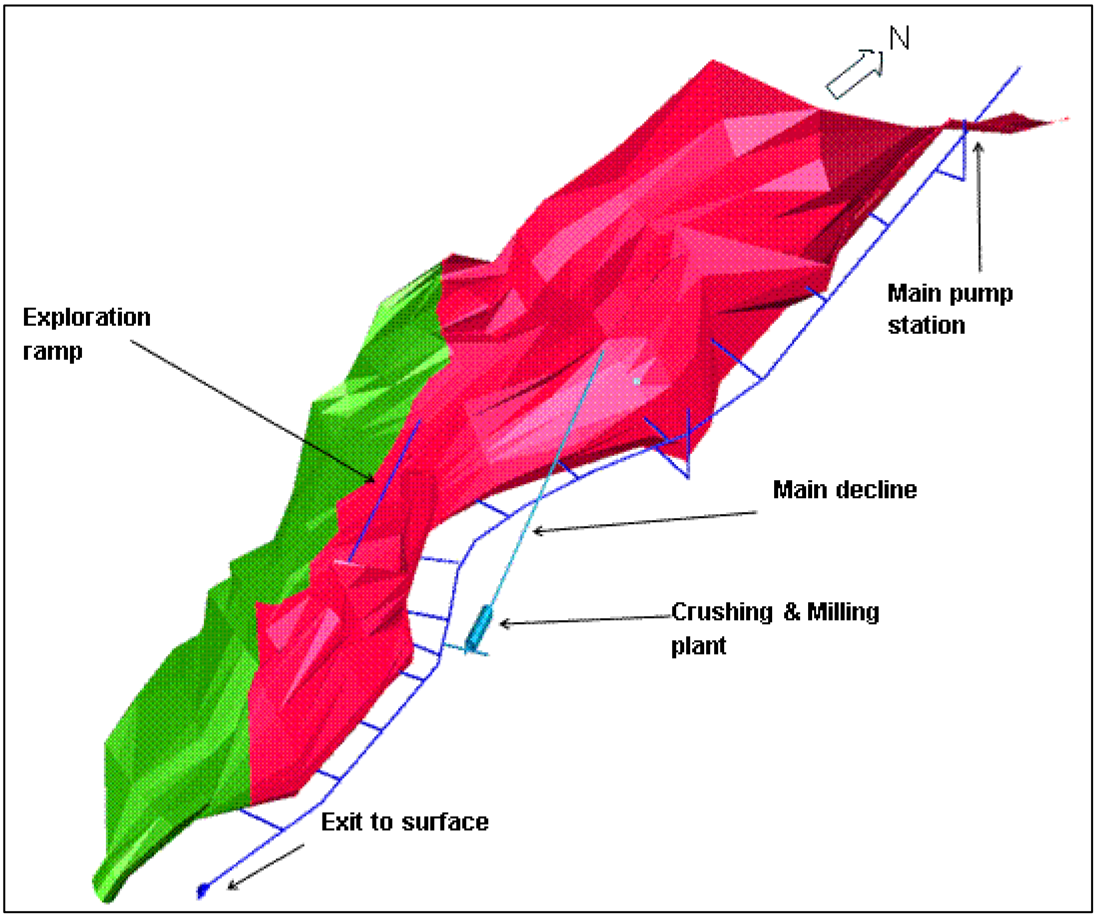

Figure 16-7: Oblique isometric view of the principal mine infrastructure relative to the mineralised layer at Berlin. (Pink is the carbonate-hosted mineralisation and green the oxidized, sandstone-hosted mineralisation

16.6.4 Underground Access and Development

Mine development for footwall drifts, access drifts and declines is planned to comprise excavations of 4.8 m high by 4.6 m wide. The inclination of development tunnels is planned to be set at 0-1% on levels and at grades of 10% to 12% in ramps, never exceeding 15% in any part of the design. All drifting is expected to be done conventionally using drill jumbos in conjunction with 28 t underground haul trucks and 6 m^3 LHD's. Utilities in these developments would include water, electricity, ventilation ducts, communication wiring and paste backfill pipes.

Development drifting from the decline would be accomplished by hydraulic drilling, blast, load and haul methods. Development drifts would typically be 4.5 m high by 4 m wide.

16.6.5 Mining

16.6.5.1 Cut and Fill Mining

The parts of the mineralised body that dip at angles greater than 45° are planned to be mined by cut and fill methods using cement-blended paste backfill for structural support. Approximately 70% of the resource area, including most of the east limb of the fold, would be mined using cut and fill. The relatively small parts of the west limb that are steeply inclined would also be mined with this method.

Mine design was based on a mineralised zone with an average thickness of 3 m with a minimum of 2 m and maximum of 5 m including stope overbreak as dilution.

Due to the weak nature of the mineralised zone and overlying shale, a relatively small sublevel height of 20 m is planned. Stability analysis indicated that a 30 m stope length is appropriate, but a mining plan would require the flexibility to change the stope length to as little at 10 m in areas of poor stability and to 40 m in areas of good rock quality.

Each cut and fill access point would be located 30 - 40 m from the mineralised zone to allow successive access ramps to be developed up and down to a maximum 15% grade. The lowest, and first cut would be accessed from a ramp grading -15% from the main access cross-cut. The second cut would require "back slash and fill" development to construct a ramp at approximately -10%. The third cut will be level with the access drift and the final two cuts would be accessed by ramps grading +10% and +15%, respectively.

Each 20 m sublevel would be mined in five lifts of 4 m (two down, one at the same elevation, and two up). In each stope, the lower lift would be mined first followed by successively higher lifts. In practice, the mining sequence may result in several consecutive stopes along strike being opened at the same time. This is expected to be acceptable in most cases, however in several small areas at Berlin where the mineralised zone is at its widest, the mining - backfilling sequence may need to be adjusted so that no more than two stope backs are open at a time to ensure stability of the back.

16.6.5.2 Room and Pillar Mining

Approximately 30% of the mineralised feed is planned to come from shallowly inclined (less than 45° dip) parts of the west limb which is proposed to be mined by room and pillar methods. The hanging wall in the west limb is provisionally classified as weak and therefore pillar reinforcement may be required in exceptional circumstances where the mineralised zone is greater than 5 m thick, requiring openings over 5 m high.

The excavation of each room and pillar panel would be done with conventional drill-and-blast New Australian Tunnelling Method ("NATM") techniques including the application of shotcrete and rock bolts. Panel width would vary between 2 m and 4 m depending on rock strength. On completion of mining of a panel, the cavity would be filled with cement-blended paste-backfill and mining would step to the other side of the support pillar where the same procedure would be followed. After the backfill had set, the mineralised material that had constituted the pillar between the two panels would be mined out, supported by the backfill on either side. The dimensions of each pillar would typically by 5 m by 5 m. Pillars would typically be placed on 20 m to 40 m centres, depending on roof conditions. The same low-profile LHD and tunnel drilling equipment would be used for room and pillar development and production.

16.6.6 Paste Backfill Plant

A paste backfill plant is designed to be placed near the access portal with the majority of the feed being comprised of tailings output from the processing plant. Crushed waste rock would also be fed to the backfill plant from the crusher-mill chamber. This placement would minimise the material handling cost. It is envisaged that the plant would work with piston pumps with the capacity to dispatch 100 m^3 per hour of paste though 20 cm to 30 cm diameter pipes to the mine stopes. The main system components providing paste backfill to the pump are:

- Two 109 t hoppers that feed the horizontal mixer;
- Horizontal mixer that combines the tailings, water and cement;
- Two 40,000 litre water tanks to stage the water before the mixer; and
- Two 120 m^3 capacity bins to stage the tailings before mixing.

The paste backfill requirement will be in the order of 1,000 m^3 per day.

16.6.7 Underground Crushing and Grinding

An underground crushing and milling complex is envisaged to be located in an underground silo located in granite some 80 m to 100 m below surface adjacent to the main decline. The complex would consist of a bin for mineralised material excavated into the granite, feeding into a primary crusher with a 100 tonnes per hour ("tph") capacity followed by a secondary cone crusher feeding into a 1,500 tpd capacity SAG mill. Components of the SAG mill would be transported down the main decline for final assembly underground. The grinding section could also be at the plant because the grinding process is a wet process and large space is required for the storage heaps of crushed ore before the grinding section. Further details of the crushing and milling complex are provided in Section 17.

16.6.8 Mine Power Requirements

Mine operations would require a total of 6.17 MW of installed electrical capacity. This is accounted for in three principal areas including:

- 0.76 MW for mine equipment, principally for the diesel-electric drilling equipment and the and the raise-boring machine;
- 2.72 MW for compressors, ventilation and pumps. Ventilation and pumps would operate 24 hours a day for 365 days in the year; and
- 2.69 MW for mine dewatering. This consumption estimate is conservative and is influenced by the mining consultant's experience at Serra Pelada, an underground mine that is being developed in similar rocks in similar climatic conditions in Brazil.

16.6.9 Ventilation

The basis for the design of the ventilation system for the conceptual mine was the assumption that the volume of air required to remove exhaust gases produced by underground diesel equipment is adequate for the dilution of radon gas to acceptable levels as well as to provide for its extraction from the underground environment. Equipment utilisation factors were used to estimate the number of diesel machines in use at any one time and the associated volume of diesel fumes generated. With additional input parameters including resistance, k-factor (friction factor), distance, length, area, perimeter and whether air flow is fixed or variable, Ventsim Mine Ventilation Software was used to generate an air flow model for the maintenance of required air quality (Figure 16-8). The ventilation designs ensure that airflow is such that workers are upwind of potential sources of elevated levels of radon gas and radioactive dust.

The ventilation system for the Berlin Project anticipates a total air flow of approximately 40,000 m^3 per minute during full production. The principal fresh air intake would be through the access decline and air would be extracted through the exhaust shaft which would contain two 250 horsepower ("HP") exhaust fans. A system of ventilation raises would be located along the decline, providing a connection with the stoping zone in the deposit's footwall. These connections, in conjunction with air-control doors, regulators and auxiliary fans, would provide the required airflow through the mine.

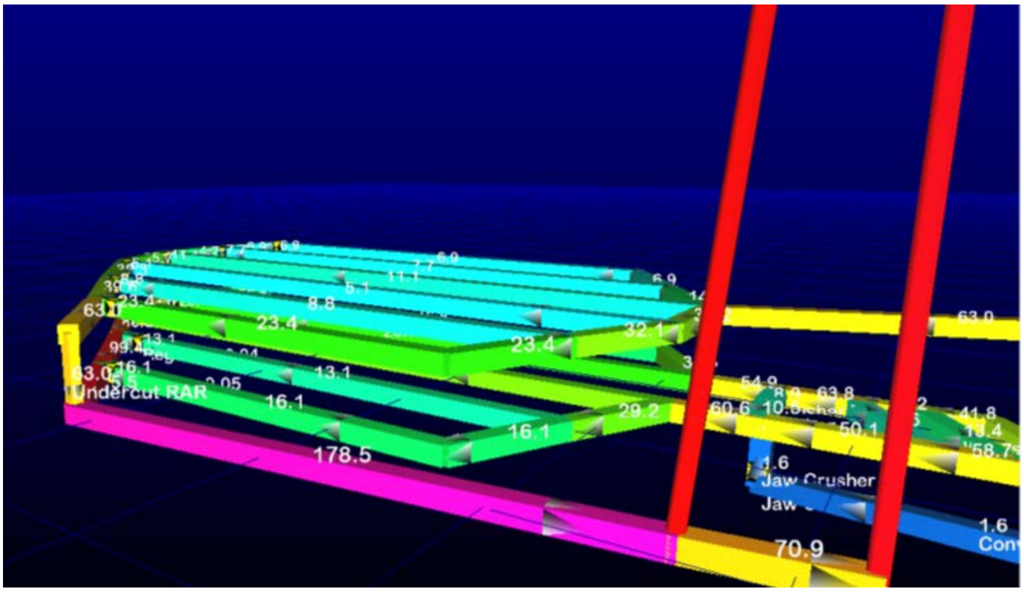

Figure 16-8: Schematic of the ventilation circuit and air quality analysis for proposed underground development on the west limb of the Berlin syncline. Figures refer to the concentration of air pollutants and colours to air temperature (Yellow is inlet air temperature of <22°C, light blue: 22°-24°, green 24°-28°, pink: >28°C)

16.6.10 Dewatering

Dewatering of an underground operation at Berlin would commence two years before production start-up. The dewatering program would be based on 12 large (300 HP) pumps installed on 30 cm diameter bore holes located strategically within the mine workings and in the sump at the bottom of the northwest decline. This dewatering program is designed to cope with large volumes of underground water similar to those encountered in the underground development of the Serra Pelada deposit in Brazil.

Water from the mine would be pumped to settling tanks at surface for use in the plant. Water quality would be monitored and tested for radioactivity before being discharged into the environment.

16.6.11 Mine Services

Design consideration was given to mine services such as underground explosives storage, fuel storage and distribution, compressed air supply, water supply, underground transport of personnel and materials and underground maintenance and wash bays.

16.6.12 Tailings

Two alternative processing options are under consideration, and the one selected would have a material impact on the way that tailings are managed:

- One processes ROM mineralisation without beneficiation. In this case, tailings would be used in backfill, and the excess would be gravitated to a tailings facility more fully described in Section 18.2.4.
- The second option uses acetic acid to remove the calcite prior to the metal extraction process. Acetic acid dissolves 50%-60% of the mineralised material and only the remaining mass undergoes further processing for the extraction of metals. Hence, tailings volumes are much lower, to the extent that they would all be used in backfill, so surface disposal of tailings is avoided.

16.7 Safety

The principal safety structure of the mine would consist of evacuation through the main access and production declines as well as emergency escape hoists in the ventilation shafts. Additional mine safety design criteria include fire suppression, mine rescue and refuge stations. The use of personal radiation monitoring devices would be mandatory for all personnel.

16.8 Mine Equipment

Underground mining at Berlin is designed to be highly mechanised and completely trackless. All mobile equipment is planned to be purchased new. The equipment is planned to be overhauled on reaching 60% of life expectancy, measured in operating hours. The overhaul of equipment is expected to cost approximately 50% of the replacement value of new equipment. On reaching the end of the extended life, the equipment would be sold as scrap and replaced. The only exception envisaged is the raise-boring machine that is expected to retain reasonable value at the end of the mine-life.

Equipment hours were estimated on the basis of a first principles cost model. Expected hours of operation were calculated for each item of equipment on the basis of the mine development and operation schedule. The mobile equipment list is shown in Table 16-1.

Table 16-1: List of major mobile equipment items required for the conceptual mine plan

Description	Quantity
Mine truck (28t) 300HP	5
LHD 3m3 (Development) 139HP	4
LHD 5m3 (Production) 231HP	4
Jumbo Twin Boom (88-104kW)	3
Jumbo Single Boom (24.5kW)	3
Roof Bolter 57HP	3
Shotcrete truck 210HP	3
Scissor lift truck 82HP	3
Service transport truck	3
Jackleg and stopper drill	10
Long-hole drill 40HP	4
Raise Boring Mach 2.5-3.7m diam. 350HP	1
Surface FEL - Cat 966 type 180HP	2
Grader - Udg. 138HP	1
Pick-up 150HP	4
Total	**53**

16.8.1 Development Equipment

All mine development and production would be done with mine-owned and operated equipment, with the exception of the proposed exploration ramp which would be excavated by a contractor. The composition of the decline fleet was determined by matching the planned advance rate of 4 m per day per heading with the mine plan - required drifting lengths and headings on an annual basis. Another factor taken into account is the labour law stipulation of a maximum of 8 hour shifts for the underground crew. The fleet defined for the construction of the declines includes single-boom jumbos, a roof bolter, LHDs with a 3 m^3 capacity and 28 t capacity trucks.

After the access decline is complete and multiple mine development headings are available, two other development equipment fleets would be required to advance multiple development ends. Each fleet would include single-boom drill jumbos, a roof bolter, 3 m^3 capacity LHDs and 28 t capacity trucks. A raise-boring machine with a capacity to operate from surface for the construction of shafts, as well as underground for the development of raises and ore passes, is required.

16.8.2 Production Equipment

In-stope drilling would be undertaken with long-hole drill machines mounted on twin-boom jumbos. Mucking of mineralised material from the stopes would be with 5 m^3 capacity LHDs that would load 28 t haul trucks which would transport the material to the underground crushing facility. Auxiliary equipment for the production fleet would include small LHDs to be used for cleaning up and preparing the stopes, shotcrete trucks, scissor lift trucks and an underground motor grader.

16.8.3 Support Equipment

Support equipment includes major equipment that is required to install mine services, to transport personnel, water pumps, compressors and equipment to provide temporary ventilation. Service trucks, fuel and lube trucks, along with skid steers and forklifts, are included as service equipment.

16.9 Personnel

The average number of employees envisaged over the life of mine is approximately 180 as listed in (Table 16-2).

Table 16-2: Summary of personnel requirements for production in the mining scenario modelled for the Berlin resource area

Description	Quantity
Mine Superintendent	1
Mine Maintenance Superintendent	1
Production Foreman	1
Mine Shift Foreman	4
Drill Foreman	4
Blasting Foreman	4
Surveyor	3
Surveyor Helper	3
Jumbo Operator	6
Jumbo Offsider	6
Longhole Operator	8
LHD Operator/FEL in surface	16
Raise Borer Operator	3
Helpers in General/Miners	15
Backfill Crew	6
Bolter	9
Grader	3
Diamond Driller	3
Utility Vehicle Operator	6
Mechanician	9
Eletrician	6
Welder	3
Tireman	3
Udg. Truck Operator	12
Maintenance Labourer	4
Lube Truck Operator	4
Piping helper	6
Pumping Operator	6
Ventilation Technician	3
Safety Technician	6
Labourer	16
Total	**180**

16.10 Mine Development Schedule

16.10.1 Exploration Ramp

Development of the proposed exploration ramp would ideally be undertaken during the prefeasibility or feasibility stage of the project.

16.10.2 Pre-Production Development

Total pre-production development is estimated to take 24 months (Year -2 and Year -1 of the proposed development schedule).

This phase would include decline development, excavation of the underground crushing and milling chamber, ventilation system development, installation of the pump station and pre-development in readiness for mining of the mineralised body. During this time, the backfill plant and the processing plant would be installed.

Drift driving and chamber development would be done using a drill, blast, load and haul cycle. The mining fleet, personnel list and support services were designed to achieve an advance rate of 4 m per day.

The waste from pre-production development would be hauled to the surface where it would be crushed and screened for future use as filling material. A summary of horizontal and vertical development metreage is shown in Table 16-3.

Table 16-3: Summary of pre-production development parameters

Horizontal/ Ramp Development	Unit	Quantity
Ramp (4.6 x 5 m)	m.l	760
Tailing Storage Udg. (4.6 x 5 m) 11,500 m^3	m.l	500
Drift Development – Waste (m)	m.l	500
Total	**m.l**	**1,760**
Vertical Development/ Raise Boring Works		
Exit Shaft (3.6 m diam)	m.l	250
Exhaust Ventilation Shaft (3.6 m diam)	m.l	250
Ore Storage (3.6 m diam)	m.l	200
Mine Ventilation Raises (2.3 m diam)	m.l	800
Total	**m.l**	**1,500**
Secondary Mine Openings		
Trucks Bypass (m^3)	m^3	10,000
Side Excavations (m^3)	m^3	10,000
Underground Electrical Room 1 (m^3)	m^3	1,750
Udderground Electrical Room 2 (m^3)	m^3	1,750
Explosives Magazine (m^3)	m^3	400
Total	**m^3**	**23,900**

16.10.3 Production Development

The processing plant has been optimised at a nominal 500,000 t annual throughput. Production is envisaged to ramp up through 250,000 t of mineralised material in Year 1 to full 500,000 t production in Year 2 of the proposed mine development schedule. Production would be maintained at this level through Year 14, and in Year 15, the remaining mineralised material of 210,000 t would be mined and processed. At the full production rate, daily output from the mine would be 1,430 t of mineralised material and approximately 715 t of waste, totalling 2,145 tpd with 350 working days per year.

After the first year of stope production, drift development would have advanced far enough to allow for the operation of only one development fleet. Raise-boring for ventilation purposes would be required throughout the mine life. All waste rock from development would be hauled to the surface where it would be crushed and screened and pumped underground again for use as back-filling material.

The final balance of envisaged production of mineralised material and waste indicates that 6.96 Mt of mineralised material and 3.49 Mt waste would have been excavated from both horizontal and vertical developments, yielding an average rate of 0.50 t of waste per tonne of mineralised material.

A brief verification of the loading and transport capacity, which forms the basis of the production fleet design, is presented in Table 16-4. Assumptions are that mine operation would be based on two 8 hour shifts per day for production and one 8 hour shift for maintenance and auxiliary services. The loading calculation assumed a bulk density of 1.25 t/m^3. The conclusion is that a design factor of the order of 41% is applied to the main development and production fleet.

Table 16-4: Verification of loading and transport capacity

Equipment	Qty	Size m^3	Shifts per day	Avail %	Util. %	Effect. h/day	Cycles Per hour	Daily Cap m^3/ day	Daily Cap t/d	Required t mineralised material/ day	Required t waste/ day
LHD 5 m^3	4	5	2	70%	60%	7	12	1613	2016	1430	
LHD 3 m^3	4	3	2	70%	50%	6	12	806	1008		715
LHD 30 t	5	18	2	70%	40%	4	6	2419	3024	1430	715

Description	Designed Capacity	Required Capacity	Designed %
Production LHDs	2016	1430	141%
Production Trucks	3024	2145	141%
Development LHDs	1008	715	141%

The results of the trade-off studies will be used in the technical and cost analysis in the pre-feasibility stage of the project.

16.11 Summary

The project was designed to achieve a nominal production rate of 1,430 tpd (500,000 tpa) in Year 2 and sustain that rate for 14 years based on the mining of approximately 80% of the current resource.

The basic structure of the proposed mine would be a main access ramp, a production ramp and two ventilation shafts. The location of a proposed exploration decline would be optimised to form a useful part of the mine infrastructure.

Due to the shape of the mineralised layer, two mining methods would be employed: cut and fill in the steep-dipping east limb of the Berlin syncline and mainly room and pillar in the shallow-dipping west limb. Both mining techniques use trackless, fully mechanised mining techniques.

An underground crushing and milling facility (milling also considered to be at main plant), described in Section 17.3.2, is envisaged as a means of mitigating contamination of the environment with radioactive dust.

Depending on which processing method is used, all of the tailings generated from the plant could be pumped back underground as cement-blended paste backfill. If an alternative processing route is used, then surface tailings facilities would be needed to accommodate the excess material not required for underground stability. In the latter case, tailings would gravitate downhill in a slurry pipeline to a tailings facility located on a stable metamorphic and igneous rock base as further described in Section 18.2.4.

Geotechnical studies are in their infancy in the Berlin Project and a large amount of additional geotechnical information is required to further develop mining concepts as the project advances beyond this preliminary economic assessment. Preliminary geotechnical information suggests that NATM drilling and blasting tunnelling techniques are appropriate for development and mining at Berlin.

The possibility of using bore hole mining techniques for the shallower, softer parts of the mineralised unit requires investigation.

16.12 Further Work

- Undertake extensive geotechnical test work, some of which should be on bore holes designed specifically for this purpose using orientated core. This test work should include uniaxial and triaxial compressive strength tests on core samples from a representative number of sites throughout the resource area.
- Strength test work is required on cement-blended paste backfill material;
- Pump tests should be performed on various rock intervals to define the hydrogeological characteristics of the resource area;
- An exploration decline should be excavated by a contractor. The main purposes of this proposed ramp are as follows:
  - Collection of geotechnical data;
  - Testing of mining methods and stope conditions;

- Testing of backfill composition and strength; and
- To provide a bulk sample of mineralised material for a pilot plant in the feasibility stage of the project.

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17 RECOVERY METHODS

17.1 Process Selection

The process plant is designed to recover and extract uranium (U), vanadium (V), molybdenum (Mo), nickel (Ni), zinc (Zn) and phosphorous (P) and some rare earth elements (REE), including yttrium (Y) and neodymium (Nd), and concentrate these elements in the different compounds described in this report. U is extracted mainly from uraninite (pitchblende) minerals contained in the mineralised material from the Berlin Project.

Two process options were considered for U3O8 Corp's Berlin Project PEA study. Both require further assessment through ongoing metallurgical testing and trade-off studies to be undertaken as the project advances. A selection between these and additional processes, such as flotation, would be made in the next stage of development of the project. The two processes considered in this report are:

- Option A: Mineralised material undergoes a pre-leach step using acetic acid as the leachant to react with calcite and produce gypsum as a potential by-product before the acidic ferric iron leach step described in Section 13.5.4; and
- Option B: The same leaching process is used without the acetic acid pre-leach step. The relatively small amount of gypsum formed in the option B process exits via the tailings and is not recovered.

The processes are divided into the following main discrete areas:

- Comminution;
- Leaching; and
- Metal extraction and recovery, and uranium product calcining.

A flowchart that contains all of the major processing stages (Figure 17-1) illustrates the overall process flow for Option A.

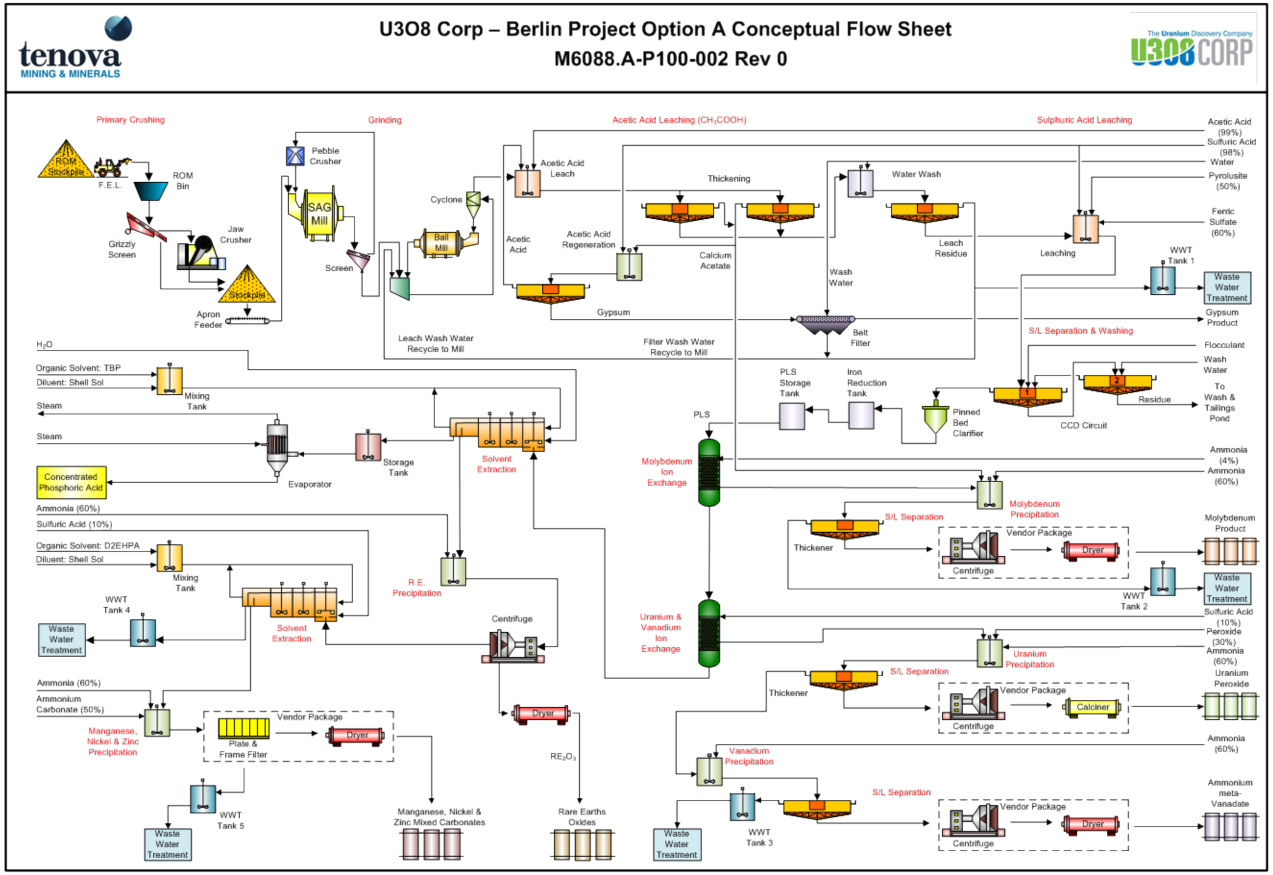

Figure 17-1: Flowchart of overall process flow for Option A

The plants for Options A and B were designed for a feed rate of 122 tph and an annual utilisation of 8,200 hours per year. This corresponds to an annual treatment rate of 1 Mtpa of dry material. At the request of U3O8 Corp, alternative cases were developed for both options to process 0.5 Mtpa. This corresponds to a plant throughput of 61 tph of dry material. The final cost estimates for both options in the PEA study are based on the throughput of 61 tph of dry material.

Table 17-1 summarises the process design documents produced. The process design documents for Options A and B are all based on an annual treatment rate of 1 Mtpa. Capital costs however, were estimated for the annual treatment rate of 0.5 Mtpa of dry material. Table 17-2 provides the annual production rate of the final compounds produced.

Table 17-1: Design document list

Document Number	Document Name	Appendix Number
M6088.A-M810-001	Option A Process Plant Mechanical Equipment List	Appendix C
M6088.A-M810-002	Option B Process Plant Mechanical Equipment List	Appendix D
M6088.A-P100-002	Option A Process Conceptual Flow Sheet	Appendix E
M6088.A-P100-003	Option B Process Conceptual Flow Sheet	Appendix F
M6088.A-P150-001	Option A Processing Plant Mass Balance	Appendix G
M6088.A-P150-002	Option A Processing Plant Mass Balance Calculations	Appendix H
M6088.A-P150-003	Option B Processing Plant Mass Balance	Appendix I
M6088.A-P670-001	Option A Process Design Criteria	Appendix J
M6088.A-P670-003	Option B Process Design Criteria	Appendix K

Table 17-2: Final products Option A at 0.5 Mtpa throughput

Compounds	Yearly production (t)	Alternative compounds for sale	Yearly production (t)	Net Recovery %
U_3O_8	532	U_3O_8	532	96.1
NH_4VO_3	2,137	V_2O_5	1,662	66.0
H_3PO_4	63,485	P_2O_5	45,988	98.9
$Y(OH)_3$	257	Y_2O_3	207	83.5
$NiCO_3$	1,482	Ni	733	63.7
$Nd(OH)_3$	33	Nd_2O_3	28	48.6
$CaMoO_4$	284	Mo	136	45.4
$ZnCO_3$	2,943	Zn	1,534	95.9
Gypsum	38,8270	Gypsum	388,270	
		$MnCO_3$	1,807	

No test work results are available to substantiate the product recovery efficiencies in the solids liquids separation system after acetic pre-leach and acidic ferric iron leach, nor in the metals recovery section. Such test work is typically done at more advanced stages of the project, such as during PFS, since a bulk sample of several tonnes of material is usually required to generate sufficient PLS for representative extraction test work to be done. Thorough test work is recommended to determine more accurately these net recovery efficiencies.

Estimation of the net recovery efficiencies has been based on the acidic ferric iron leach recovery efficiencies and on the assumption that most of the uranium solubilised in the acetic acid pre-leach would be recovered in the downstream thickening and washing circuit. Estimates of losses of the different elements for the processes downstream of the acidic ferric iron leach have been based on experience with similar processes used on other projects.

17.2 Production Basis

The production parameters used as the basis for the Capital Cost Estimate ("CAPEX") and Operating Cost Estimate ("OPEX") are:

- Annual production – 0.5 Mtpa of dry feed;
- Mill feed grade – 1,179 ppm of U_3O_8 in ROM mineralised material;
- Net metal recovery of 96.1% of U (Table 17-2); and
- Mill design throughput: – 61 tph dry ROM.

Table 17-3 illustrates the metal extraction and PLS composition after the acidic ferric iron leaching process for Options A and B. The downstream recovery efficiencies assumed resulted in the net metal recoveries as illustrated in Table 17-3. The extraction percentages assumed are the same for both options.

Table 17-3: Options A and B: metal extraction and PLS composition after acidic ferric iron leaching

OPTION A – PLS Assay after Acidic Ferric Iron Leach					OPTION B – PLS calculated after Acidic Ferric Iron Leach		
Element	Element in PLS (ppm)	Element Mass Flowrate in PLS (kg/h)	Element Mass Flowrate in PLS (tph)	% Leach Extract	Element Mass Flowrate in PLS (tph)	Element Mass Flowrate in PLS (kg/h)	Element in PLS (ppm)
U	595	115,668	0.1157	98	0.1195	119,531	464
V	1,391	270,410	0.2704	73	0.2034	203,389	790
Mo	183	35,575	0.0356	51	0.0330	33,038	128
P	20,291	3,944,590	3.9446	100	3.9446	3,944,590	15,328
Ni	857	166,601	0.1666	60	0.1819	181,902	707
Zn	1,807	351,281	0.3513	98	0.4011	401,069	1,559
Mn	146	28,305	0.0283		0.0276	27,590	107
Y	195	37,908	0.0379	91	0.0403	40,301	157
Nd	44	8,554	0.0086	64	0.0062	6,226	24

17.3 Process Description

17.3.1 General Overview

The mineralised material is crushed and ground to liberate the uraninite and other minerals from the gangue for efficient extraction of U and other elements during leaching. The plant is designed to produce a milled product with a P_{100} of 106 µm and P_{80} of 75 µm.

In the case of Option A, the milled solids would be pre-leached in acetic acid to obtain an overall mass loss of 50% to 60%. The pre-leach removes effectively all of the calcite and a negligible amount of other metals from the mineralised feed. The pre-leached solids residue is then washed in a counter current decantation ("CCD") circuit to recover calcium acetate which is converted to gypsum and acetic acid. The recovered acetic acid is recycled for reuse.

In the acidic ferric iron leach, ferric ions (Fe^{3+}) are used to oxidise the U in the mineralised material. The mineral slurry is contacted with concentrated sulphuric acid at a temperature of 65°C. The remaining calcite reacts to form gypsum while apatite is also leached to solubilise the valuable metals into the PLS in an oxidised form, while liberating phosphate. Manganese is added in the form of pyrolusite to oxidise Fe^{2+} (ferrous ions) to Fe^{3+} (ferric ions).

The solids are washed in a CCD circuit and the overflow is clarified in a pinned bed clarifier ("PBC") to form a clear PLS. The Fe^{3+} ions in the PLS are reduced to Fe^{2+} by adding iron metal. This is to prevent Fe^{3+} from interfering in the downstream IX and SX processes.

In the following process, Mo is selectively extracted from the PLS by an IX process, then eluted, precipitated and dried to form a Mo salt ($CaMoO_4$).

Thereafter, U and V are co-extracted by an IX process and then eluted. A compound of U is first precipitated from the solution and then calcined to form U_3O_8 or UO_3, collectively known as yellowcake. Vanadium is then precipitated from the resultant solution and dried as ammonium meta-vanadate (NH_4VO_3).

The PLS, now depleted of Mo, V and U, is fed to the phosphoric acid SX process for concentration. Extraction is carried out with the organic liquid tributyl phosphate ("TBP") and stripped with water. The extracted phosphoric acid is still relatively dilute in the PLS and requires further concentration to about 83% H_3PO_4 (or 62% P_2O_5), the preferred concentration for the saleable product. The phosphoric acid is concentrated by evaporation of water from the PLS in an indirect heated evaporator.

The raffinate from the phosphoric acid process proceeds to the REE precipitation section. REE metals including yttrium and neodymium are recovered by precipitation by adding ammonia to the raffinate. The REEs precipitate as hydroxides ($Y(OH)_3$ and $Nd(OH)_3$).

The Ni, Zn and Mn are recovered by SX in the next process step. These metals are co-extracted with di-(2-ethylhexyl) phosphoric acid ("D2EHPA") extractant. They are then stripped from the extractant with sulphuric acid as a concentrated blend. The blend is co-precipitated from the strip solution with ammonium carbonate and is then filtered and dried to produce a blend of Ni, Mn and Zn mixed carbonates.

17.3.2 Comminution

17.3.2.1 Background

The role of comminution is to prepare the mineralised material for leaching while taking into account the following criteria:

- The mineralised material must be ground sufficiently fine so that the calcite solids can be leached in acetic acid which is a weak acid;
- The mineralised material must be ground sufficiently so that the U and other elements are liberated to be readily available for leaching; and
- Over-grinding should be avoided as it adds costs and results in slurry flow and solid/liquid separation issues.

Variability is expected in the breakage characteristics of calcite and apatite in the mineralised material. Detailed investigation of variability in the grinding characteristics of the mineralised material, and the effect thereof on comminution and thus leaching, should be done in the next stages of the project's development.

17.3.2.2 Primary Crushing

The primary crushing circuit is fed from the ROM mineralised material. ROM material is transported to the ROM pad by mine trucks. The annual throughput of the crusher would be 0.5 Mtpa (dry).

The proposed primary crushing circuit includes:

- Underground jaw crusher;
- ROM pad;
- Static grizzly;
- ROM bin; and
- Vibrating grizzly.

A 10 m high pillar separates the ROM pad and a conveyor that transfers the mineralised material at the bottom of the feeder to the primary crusher. Oversize material from the grizzly is stockpiled for rock breaking. Mineralised material from the primary crusher is discharged by a variable speed apron feeder to the grinding circuit.

The basic process design was done for the 1 Mtpa throughput, but the costing for Options A and B were based on a 0.5 Mtpa throughput.

The crushing station would be located in an underground chamber located 80 m to 100 m below surface along the main decline.

Figure 17-2 illustrates an example of a crushing station in an underground chamber at Ridgeway Gold Mine, located approximately 25 km northeast of Columbia, South Carolina. It is owned by Kennecott Minerals Company (Jaakonmaki, 2011).

Figure 17-2: Example of a crushing station in an underground chamber

The maximum size of the equipment to be used in the underground installation will require careful design in later phases of the project development (Figure 17-3). The access tunnel must be sized to allow the transport of the large pieces equipment to the chamber. It is envisaged that the larger components of the equipment would be assembled in the underground chamber.

The size distribution of the ROM feed to the crusher is as follows:

- F_{100} material to be smaller than 500 mm;
- F_{95} material to be smaller than 300 mm;
- F_{80} material to be smaller than 110 mm; and
- F_{50} material to be larger than 50 mm.

The crusher feed would be sorted by a static grizzly followed by a vibrating grizzly and rock breakers would be used to break oversize rocks. The product from the mill would be classified by two screens and would have a P_{100} of 200 mm and a P_{80} of 35 mm.

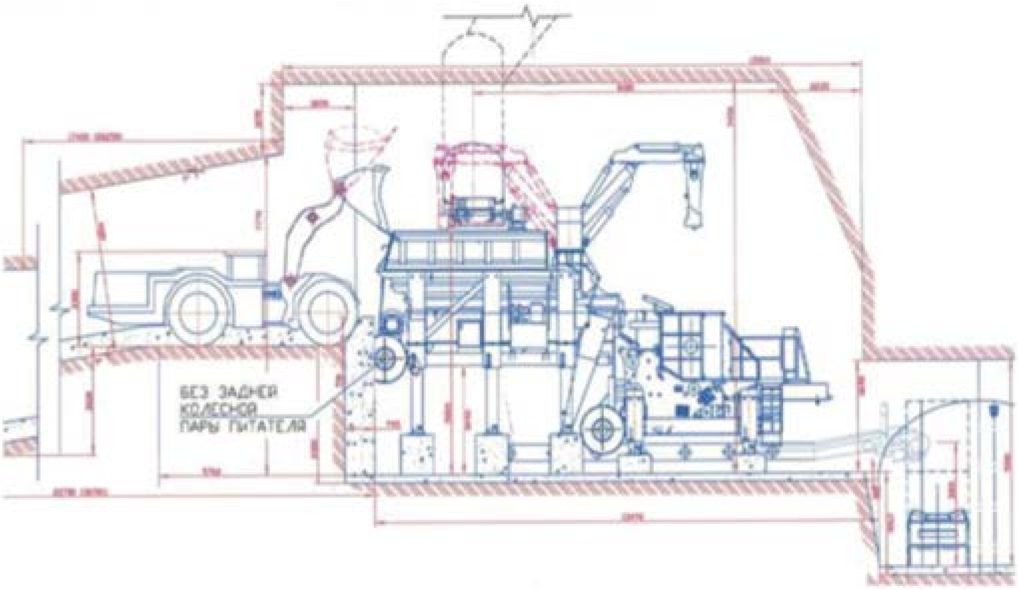

Figure 17-3: Underground crushing station arrangement (Jaakonmaki, 2011)

Bond Work Index (BW_i) tests were undertaken on seven representative samples of the various types of mineralised material at SGS Lima in Peru and the results are shown in Table 17-4. An average mineralised material BW_i of 12.0 kWh/t was used for the crushing simulation and equipment specification.

Table 17-4: Bond Work Index

Reference	BWi (kwh/t)
Fresh mineralised limestone 1	10.95
Fresh mineralised limestone 2	11.61
Mineralised limestone	14.03
Mineralised limestone (moderately fractured)	11.28
Mineralised limestone (highly fractured)	10.44
Hard Sandstone (bottom mineralisation)	12.83
Leached mineralised limestone (near surface)	6.59

The ROM bin would be excavated in a granitic stock that is composed of structurally sound, competent rock. In practice, it is possible to excavate any type of chamber in competent rock with the shape and size required to accommodate most common crushing equipment underground at acceptable expenditure.

The principal benefits of underground crushing are that transport distances are shorter and fine-grained mineralised material is not subjected to the elements, such as wind and rain, which may interfere with the performance of the operation. In the case of the Berlin Project, the underground installation of equipment would minimise the use of flat areas in the mountainous terrain where the deposit is located. It would also reduce the risk of contamination of the environment by dust and runoff from the crusher-mill complex. Ventilation is required to maintain clean air in the crushing area for workers and equipment. This part of the operation would be separate from the main ventilation circuit of the mine.

A detailed study is recommended to address the underground layout and space requirements of the crusher and mill complex in the next phase of the project.

17.3.2.3 Primary Grinding: SAG

Primary grinding would be achieved by SAG. The SAG feed weightometer would control the flow rate to the SAG mill by means of an apron feeder. The SAG mill parameters are described in Table 17-5.

Table 17-5: SAG mill design criteria

SAG Mill Design Criteria	Unit	Value
Feed Size - F_{100}	mm	200
Feed Size – F_{80}	mm	110
Grinding Work Index used	kWh / t	12.0
Mill Product Size – P_{100}	mm	5
Mill Product Size – P_{80}	mm	1.5

The feed rate set point for the SAG mill would be controlled by a combination of mill power, mill load and possibly mineralised material grade and/or other variables by using an algorithm to be defined at a later date. The feed conveyor system to the SAG mill would be equipped with magnets to remove tramp iron. The process recycle water used for the wet screening is the recycled solution from the different thickeners and precipitation processes. A possible apron feeder system is shown in Figure 17-4 (Jaakonmaki, 2011)

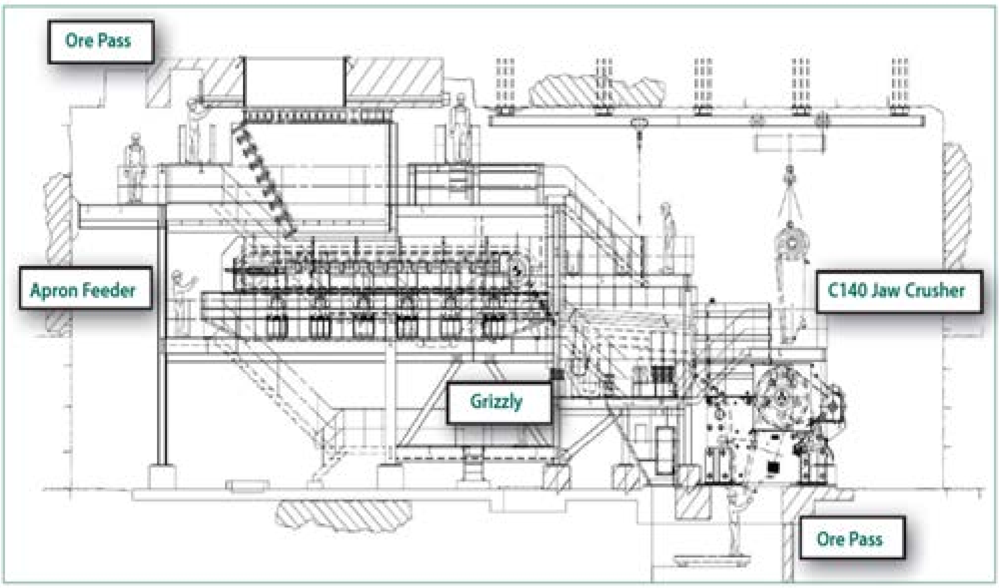

Figure 17-4: Possible apron feeder arrangement

The grinding process comprises a single variable speed SAG mill, 5.5 m in diameter and 6.7 m in effective grinding length ("EGL"), operating with an 18% ball charge. In the next phase of the study, the viability of autogenous grinding ("AG") vs. SAG should be investigated. Should AG be viable, the ball milling operation could probably be removed from the current process design flow sheet. More grinding test work and further design will be required to refine these parameters.

17.3.2.4 Secondary Grinding – Ball Mill

A ball mill would operate in a closed circuit with classifying cyclones to produce a P_{100} size of 106 µm and a P_{80} of 75 µm with a 200% re-circulating load. Table 17-6 shows the design parameters.

Table 17-6: Ball mill design criteria

Ball Mill Design Criteria	Unit	Value/Description
Equipment Type	-	Ball Mill
Throughput	Mtpa (dry)	0.5
Recirculating Load	%	200.0
Feed Slurry Solids Density	%w/w	50.0
Slurry Dilution Source 1	-	Process Water
Slurry Dilution Source 2	-	Leach Wash Water
Ball Mill Feed Storage	-	Sump
Ore Delivery Method	-	Slurry Pump
Feed Size - F_{100}	mm	5
Feed Size - F_{80}	mm	1.5
BWi	kWh / t	12.0

Thickener overflow in Option A is returned to the ball mill, SAG feed and the discharge hopper to maintain the correct feed density to the cyclone cluster. A trommel screen in the circuit removes scats oversize from the mill discharge. The screen undersize gravitates to the mill discharge hopper and is pumped to the cyclone cluster. The cyclones classify the ball mill feed to deliver an under-flow product with a P_{80} of approximately 75 µm. The cyclone underflow material discharges back to the mill for regrind. Cyclone overflow gravitates to the first leach tank. A particle size analyser ("PSA") is provided for online size analysis of grinding streams.

17.3.3 Leaching and CCD Plant Option A

17.3.3.1 Purpose and Background

Conventionally in U extraction the criteria in deciding whether to use acid rather than alkaline leaching is based on the carbonate content of the host rock. Typically acid leaching is preferred for a carbonate content of less than 10% (Lunt, *et. al*., 2007).

The mineralised material contains is approximately 50% calcite which accounts for the high acid consumption described in Section 13.5.3. In the design for Option A, this is addressed by removing the majority of the carbonate minerals in a pre-leach chemical mineral beneficiation step using acetic acid. An advantage of this chemical beneficiation is that high-quality gypsum is produced from regeneration of the acetic acid – and the gypsum represents a potential by-product.

Acetic acid is regenerated with less costly sulphuric acid. The overall amount of sulphuric acid used in the acidic ferric iron leach process and in the regeneration of acetic acid (Option A) is similar to that required for the stand-alone acidic ferric iron leaching process considered in Option B.

In the acetic acid pre-leaching test work completed by SGS OreTest, two of the solutions produced by acetic leaching were subjected to acetic acid regeneration tests. These tests showed that about 5% of the U reported to the acetic acid pre-leach solution. Washing and regeneration recovered almost all of this dissolved U and clean gypsum was produced. The process is designed to keep most of the solution from the acetic acid leaching section in a closed loop.

17.3.3.2 Acetic Acid Leaching

Acetic acid pre-leach is carried out at a slurry density of approximately 12% solids in a weak solution of acetic acid. This pre-leach process step conditions the remaining solids to facilitate more efficient liberation of minerals in the subsequent acidic ferric iron leach step as is further described in Section 13.5.5. Acetic acid pre-leaching is carried out at a low concentration of weak acid in a series of tanks. A residence time of more than half an hour is used.

The remaining leach residue is washed to recover calcium acetate for acetic acid regeneration and pumped to the acidic ferric iron leaching process. Further test work during the next phase of the project is required to determine the efficiency and the size of this washing circuit for more accurate design.

The acetic acid pre-leach liquor from the thickening washing step is contacted with sulphuric acid to regenerate the acetic acid. Clean gypsum is produced during the regeneration step as is described in Section 13.5.5. This regeneration reaction of acetic acid progresses to completion when insoluble gypsum exits from the system by precipitation. The gypsum is washed using a belt filter. Further test work during the next phase of the project is required to determine the efficiency and the size of this washing circuit.

17.3.3.3 Acidic Ferric Iron Leaching

17.3.3.3.1 Purpose and Background

Acidic ferric iron leaching is conventionally used to liberate and oxidise U from the ore to a soluble form by adding an oxidant such as ferric ions. Sulphuric acid is usually used to provide the acidity required. This leach process is extensively described in Section 13.5.4.

17.3.3.3.2 Process Description

In the acidic ferric iron leaching process, the residue from acetic acid pre-leaching is contacted with concentrated sulphuric acid and ferric iron. The metal-bearing flourapatite in the residue reacts with H_2SO_4 and generates compounds like phosphoric acid and sulphates in solution as shown in Reaction 4.

$$Ca_5(PO_4)_3F + 5H_2SO_4 \rightarrow 5CaSO_4 + 3H_3PO_4 + HF$$

Reaction 4: Apatite and Sulphuric Acid

The leaching of calcite and dolomite to form gypsum (calcium sulphate) as an insoluble solid occurs in an oxidising environment. The oxidising environment results from ferric ions which are regenerated by pyrolusite (natural manganese dioxide). In this process, U, recently liberated from the mineralised material, is oxidised from U^{4+} to U^{6+} by the ferric ions (Fe^{3+}) (Venter and Boylett., 2009). Reaction 5 and Reaction 6 illustrate the chemical reaction processes.

$$UO_2 + 2Fe^{3+} \rightarrow (UO_2)^{2+} + 2Fe^{2+}$$

Reaction 5: Oxidation of Pitchblende

$$2Fe^{2+} + MnO_2 + 4H^+ \rightarrow 2Fe^{3+} + Mn^{2+} + 2H_2O$$

Reaction 6: Pyrolusite Oxidation of Ferrous Iron to Ferric Iron

As a result the valuable metals, U, V, Mo, some REE, Ni, Zn and P are leached into the PLS in an oxidised form. Manganese ions are also released as a result of the oxidation of ferrous to ferric ions by using pyrolusite as an oxidant.

Table 17-3 contains a list of the elements in solution calculations based on test work results (P Miller, 2012). Acidic ferric iron leaching is carried out in two parallel series of seven tanks of 189 m^3 each fabricated from rubber-lined carbon steel.

17.3.3.4 Counter Current Decantation ("CCD") and Pinned Bed Clarifier ("PBC")

A series of counter current operated thickeners ensure the efficient washing of the solids to generate the PLS, the approximate composition of which is shown in Table 17-3. At this stage two, CCDs are designed into the circuit. Up to seven CCDs could be required. Further test work during the next phase of the project is required to determine the efficiency and the size of this washing circuit for more accurate design.

A PBC ensures that no solids are contained in the PLS.

17.3.3.5 Sulphuric Acid Production

Sulphuric acid is consumed in the processes for Option A as detailed in Table 17-7.

Table 17-7: Option A 0.5 Mtpa case H_2SO_4 consumption

Section	Amount (tph)
Acetic acid regeneration	26.98
Fluorapatite leach	10.41
Dolomite leach	0.88
Pyrolusite reaction	0.047
Molybdenum reaction	0.051
Vanadium reaction	0.196
Nickel reaction	0.137
Zinc reaction	0.301
Yttrium reaction	0.033
Neodymium reaction	0.003
U IX elution	0.118
V IX elution	0.236
Mn, Ni, Zn SX stripping	1.05
TOTAL	**40.43**

The CAPEX for the sulphuric acid plant (described in Section 21) is significant, but it would be more advantageous than importing sulphuric acid to the Berlin site for the following reasons:

- A long term supplier of inexpensive sulphuric acid may not be available in market for the large quantities required;
- Availability of acid transportation ships to transport sulphuric acid to the harbour may be low;
- Large acid off-loading and pumping systems at harbour are capital-intensive;
- Large permanent storage facilities (tanks) would be required at the harbour;
- High acid inventory costs;
- Logistical difficulties in coping with the large number of trucks required to transport the acid from the port to site;
- Shared access roads pose a potential safety risk; and
- High maintenance cost for shared roads.

Burning pyrite or sulphur are cost-effective ways of producing sulphur dioxide for the acid plant.

The pyrite to ferric reduction process (Section 17.3.3.6), produces only a small amount of SO_2 off-gas that could be used in acid production. Approximately 2 tph of H_2SO_4 can be produced from the waste SO_2 in producing ferric iron from pyrite for the 0.5 Mtpa case. This is a small amount of the overall H_2SO_4 requirement of 40.43 t/hr (Table 17-7).

The conversion of SO_2 gas to produce H_2SO_4 would be done in a conventional contact acid process that produces steam as a by-product. This steam would be used for efficient heating of the plant processes and a large quantity of electricity could be produced from the steam.

Purchasing sulphur is one of the main reagent costs. The price of the sulphur will play an important role in the OPEX of the sulphur burning acid plant. At this stage, it is envisaged that the sulphuric acid would mainly be produced from high purity sulphur obtained from local Colombian sources.

17.3.3.6 Ferric Sulphate Production from Pyrite

It is more cost effective to produce ferric by the roasting of pyrite than by pressure oxygen leaching ("POX") of ferrous iron.

The roasting of pyrite produces SO_2 that is combined with the SO_2 gas from the sulphur-burning process and is fed into the acid plant.

17.3.4 Metal Extraction and Recovery Plants Option A

17.3.4.1 Background and Purpose

The design philosophy for the extraction and recovery of valuable metals from PLS was based on:

- The concentration reaction of each metal in the different stages of PLS processing;
- The flow-rate at which the metals are transported affects the size of equipment; and
- The surrounding environment that affects the type of chemicals used and materials of construction.

Iron and manganese are also solubilised during the acidic ferric iron leaching process. Fe^{3+} has the potential to interfere in the extraction and recovery operations downstream of the CCD and clarification process. This is the main reason that relatively low concentrations of ferric iron are used with acid in the leach process.

Molybdenum tends to foul the IX process in U extraction systems. Therefore, Mo is extracted in the first step of the processing of the PLS.

In the case of U recovery, due to the high ratio of V to U concentration in the PLS, V competes strongly with U for sites of the IX resin. Given enough retention time, V depletes the U from the resin resulting in V extraction ahead of U extraction in the IX process (Merritt, 1971).

17.3.4.2 Overall Process Description

A comprehensive literature survey and combined experience of the PEA study team members was used to design, what is by necessity, a complex process to recover a wide range of metals. The following criteria were considered:

- Design for the elimination of Fe^{3+} in the PLS;
- Discrete extraction process for Mo separate from the V and U extraction circuit;
- Design for the precipitation of Mo, U and V taking into account the preferential loading of V to U; and

- Design for an overall extraction sequence optimising reagent consumption.

This has been achieved by:

- Reducing the Fe^{3+} to Fe^{2+} ions by adding Fe metal in the iron reduction tank before the Mo IX process;
- Selectively extracting Mo prior to U and V extraction (Appendix B Figure 1);
- Extracting U and V together (Appendix B Figure 2); and
- Precipitating the following three elements separately:
  - Mo as calcium molybdate ($CaMoO_4$).
  - U as uranyl peroxide ($UO_4.2H_2O$) at low pH.
  - V as ammonium meta vanadate (NH_4VO_3) at a higher pH level.

It is emphasised that for this complex separation system only a thorough integrated laboratory-pilot plant study will confirm its actual technical viability. No test work has been completed subsequent to the acidic ferric iron leach. The design for the process plant downstream from the acidic ferric iron leach has been based on theoretical and practical experience.

17.3.4.3 The Recovery of Mo, U and V

The following extraction systems cover the first steps in treating the PLS produced by the Option A leaching and CCD process.

17.3.4.3.1 Mo Recovery

Based on the information available on the distribution coefficients (K_d) for the various ions, a chelating resin was chosen for the selective extraction of Mo at very low pH (Appendix B Figure 1). The affinity of an IX resin for a specific ion is usually defined as a constant, K_d. By definition, K_d is the ratio between the metal ion per gram of dry resin and the amount of metal ion per mL of liquor at equilibrium.

From the graphs in Appendix B (Figure 1), it can be seen that by maintaining the pH at zero in the feed to the Mo IX extraction system, only Mo will be selected by the resin. The likelihood for Mo selection by the resin is about 100 times higher than that for the competing U ion.

Mo is then eluted from the resin with a 4% ammonia solution. In the process step that follows, Mo is precipitated from the eluate as $CaMoO_4$ as per Reaction 7.

$$Mo^{6+} + Ca^{2+} + 8OH^{-} \rightarrow CaMoO_4 + 4H_2O$$

Reaction 7: Calcium Molybdate Precipitation

In Option A, this reaction is achieved by adding a small volume of the calcium acetate from the stream before acetic acid regeneration. The precipitate is then centrifuged and dried. Once the Mo is extracted, the PLS can continue to flow to the next step, U and V extraction.

17.3.4.3.2 U and V Recovery

Following Mo extraction, the PLS flows to the next process step where U is extracted in its sulphate complex form as per Reaction 8.

$$UO_2^{2+} + 3SO_4^{2-} \rightarrow [UO_2(SO)_3]^{-4}$$

Reaction 8: Uranium Sulphate Complex

The system for the recovery of U and V has been designed on the basis that these elements are easily extracted from sulphuric and phosphoric acid solutions. The U and V selection priorities are shown in Appendix B Figure 2.

The fouling effect of Mo can be seen clearly from the graphs in Appendix B Figures 1 and 2. Resins prefer Mo above U and V in the relevant environments. However, most Mo is recovered by the Mo IX process before the U & V IX process. The possibility that Mo could have a fouling effect on the U and V IX resins is therefore very low.

U and V are recovered by strong anionic resins. U and V are co-eluted from the resin with a 10% sulphuric acid solution.

17.3.4.3.3 U Precipitation

The separation of U from the IX eluate is based on the precipitation of uranyl peroxide ($UO_4 \cdot 2H_2O$) at pH 4 (Merritt, 1971, Gupta*et al*., 2004, Hardwick, 1984]. Vanadium remains in solution at this pH. The reaction occurs as per Reaction 9.

$$UO_2SO_4 + H_2O_2 + 2H_2O \rightarrow UO_4 \cdot 2H_2O + H_2SO_4$$

Reaction 9: Uranyl Peroxide Precipitation

This chemical reaction also forms sulphuric acid, thus the solution is further acidified, reducing the likelihood that V will precipitate simultaneously with U.

The uranyl peroxide precipitation product then undergoes solid/liquid separation in a thickener. The underflow is pumped to the final product centrifuge and U calcining plant. The overflow reports to V precipitation.

17.3.4.3.4 V Precipitation

Ammonium hydroxide is added to the solution discharged from U precipitation to increase the pH to 7 resulting in V precipitating as ammonium meta-vanadate. Reaction 10 illustrates the process.

$$V_2O_5 + 2NH_4OH \rightarrow 2NH_4VO_3 + H_2O$$

Reaction 10: Vanadium Precipitation

Following a solid/liquid separation step the precipitate is then centrifuged and dried.

17.3.4.4 Phosphoric Acid Extraction and Concentration

Following the Mo, V and U extraction the PLS proceeds to the SX system for phosphoric acid extraction. Phosphoric acid SX is carried out with TBP and stripping is done with water (Raiter, 2010).

TBP extracts neutral complexes by a solvating action. As a result of this characteristic, the stripping of those complexes is readily achieved by washing with water. The phosphoric acid in the PLS (6.2% to ~8.3% P_2O_5) is concentrated to about 24% PA (~18% P_2O_5). This liquor has to be concentrated further to about 83% H_3PO_4 (62% P_2O_5) by evaporation of water before being sold as a product (Raiter, 2010).

17.3.4.5 Rare Earths Recovery

As all other valuable metals are now contained in the phosphoric acid SX raffinate at a low pH, the alkalinity of the PLS is elevated with ammonia to precipitate the REE as hydroxides. They precipitate and are subsequently filtered and dried to produce a mixture of REE hydroxides.

17.3.4.6 Heavy Metals Recovery (Zinc, Nickel and Manganese)

After REE recovery, the relatively alkaline PLS proceeds to the second SX system for the extraction of a mixture of Mn, Ni and Zn using D_2EHPA. Sulphuric acid is used for stripping. This occurs according to the extraction parameters described in Appendix B Figures 1 to 3.

The solvent extraction of these metals depends on the concentration of the extractant as shown in Appendix B Figure 5. Laboratory and pilot work in the next phase of the project development is required to confirm the design parameters.

Ni, Zn and Mn are precipitated from the strip solution with sodium carbonate and then filtered and dried to produce a mixture of Ni, Mn and Zn carbonates. The precipitation process reactions are illustrated in Reaction 11.

$$MnSO_4 + (NH_4)_2CO_3 \rightarrow MnCO_3 + (NH_4)_2SO_4$$

$$NiSO_4 + (NH_4)_2CO_3 \rightarrow NiCO_3 + (NH_4)_2SO_4$$

$$ZnSO_4 + (NH_4)_2CO_3 \rightarrow ZnCO_3 + (NH_4)_2SO_4$$

Reaction 11: Manganese, Nickel & Zinc precipitation reactions

The advantage here is that with only a limited manipulation of pH and the subsequent purification of the Ni concentrate, the two remaining metals can be recovered separately. The disadvantage is the amount of organic material required for this process step (at least double) and the high amount of ammonia required.

17.3.5 Uranium Product Calcining and Uranium Oxide Product Packaging Plants

17.3.5.1 Introduction

The role of the calcining and packaging sections is to remove all moisture from the uranyl peroxide precipitate, upgrade the product into its most marketable state and finally package the product safely in drums for transport.

The facility would comprise a centrifuge, a calciner and a drum packaging facility.

This area is access controlled to limit exposure to pure products. The circuit consists of:

- Uranium product thickener underflow storage tank;
- Product centrifuge;
- Product dryer;

- Product bin;
- Product packing; and
- Gas scrubbing system.

The underflow product slurry from the final product thickener is pumped to the yellowcake storage tank, and then to the product centrifuge where excess moisture is removed to produce a $UO_4.2H_2O$ cake. The cake is subsequently calcined in a rotary kiln at 350°C to form the final product UO_3. The project annual production capacity is for the 0.5 Mtpa mineralised material throughput case is an equivalent of 532 t of U_3O_8.

The facility has been designed to control yellowcake dust generated as part of the drying and drum packing. These include room ventilation, filtering of air in the contaminated room and the handling of yellowcake waste (wash down etc.).

17.3.5.2 Process Description – Overview

The uranium storage tank has the primary function of providing buffer storage capacity. The buffer capacity is large enough to ensure that upstream processes can operate continuously during operations and during minor plant maintenance shutdowns.

The storage tank is fitted with an agitator to keep the solids in suspension. Water can be added to the storage tank to reduce the solids concentration, if required. The uranium slurry is then pumped from the storage tank to the centrifuge.

The centrifuge will have a liquid discharge and a solids discharge. The discharge liquor will be gravity fed back to the product thickener. The solids discharge of the centrifuge will consist of slurry with a paste like consistency and a solids content of approximately 70%$^{w}/_{w}$. The centrifuge solids discharge into the corresponding screw feeder that in turn feeds the product dryer.

A spillage sump pump is located at the uranium product handling area. This will ensure that products are isolated in the particular area and cross-contamination is eliminated.

The product preparation and packaging area consists of an electric rotary product dryer, transfer to a storage hopper, off gas system, bag-house and the drum packing plant module.

The uranium drying plant and drum packing plant are considered to be hazardous areas due to exposure to uranium oxide dust. Hence, these plants need to be contained in sealed rooms which are maintained under a slight negative pressure. This negative pressure is achieved by a dust extraction fan that is attached to a bag-house filter. Access to both plants is typically via a clean room/dirty room arrangement.

Closed-circuit television ("CCTV") cameras will be used to allow remote monitoring of critical operational areas of the plant from the local control room.

The uranium drying and drum packing plant will be located in a secure building within the main plant perimeter, with restricted personnel access.

The modules for the uranium drying plant will consist of:

- Yellowcake storage tank and centrifuge module;

- Electric horizontal rotary dryer module;
- Dryer off-gas system and the bag-house filter module; and
- Drum packing plant module (drum conveyors, drum filling, lidding and wash stations).

The drying and packing modular containers will be supplied with a dust extraction and filtering system (baghouse), ensuring that at all times these contaminated areas will be under slight negative pressure, preventing any dust from escaping into the warehouse area. All modules will be bunded and lined (roof and walls) with suitable material to facilitate washing down and cleaning and to provide sealing, preventing the escape of any dust from the process plant area.

17.3.5.2.1 Dewatering

Dewatering consists of a centrifuge complete with a buffer feed tank, centrate holding tank, feed pumps and centrate return pump.

17.3.5.2.2 Uranium Drying

The product dryer is used to convert wet uranium oxide to a dry saleable product, UO_3. This oxide product is suitable for transport as it is insoluble in water and one of the most stable forms of uranium.

The feed conveyors transfer the uranium compound to an electrically heated product dryer which will reduce the moisture content of the feed to ≤1%$^{w}/_{w}$. The product dryer will have the capacity to heat the product to 350°C, and is housed as a modular component in its own container. The dryer includes an inlet feed screw which delivers wet cake to a furnace tube and a discharge hopper. The tube is fitted with flights to allow the cake to be transported along the length of the heated tube. The dryer has three heat zones which are heated electrically and automated. Product is discharged from the tube into a common product storage bin via a rotary product dryer discharge feeder.

The first stage of drying of the thickener underflow will be via a centrifuge, which will increase the solids concentration of the yellowcake to approximately 60% to 65%$^{w}/_{w}$. Water soluble contaminants will remain with the residual water in the discharge cake.

The centrifuge solids product will discharge into a screw conveyor. The screw conveyor will transfer the cake to an electrically heated horizontal rotary dryer. The dryer heats the cake to 350°C, producing UO_3 with a moisture content of ≤1%. It discharges product into a chute from where the dried cake is passed via a drag chain tube conveyor to the drum packing plant.

The dryer off gases pass through a scrubber and spray condenser system before exiting via the bag-house. The fully automated plant normally operates on a continuous basis.

17.3.5.2.3 Uranium Packing Plant

The drum packing plant module will operate automatically, with drum weighing, filling, sampling and un-lidding/lidding performed as automatic operations. Drums are also automatically washed and dried once they have been filled and lidded. A label is printed (with the appropriate details) which is manually stuck to the drum by the operator.

The product storage bin receives the uranium product from the product dryer and has two to three days production capacity. The product drum filling rate is controlled by the product storage bin discharge feeder and material passes through vibrating screen, to remove lumps or foreign material. The undersize continues to drum loading while oversize and foreign material are rejected. The product sampler will take cross cut samples during the drum filling sequence for monitoring of product quality.

The empty drums are manually loaded onto the feed conveyor but from there the filling, lidding, washing and weighing sequences are automated. The drums first pass through an air lock into the packing module under negative pressure to ensure no product dust is able to leave the area. The drums are then conveyed to the filling position where the product is loaded at a controlled rate until the weightometer detects the target drum weight. The drum then moves to the lidding position for the drum lid to be securely attached for transport. The drum is then spray washed, dried via a blast of air to ensure no product has settled on the outside of the container prior to passing through the other airlock. It is weighed again by weightometer where the official weight is recorded and individual identifier stickers are printed. The operator will affix the labels to the drums before they are removed from the conveyor and loaded into sea-container.

17.3.5.2.4 Off-Gas Handling & Bag-house Module

The off-gas system consists of two main duties namely dry dust extraction scrubbing and off-gas scrubbing. The dust scrubbing occurs after maintaining negative pressures in the five modular plants. In the process technician work areas, the off-gas scrubbing duty consists primarily of the dryer off-gas capture. An individual scrubbing module is proposed for the dryer, with a second system for reagent and SDU vent gas scrubbing and the building dust management.

The off-gas system uses a liquid ring vacuum pump to draw evaporated steam away from the dryer. The off-gas (steam and entrained solids) passes through a spray condenser, and then through a venturi scrubber and a cyclonic separator, condensing the steam and removing any entrained solids. The gas then passes through a mist filter before passing through the bag-house. The solids phase will be pumped back to the thickener. The spray condenser water operates on a closed loop. Water from the seal tank is cooled by a heat exchanger. A refrigerated chiller is used to remove the heat load. All of this equipment will be housed in the off-gas & bag-house module.

The dry dust scrubbing system utilises multiple dry filter cartridges in a bag-house where a fan maintains negative pressure in the unit. The solid particulates contained in the drawn air are retained on the filters. Air is supplied to provide reverse flow air pulses on a suitable cycle to detach the solids from the filters. The solids drop into the bag-house catchment where a control valve discharges them into a sealed bin. The bin is manually emptied as required.

17.4 Option B: Direct Acidic Ferric Iron Leach without Acetic Acid Pre-Leach

17.4.1 Background and Purpose

In general, the size of the equipment in Option B has been assumed to be similar to Option A, as are the reagent consumptions. The comminution circuit is similar for Options A and B. In Option B, the acetic acid pre-leach is excluded from the process (Figure 17-5). Approximately the same amount of sulphuric acid is used in Options A and B.

Refer to Table 17-1 for the Option B specific design documents.

The main differences between Options A and B leaching circuits and the effect on the Option B design are:

- The reaction of sulphuric acid with large quantities of calcite will result in a volatile reaction and could have an effect on the leaching efficiencies. Despite this, test work shows that there is little change in extraction efficiency with changes in the amount of acid used in acidic ferric iron leach tests (Section 13.5.4). Laboratory and pilot test work in the next phase of project development is required to confirm these results at larger scale;

- Higher flow rates influence CCD washing efficiencies through increased liquid volumes;

- Greater quantities of gypsum are formed in acidic ferric iron leach and increase the volume of slurry to the CCD and tailings circuits; and

- Higher flow rates to the metal recovery plants may require larger equipment and may affect design efficiencies.

To date, small scale laboratory test work has been completed to determine leaching efficiencies. Larger scale test work is required in the next phase of project development to determine the effect of acidic ferric iron leach reacting with large quantities of calcite on the environment.

Test work has not been conducted on the effect of gypsum in the CCD circuit. For the purposes of this study, the washing efficiencies for both cases has been assumed at 100%. Test work is required in the next phase of project development to determine the washing efficiencies for both options.

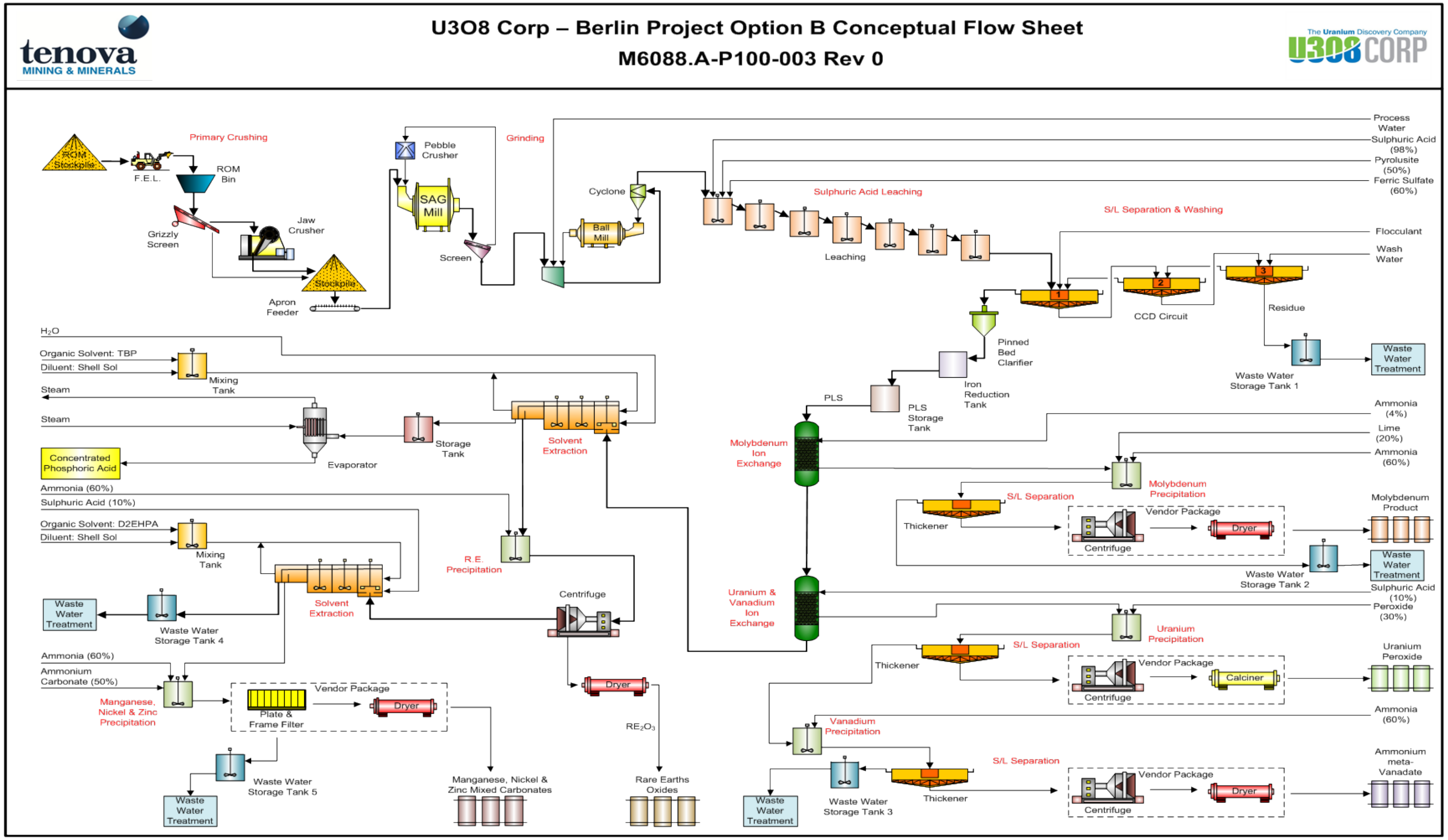

Figure 17-5: Flowchart of overall process flow for Option B

17.4.2 Option B: Process Description: Leaching and CCD

Calcite and apatite react with sulphuric acid in the leach process resulting in an increased amount of gypsum. This reaction is illustrated in Reaction 12.

$$CaCO_3 + H_2SO_4 \rightarrow CaSO_4 + CO_2 + H_2O$$

$$Ca_5(PO_4)_3F + 5H_2SO_4 \rightarrow 5CaSO_4 + 3H_3PO_4 + HF$$

Reaction 12: Calcite and Fluorapatite Reactions with sulphuric acid

The mineralised material would have a slurry density of 30% solids and would be fed directly to the acidic ferric iron leach.

The leaching of calcite and apatite and the formation of large amounts of carbon dioxide (CO_2) and calcium sulphate or gypsum occurs in an oxidising environment perpetuated by ferric ions and sustained by pyrolusite. This takes place in a series of oxidation-reduction reactions directed at the oxidation of U in the mineralised material similar to that of Option A.

As in Option A, valuable metals are being leached into the PLS in an oxidized form. Based on the SGS OreTest test work (Section 13), the recovery efficiencies have been assumed similar for both options. The grade however differs because of differences in flow rates and solids mass percentage. For details, see mass balance document for Option B referred to in Appendix G.

As leaching occurs in a complex and aggressive environment, a long residence time is required to achieve the desired oxidation and liberation efficiencies. The retention time for the leaching train has been maintained at 10 hours however, the size of leaching tanks has been more than doubled to take into account the extra gypsum formation.

While the oxidative environment employed in this leaching section has been directed towards U oxidation, the leaching of V and Mo have also benefited from it. As in Option A, ferric ions could also be released into the PLS which will complicate the separation of metals from the PLS further downstream.

Direct acidic ferric iron leaching may result in lower metal extractions and will result in an increase in PLS volume in comparison to Option A. At this stage however, for Option B, maximum dissolution efficiency of PLS components has been assumed similar to Option A.

The size and number of CCDs has been increased to ensure efficient washing of solids to generate the PLS. CCD size and number increases are:

- Thickener surface area more than doubled;
- Thickener diameter increases about 44%;
- Thickener numbers increases from 2 to 3; and
- PBC size increased by about 30%.

Test work is required to determine the sizing of the thickeners. The quantity of thickeners required could increase to more than seven for the high washing efficiency assumed. Pilot test work is also required to determine a more accurate washing efficiency to be used in the next phase of project development. The inclusion of belt filters should be considered.

A final PBC ensures that no solids remain in the PLS prior to the IX and SX processes. Specific test work is required to confirm the PBC design for both options in the next phase of project development.

17.4.3 Option B Metal Recovery Process Description and Comparison

17.4.3.1 Summary

The mass flow rate increases by 24% with a minimal effect on the metal recovery plant.

In general, the size of the equipment in Option B has been assumed to be similar to Option A, as are the reagent consumptions. The comminution circuit is similar for Options A and B. The acetic acid pre-leach has been excluded in Option B.

Table 17-8: Option A vs. B: PLS CCD overflows to metal recovery plant

	Option A	Option B	Increase
Total (Pulp) Mass Flow Rate (tph)	194.4	257.3	24%
Total (Pulp) Volume Flow Rate (m³/h)	180.0	245.1	27%

17.4.3.2 Mo Recovery

Mo is eluted from the resin with a 4% ammonia solution. Reaction 13 illustrates the precipitation of Mo from the eluate as $CaMoO_4$.

$$Mo^{6+} + Ca^{2+} + 8OH^{-} \rightarrow CaMoO_4 + 4H_2O$$

Reaction 13: Calcium Molybdate Precipitation

In Option A, this reaction is achieved by using a small portion of the calcium acetate before acetic acid regeneration. In Option B, the Mo precipitation is achieved with calcium chloride. This precipitate is then washed, filtered and dried similar to Option A.

17.4.3.3 U and V Recovery

Once Mo is extracted, the PLS continues to the U and V extraction IX step. U and V are extracted in their sulphate complex form as per Reaction 14 (shown for U only).

$$UO_2^{2+} + 3SO_4^{2-} = [UO_2(SO)_3]^{-4}$$

Reaction 14: U Extraction in Sulphate Form

Option B recoveries may be reduced by the higher PLS flow rates (the higher PLS flow rates will result in a lower metal concentration in solution). Test work should be conducted in the next project phase to determine if equipment sizes should be increased and if the U and V recoveries remain the same.

Option B recoveries may be affected by the higher PLS flow rates. Test work should be conducted in the next project phase to determine if equipment sizes should be increased and if the V recovery remains the same.

A PLS flow rate increase of more than 30% would necessitate a review of the size and number of IX equipment units. For the purposes of this study, the flow rate increase does not warrant a review of equipment size and numbers. This should be reviewed in the next phase of project development.

17.4.3.4 Phosphoric Acid

The size of the PLS and organic tanks in the phosphoric acid SX circuit are increased in Option B with the remaining equipment being unchanged.

17.4.3.5 REE Recovery Option B

The recovery of REE is based on increasing the pH of the PLS. By increasing Option B's PLS volume, the REE concentration is lowered, and thus precipitation becomes less efficient. The recovery of REE in Option B is thus less likely to be viable.

17.4.3.6 Acid Zn, Mn and Ni Recovery in Option B

If REE extraction is excluded from the Option B process the feed PLS to the Zn, Mn and Ni recovery sections will have a low pH. Such a pH has the benefit of being suitable for the separate extraction of Zn, while Ni and Mn can also be extracted separately at higher pH values.

17.5 Further Work and Discussion

17.5.1 Comminution

It is recommended that further investigations be carried out in the next stages of the project development into the effects of varying characteristics of the mineralised material on comminution and leach.

The design of the milling circuit should be developed further in terms of:

- The type, power draw, diameter, grinding length and the ball charge of the SAG mill;
- AG vs. SAG grinding should be investigated;
- Should AG be viable, the ball milling operation could probably be removed from the process flowsheet. The process recycle water used for wet screening is a solution recycled from the various thickeners in the process. The effect of these on the grinding equipment and process should be determined by locked-cycle test work;
- Variability in breakage characteristics of the mineralised material is expected because the material will contain varying amounts of competent carbonate and apatite. Detailed investigation of the characteristics of the mineralised material should be investigated further;
- A detailed study of the underground layout of the mine and the space required for the crushing station is recommended with focus on providing enough space for a SAG mill; and
- The SAG mill could also be located at the plant, because SAG (or AG) and ball mills are wet processes and dust will not be a problem.

17.5.2 Leaching

17.5.2.1 Option A

The acetic acid leach residue would be washed and transported to the acidic ferric iron leach process. Further test work during the next phase of the project is required to determine the efficiency and the size of this washing circuit. The mass balance around the acetic acid leaching should also be investigated and developed in more detail. Further detail modelling of the circuit flow is required.

17.5.2.2 Option B

- The reaction of the acidic ferric iron leach with large quantities of calcite in the ROM material in the leaching section results in a volatile reaction. The effect on leaching efficiencies and other potential influences on design should be determined by pilot plant test work;
- Investigate the effects of higher flow rates on downstream processes; and
- Investigate the effect on downstream processes of large quantities of gypsum formed during acidic ferric iron leaching.

Test work on the characteristics of the different slurries is required to determine the size of the thickeners. The quantity of the thickeners required could increase to more than seven for the high washing efficiency assumed in Options A and B. Pilot test work is also required to determine the washing efficiency for the residues of Options A and B (that result in different slurries fed to the CCD). Current washing efficiency assumed is 100% for both cases. The use of belt filters instead of thickeners for washing should also be considered.

Test work is required to determine the design parameters of the PBC.

17.5.3 Metals Recovery Plant

It is emphasised that only an integrated pilot plant campaign for this complex separation system will confirm its technical viability. No test work results were available for the design of this part of the plant.

Difference in Mo recovery: The recovery system could be affected by the higher flow-rate of the PLS. A more detailed design would determine if the system equipment size should be increased and if the recovery of Mo will stay the same. Test work to compare the two systems using the different precipitation reagents is required for the next stage.

The technical viability of REE recovery, especially for Option B, needs to be determined by test work.

The power requirements for the different sections should be designed taking into account future test work results. During the study, conservative assumptions were made in preparing the energy balances, including the heat and power available from the acid plant. Further more detailed design work is required to determine the heat and energy requirements. It may result in further lowering of the CAPEX and OPEX costs assumed.

The SX of Zn, Ni and Mn is complex and it is almost impossible to predict the extraction behaviour. The theoretical design of the plant must be followed up with extensive laboratory and pilot plant test work in the next stage of the project development.

18 PROJECT INFRASTRUCTURE

18.1 Access

18.1.1 Regional Roads

Principal access to the site is by an all-weather, gravel road from the transport hub at the town of La Dorada. La Dorada is a focal point where the paved highway between the Bogota and Medellin, the Rio Magdalena and a railway line meet. From La Dorada, a paved road provides access to the port town of Santa Marta on the Caribbean coast some 850 km to the north. An additional paved road links La Dorada with the port at Buenaventura that lies 450 km to the southwest on the Pacific coast.

Dredging, funded by the Colombian government, is currently underway on the section of the Rio Magdalena closest to La Dorada to enhance the navigability of the river by barge from the port of Puerto Salgar at La Dorada to the port of Barranquilla on the Caribbean coast. The railway line that passes through the town of La Dorada is defunct at present, but has been prioritized in the government's infrastructure development program and is slated to be brought back into service in 2015. The railway provides an alternative link between La Dorada and the port of Santa Marta. Upgrades to the railway system would also provide links from La Dorada to Ibague in the south and Bogota in the southeast (Figure 18-1).

Given the importance of La Dorada as the principal hub through which most equipment and reagents would be transported, the road from there to the project site requires upgrading from the current partially paved 5 m wide road bed to a 7.5 m width. The first 41 km of the road is located on a watershed through savannah over sandstone and conglomerate strata that provide a solid road base. The final 15 km of the existing access road is in steeper terrane with a change in altitude of 570 m from 280 m amsl in the savannah to 850 m amsl at the plant site. The existing road is on a granite and metamorphic schist base, and passes over two concrete bridges that have load capacities in excess of 75 t.

18.1.2 Site Roads

Site access roads are planned to be 5 m wide while haul roads would be 15 m wide. Both types of road bed would be stripped of humus and paved with granular road metal. Granites within the project area could potentially provide road metal of the required quality. Drainage ditches and culverts will be placed in accordance with the site drainage requirements.

18.1.3 Security

Mine infrastructure and the plant site would be enclosed with a 2 m chain link fence. Access to the plant site would be restricted through one access gate that would be manned 24 hours a day. Electrical substations would be fenced in a similar way.

Figure 18-1: Map showing the regional infrastructure of importance to the Berlin Project

18.2 Site Location

18.2.1 General Layout

18.2.1.1 Mine Complex

The optimal position for the portal for an underground mine designed for the exploitation of the initial mineral resource is shown in Figure 18-2. The current design contemplates crushing and milling of mineralised material in an underground chamber to mitigate against dust that contains radionuclides escaping into the environment. A workshop and warehouse would be located within the mining complex. The workshop would be constructed with prefabricated concrete with a spread footing design. The workshop would include two indoor mobile equipment repair bays equipped with an overhead travelling crane, a small vehicle repair bay and one outdoor wash bay equipped with high pressure water monitors and a sloped concrete pad to an oil/water separator. The mine site would incorporate an area for fuel storage tanks and a lubricant storage shed.

18.2.1.2 Processing Plant

The most suitable location for the plant site in reasonably close proximity to the portal and mine infrastructure is a relatively flat area located 630 m to the southeast of the portal. The conceptual plant layout is illustrated in Figure 18-3. The sewage processing plant lies within the perimeter of the processing plant.

18.2.1.3 Support Facilities

Support facilities include an administration building, clinic and a recreation area that contains the kitchen, mess hall and recreation facilities. The site area for the plant and buildings would cover an area of approximately 250,000 square metres ("m^2"). Local building materials would be utilized wherever practical and cost effective. Local buildings are primarily concrete and block-work structures.

18.2.2 Waste Rock Facility ("WRF")

It is envisaged that approximately half of the waste rock (i.e. 125,000 tpa) would be crushed in the underground crushing facility for blending with tails for backfill in the underground mine. Excess waste rock will be trucked to surface using the underground production trucks and deposited close to the production ramp (Figure 18-2).

The foundation of the WRF will be cleared and grubbed, and all low resistant materials (SPT<8) removed. Waste rock would be dumped in 5 m lifts and spread with a bulldozer and compactor. Construction will be continuous throughout the life of the facility, and the total capacity of the facility is estimated at one million cubic metres.

All waste material is considered to be inert, and this will be confirmed through further test work of the ROM waste as the project advances. The *in situ* density of the waste rock in the WRF is estimated to be 2.0 t/m^3.

Internal composite-type drains in the major drainages will collect base flow and percolated surface flow, while a series of surface drainage structures will collect runoff and safely carry it away from the facility. Internal surface drainage structures will be sized for 100 year return storm event and the peripheral structures for 500 year event. A small downstream sediment control dam will collect runoff, and settle the fines from the WRF during operations to ensure compliance with local environmental water quality requirements for suspended solids. The geometry of the WRF will be adequate for long-term stability of the facility.

Reclamation would involve covering the WRF with approximately 30 cm of organic material, re-vegetation with native species and final installation of surface drainage canals. The downstream sediment control facility would be decommissioned at closure.

BERLIN OPERATION INFRASTRUCTURE

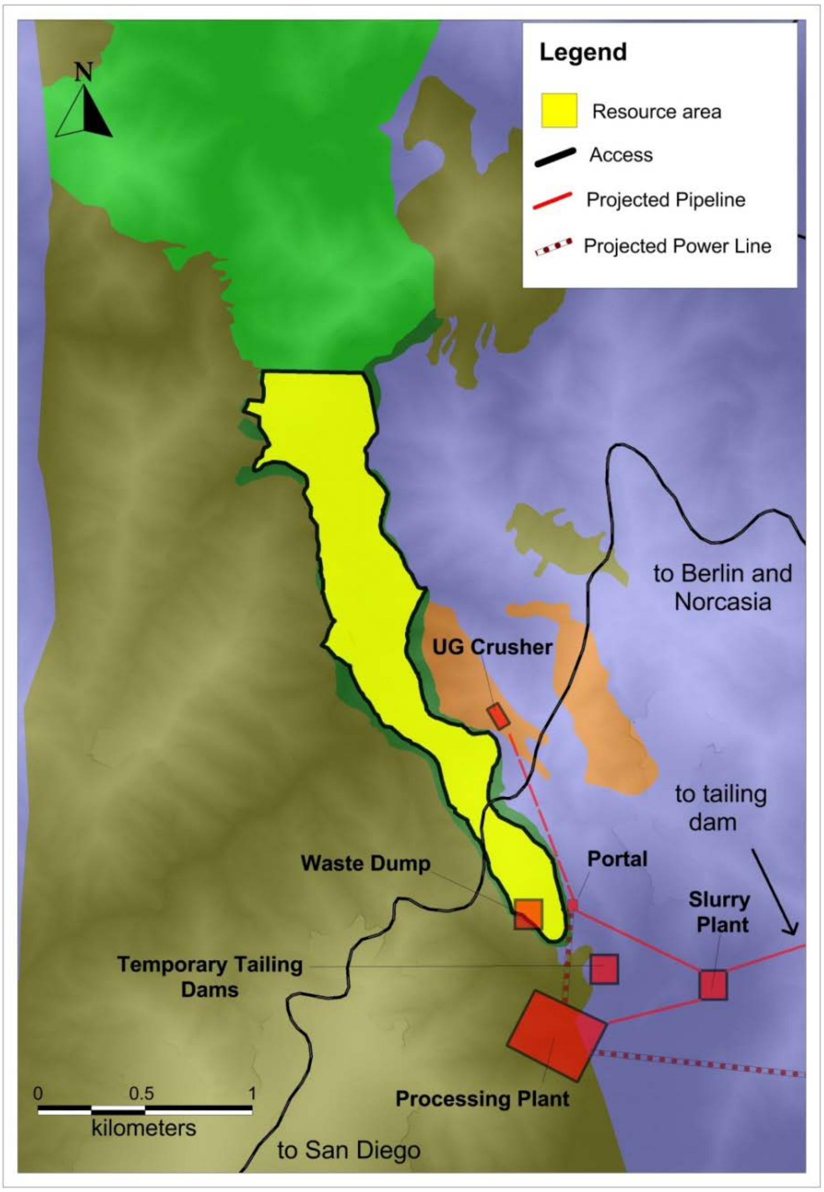

Figure 18-2: General layout of infrastructure related to the Berlin Project

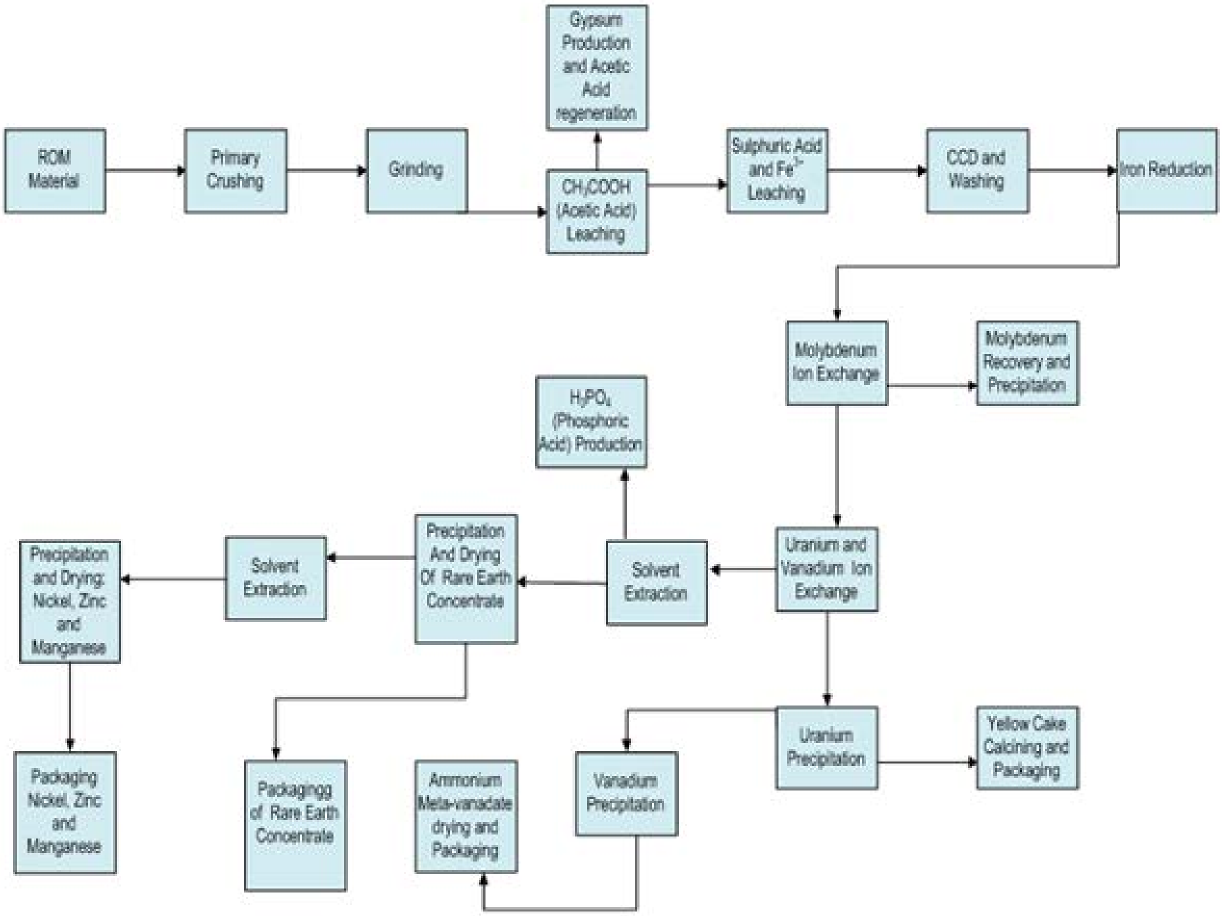

Figure 18-3: Conceptual layout of the processing plant envisaged for the Berlin Project

18.2.3 Waste Management

Solid waste generated from the mine plant site, including ancillary buildings, would primarily be domestic and industrial non-hazardous waste. A comprehensive Waste Management Plan will be developed for the project.

Solid waste will include:

- Refuse from construction (for example: scrap wood, metal and concrete);
- Refuse from the mine (for example: empty drums and packing materials); and
- General domestic garbage from the offices and ancillary buildings (for example: paper, refuse and food).

Construction debris, inert waste and used tires would be placed in designated cells. Solid domestic and industrial waste from the mine plant facilities will be recycled and re-used in an approved manner, where feasible. Other solid waste will be placed in adequately lined waste receptacles.

18.2.4 Tailings Management Facility ("TMF")

Two mineral processing options are under consideration at present. One involves beneficiation of the mineralised material to remove carbonate, which generates a mass reduction of over 50%. In this scenario, there is sufficient decrease in volume between the mineralised material mined and the tailing generated that all of the latter could be pumped back underground as a paste for backfill.

The second option under investigation is to process the entire run of mine without beneficiation of the mineralised material. The second option results in an excess of tailings that cannot all be accommodated in mined-out areas underground and an above-ground storage facility would be required. It is envisaged that tailings would gravitate through a slurry pipeline to a site located approximately 14 km from the plant where substratum are competent metamorphic and igneous rocks that have minimal permeability (Figure 18-4).

Two alternative tailings management approaches will be evaluated as the project advances. The first, a conventional tailings dam, is the option that is used for the basis of the operating and capital cost estimate presented in Section 21.

18.2.4.1 Conventional Mine Tailings Management Facility

The TMF would have an ultimate reservoir capacity of approximately 4 million cubic metres to contain tailings that are estimated to have an *in situ* density of 1.8 t/m^3 and would have an operational life of 15 years. The dam wall is designed as a compacted earth fill structure with an internal drain system and cut-off for seepage control. The proposed reservoir would be lined with compacted clay since the tailings have no acid rock drainage potential.

The dam embankment and clay liner borrow material will be obtained from nearby sources and shall be classified as CL (low-plasticity clay) and/or ML (low-plasticity silt) according to the Universal Soil Classification System. Sands for filters and concrete will be screened and washed from nearby stream beds. Gravels, road base and riprap will be sourced from the existing waste rock material, and crushed and screened on site as required.

The proposed spillway would be a rectangular concrete and riprap structure designed for a 1,000 year storm event. The spillway would be raised along with the dam crest. At the end of the life of the facility, a spillway designed for the 10,000 year design storm event will be constructed for closure.

Solids deposition will be via a 15 cm diameter traditional slurry pipeline with a spigoting from the dam crest in order to maintain a slope such that flow is away from the dam wall. All upstream basin runoff will be collected in the reservoir and reclaimed for use as process water using a floating barge pump.

During the final months of operation, the spigoting plan will be adjusted so that the final closure surface drains towards the spillway. The surface would be finish-graded for closure as required, capped with organic material and re-vegetated with native plants to protect against surface erosion. Riprap lined channels would be designed to collect runoff and safely convey flow towards the closure spillway.

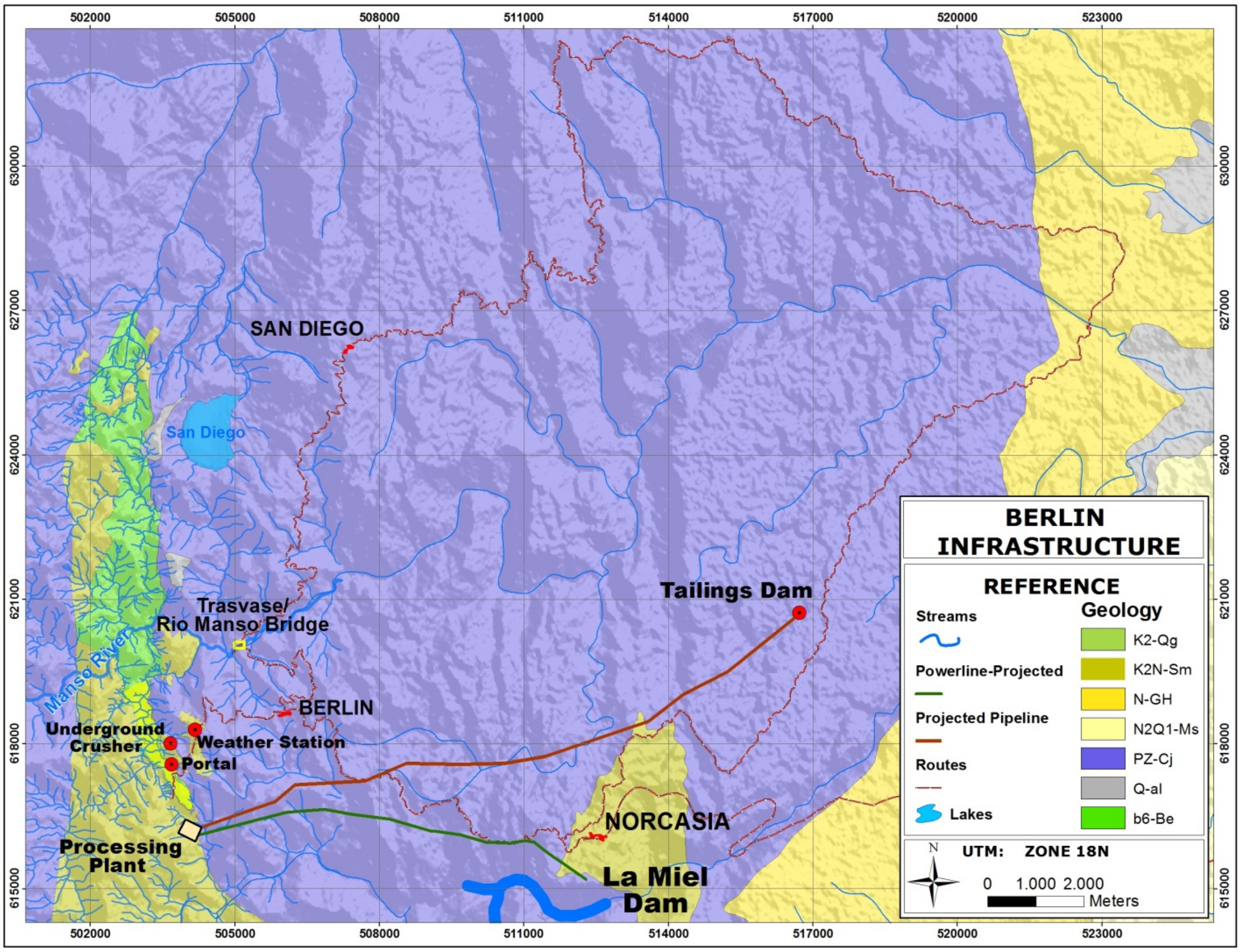

Figure 18-4: General layout of infrastructure related to the Berlin Project

18.2.4.2 Pod-type Tailings Facility

This concept involves the use of synthetic pods for the storage of the tailings (Figure 18-5). Tailings gradually dry in these pods and the fact that the tailings are confined to isolated cells within the facility minimises the risk of contamination if the tailings facility were to breach. These nested pods would be covered with a thickness of soil adequate to mask any radiation emanating from the tailings.

Figure 18-5: Synthetic pods that could be used for tailings disposal. In the case of Berlin, such pod-fields would be covered with a layer of soil thick enough to mask any radiation that may emanate from the tailings

18.3 Electricity

18.3.1 Supply

The electrical system is sized to take into account the process loads of the crushing plant, conveyors and process plant as well as loads from the ancillary buildings, including the workshop and warehouse, mine canteen and administration buildings. The current design includes spare capacity within the electrical distribution system to allow for further expansion of the process plant in the future.

Approximately 46% of the electricity required for the entire operation is planned to be generated from the sulphuric acid plant. About 75% of the plant's required power could be generated from the heat produced by the sulphuric acid plant. The remainder of the power required for the operation would be drawn from an 8 km long overhead power line that would link into the regional electrical grid that incorporates electricity generated from the La Miel hydroelectric dam located 8 km from plant site (Figure 18-4).

18.3.2 Distribution

The pole-line from the national grid would terminate at the main substation located near the plant site. The main substation would consist of the main disconnect, metering facilities, main transformer, medium voltage circuit breaker and medium voltage switchgear transformer and metering system. The medium voltage switchgear would be enclosed in a self-contained, walk-in modular switch house. Distribution from the main substation would include the following feeders:

- A 34 kilovolt ("kV") overhead power line will provide power distribution to the underground mine, surface installations of the mine, conveyor systems and the process plant; and

- The 34 kV overhead power line will provide power distribution to the electrical distribution equipment for the workshop complex, administration buildings, kitchen complex and mess hall, sewage treatment plant and fuel storage facility.

The process building and power system modules would include outdoor oil-filled transformers, motor control centres ("MCC"), power distribution centres ("PDC"), indoor dry-type transformers for the local 380 or 440 volt circuit, one-phase distribution panels and local control devices. All electrical distribution would be in cable trays using amour-interlocked PVC coated cables.

The process and plant site ancillary facilities switchgear and electrical equipment would be installed in modular electrical rooms adjacent to or within their respective buildings where economically feasible. In non-process areas, such as the administration building, kitchen and mess hall, sewage treatment plant, fuel storage facility, water tanks and workshop complex, a combination of armoured-type cable and rigid-galvanized steel conduit and wire systems would be used where wiring would otherwise be exposed. Motor control centres would contain motor starters, contactors, disconnect switches, transformers, panels, circuit breakers and fuses.

18.4 Water and Sewage

18.4.1 Water Supply and Distribution

18.4.1.1 Water Sources

Three principal water sources are contemplated. After the initial mine development, the principal source would be from underground dewatering. The second most important volume of water would be from recycling from the processing plant and the third would be a supply of fresh water and make-up water for the plant from the Rio Manso, a perennial water source located 4 km from the plant site. Water would be pumped from the river though a buried high-density polyethylene ("HDPE") pipeline.

18.4.1.2 Water Requirements

Based on the modelled 1,430 tpd ROM, the water requirement would be approximately 100m^3/h to 150 m^3/h (Table 18-1).

Table 18-1: Estimated volume of new water required for the conceptual mine and plant at Berlin

Items	Water Requirements	Remarks
Mine	12.5 m^3/h * 16hs= 200 m^3/day	Mainly for drilling, recycling from pumping
Plant	Option A: 134 m^3/hr (or 3,216 m^3/day)	Requirement for new water. The remainder of the required water would be recycled within the plant.
	Option B: 83 m^3/hr (or 1,994 m^3/day)	
Potable water	2 m^3/hour * 16 hours = 32 m^3/day	Drinking water

18.4.1.3 Water Storage

A reservoir complex would be located at a site that lies at a higher elevation than the plant. Water would gravitate through a reticulation system, comprising buried PVC pipelines, pumps and secondary reservoirs. The reservoir complex would contain a 10,000 m^3 capacity fresh water tank, a potable water tank, 5,000 m^3 capacity tanks for mine water and recycled water, a fire-water tank as well as water treatment facilities.

18.4.2 Water Treatment and Distribution

18.4.2.1 Potable Water

Fresh water would undergo treatment to provide potable water for human consumption. Water treatment would consist of filtration and hypochlorination before passing into a lined, above-ground potable water storage tank adjacent to the fresh water tank. The small mixing tank and a metering pump associated with the hypochlorinator will be isolated within a container. From the water treatment facility, water would gravitate through an underground PVC pipeline system throughout the site.

For option A for a 0.5 tpa ROM throughput about, 134 m^3/h (or 3216 m^3/day) new water will be consumed. Nearly 42% will be used for wash water for the CCD, 38% as washwater for the gypsum belt filter and the rest for the SX plants. An optimum amount is recycled. All water to the comminution and leaching component of the processing plant and a big part to the metal recovery section of the plant would be directly recycled within a partly closed loop process system. Waste water out of the system from the different plants will require treatment. This would be added to the tails stream that will assist with the regeneration of the water. Excess water from the tailings would be recycled.

Bottled potable water will be supplied to all other locations, i.e. administration building, control rooms and the process plant.

18.4.2.2 Fresh Water

The fresh water would be required to augment water requirements in the processing plant and for road watering. Road watering would be provided by a stand pipe located near the primary crusher.

Fresh water supply would be gravity flow from the fresh water tank in the reservoir complex through buried HDPE pipe to the point of service. Above ground distribution pipes would be carbon steel.

18.4.2.3 Fire Water

As both fire and fresh water tanks are elevated on the surrounding hillside, gravity flow would be utilised for the water distribution. Utilising gravity flow eliminates the need for fire water jockey pumps and diesel driven fire water pumps. This provides an additional element of security since the fire water system is independent of electricity and fuel requirements. An alarm would be sounded at the plant site should the pressure in the system drop.

The fire water system would consist of a buried fire water loop and hydrant system at the plant site and ancillary buildings, and at the process plant. Hose cabinets would be placed at the fire hydrant locations and the system supplemented with portable fire extinguishers placed within the process facilities. The administration building and mine facilities, kitchen, mess and recreation area would all have overhead sprinkler systems. Emergency showers and eyewash stations would be located throughout the process facilities.

18.4.3 Sewage Collection and Treatment

The sewage processing facility would be located at the plant site. Sewage would gravitate from the plant site and ancillary facilities to the treatment plant via a buried gravity collection system comprised of buried PVC pipe and concrete manholes.

Sewage collection in remote areas would be via a system of holding tanks that would be pumped out by truck and transported to the sewage treatment plant. The sewage treatment plant effluent would be pumped to a tile field for below-ground disposal. There would be no surface disposal into the environment.

All excess water from the plant would be treated and neutralised before being discharged into a system of earth-walled dams for further purification by natural biological processes.

18.4.4 Fuel and Lubricant Storage and Distribution

Diesel fuel would be delivered to the site by tanker truck. Diesel fuel requirements for the mining equipment and process and ancillary facilities would be supplied from a diesel fuel storage tank located at the truck shop. Diesel fuel distribution would be limited to loading and unloading facilities and metering equipment at the diesel fuel tank. Lubricants would be delivered to the site in drums. The drums would be stored in a secure area. The lubricant would be distributed to hose reels in the truck shop service bay with barrel pumps.

18.5 Communication and Fire Alarm System

The majority of on-site communication would be with wireless systems. Where hard-wiring is required for security or to provide backup to the wireless system, communication cabling would be supported on messenger wire under-built to the pole line and/or run underground in 5cm conduit to the respective buildings.

A complete self-contained fire alarm system would be installed in all buildings to meet international safety codes and insurance underwriter's regulations for fire protection.

19 MARKET STUDIES AND CONTRACTS

19.1 Market Studies

Neither U3O8 Corp. nor Tenova have conducted a market study in relation to the various commodities which may be produced from the Berlin deposit. The multi-commodity profile of Berlin provides exposure to various industries and should offer a natural hedge between the different markets.

The uranium commodity markets are volatile. Due to the potential for security of supply issues related to growing nuclear energy programs worldwide, long-term prices are higher than the spot price. About 75% of uranium sales are in long-term, multi-year contracts. The Berlin PEA is based on $60 /lb, which is the average reported long-term uranium price over the previous 12 months. The current uranium spot price is about $45 /lb, which reflects discretionary buying for typically single deliveries within 12 months of the contract award.

The uranium, vanadium, mixed rare earth oxides and base metals from Berlin would be sold to overseas markets. In the case of the mixed rare earth oxides, sales would be contracted with specific buyers such as rare earth refineries in China and around the world as there is no open market for rare earths at present. U3O8 Corp. predicts strong demand for the phosphate from the Berlin deposit from both domestic and regional agriculture markets such as Brazil. The project is also located 60 km from Colombia's agricultural heartland.

If acetic acid is used in the beneficiation step of the process, a gypsum by-product would be generated. Over 90% of gypsum produced worldwide is used in the construction industry, principally in drywall or plasterboard and cement. Potential markets for gypsum are the fast-growing local Colombian building industry as well as regional and international markets. The adoption of drywall instead of concrete for interior walls in construction has reduced building costs by approximately 25% in Brazil. At a production rate of 1.2 Mlb uranium per year, Berlin will generate about 400,000 t of gypsum, which could satisfy most of local demand. Colombia imports about 75% of its gypsum mainly from Spain and some from Canada. Two drywall plants are located directly downstream of Berlin at the mouth of the Magdalena River in Barranquilla and a third drywall manufacturer is located in Cartagena. Gypsum is also used in the cement industry and a cement plant is located in Ibague (Figure 19-1).

The Berlin Project is located approximately 50 km from the town of La Dorada, which would be the main junction for transport routes to both local and international markets. La Dorada provides road access to Colombia's largest cities of Bogota and Medellin and to the Pacific Ocean as well as port facilities on the Magdalena River that is navigable by barge to the port of Barranquilla on the Caribbean coast. Barranquilla is the largest port in Colombia and provides access to the export destinations of the Caribbean, Central America, the southern U.S. and northern South America. A defunct railway line also runs from La Dorada to the port town of Santa Marta on the Caribbean coast. The Colombian government is reported to be planning to have the railway line operational in 2015, which would offer an alternative link between the project and the Caribbean coast.

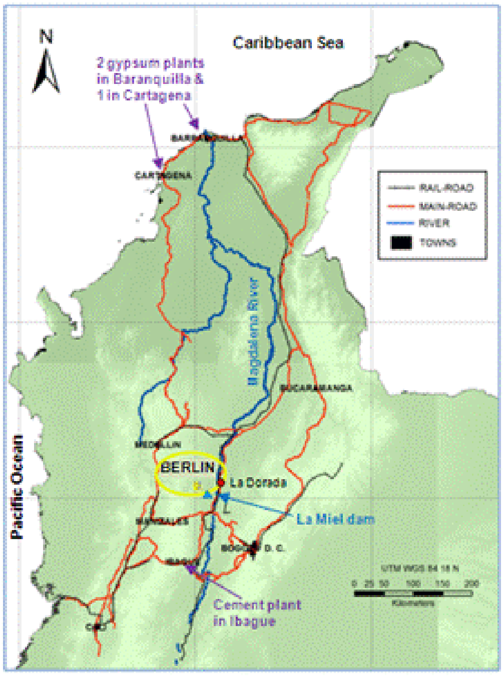

Figure 19-1: Location of infrastructure and potential markets for commodities from the Berlin deposit

19.2 Contracts

There are no sales contracts currently in place for the Berlin Project.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

Although no specific environmental license and social or community-related plans are required for the exploration stage of the Berlin Project, all work must be done in accordance with environmental guidelines issued by the Ministry of Mines and Energy and the Ministry of the Environment (Section 4.2.3). Nonetheless, U3O8 Corp. has been pro-active in its environmental and social practices and hired an external consultant to perform an initial environmental assessment of the area while staff undertook a socio-economic study of the Berlin village and neighbouring communities. These internal baseline studies helped to guide U3O8 Corp's exploration in order to minimise its environmental impact, create a safe and healthy workplace and foster community engagement at an early stage in the project.

20.1 Environment

20.1.1 Environmental Baseline Study

U3O8 Corp. conducted an initial environmental review including an inventory of the fauna, vegetation and study of surface water quality to gather baseline information of the local environment and identify areas of sensitivity.

Fauna

Fauna was characterized using visual detection and capture in mist nets and Sherman traps. In addition to observational traverses, informal interviews were held with local residents. Amphibians were studied through unrestricted searching during the night and traverses during the day. All available micro habitats were studied in detail.

The study recorded the species of birds, flying mammals, amphibians, reptiles and macro-invertebrates in the area and identified any species that may have high susceptibility to disruption or are listed as threatened on national or global registries.

Vegetation

Traverses were conducted to characterise the whole zone of influence around the Berlin Project. Areas were categorized as cultivated, grasslands, low scrub, tall scrub, secondary forests and primary forests. The most representative species were identified for each category and a record was made of those species that are exclusive to the Magdalena River valley, endemic to Colombia, distributed in other neotropical countries in Latin America or listed as vulnerable on national or global registries. In addition, an inventory of the vegetation was performed in 250 m^2 circles centred on each of the drill platforms.

Area vegetation consists of a primary forest cover, with cleared zones including weeds, secondary forests, pastures and crops of sugarcane, corn, cassava and coffee as a result of a long history of land use by the residents. The most significant coverage from a biodiversity standpoint is confined to the steep areas, mainly located on the west part of the Santa Marta River and small tributaries on the eastern side, where the steep landscape does not allow for the establishment of crops.

Physical, chemical and microbiological characterization of surface waters

Measurements were made on six point samples from streams in catchment and dumping sites in the study area. *In situ* measurements were made of pH, temperature, conductivity and volume of flow. Laboratory measurements yielded dissolved oxygen, alkalinity, hardness, settleable solids, total suspended solids, chemical oxygen demand and biochemical oxygen demand.

20.1.2 Environmental Initiatives

Protocols have been put in place at the Berlin Project to comply with environmental guidelines as well as pro-active initiatives have been undertaken with regard to restoration of exploration sites, minimising the environmental footprint of the exploration program, water monitoring, waste management and workplace health and safety, which include the following:

Man-Portable Drill Rigs

The Company uses modular rigs whose components are man-portable, so access to drill sites is by path instead of road to minimise forest clearing and associated erosion risk. The advantage to this approach is illustrated by the fact that only 50 m^3 of hardwood has been cut in access for drill rigs and for associated drill platforms. This is significantly less than the 225 m^3 that the environmental permit for drilling allowed.

Aerial Transport System

Detachable aerial cable systems have been designed, built and installed in the Berlin Project to transport personnel and equipment around the project in order to reduce the number of paths that are needed between drill platforms.

Control of Soil Erosion

Steep sections of the access paths to the drill sites and drainage channels that control runoff from the paths are lined with bamboo, purchased from local farmers, in order to minimise soil erosion.

Waste Water Management

Water quality is monitored on an ongoing basis, with samples taken before the rig arrives on site, during drilling and after drilling at each site. Drilling water is recycled, then filtered to remove pollutants such as grease, mineral waste and additives, before being returned to the stream or source from which it came.

Tree Nursery

An inventory of native plants in the Berlin Project area has been catalogued and these species are grown in a nursery on company-owned land for replanting of exploration sites as well as in other areas recommended for reforestation. Four vegetation types are being propagated: pioneer or dynamic species, forage species, native species and endangered species.

Another goal of the nursery is the conservation of endangered species that are native to the zone. The Company is working with the BioDiversa Colombia Foundation, an organization developing conservation and educational projects focused on local communities, which include outlining commitments with respect to training, rescue of species, transport and maintenance of the plantlets in the nursery.

Restoration of Exploration Areas

For restoration purposes, an inventory of the vegetation is made prior to the start of exploration activities. Once exploration work has been completed, the affected areas are restored as quickly as possible to their initial conditions. Trench sites and drill platforms are replanted with native species typical of the surrounding vegetation.

Improving Potable Water Quality

Land has been purchased in the headwaters of the stream that supplies water to the Berlin village. The removal of livestock and reforestation with selected native species that regulate the water cycle has led to a marked increase in water quality.

Materials and waste management

Waste from the exploration camp and drill sites is appropriately collected, separated, stored and disposed of. Organic waste is composted and used in the nursery, certain materials and batteries are recycled and non-recyclable waste is sent to the municipality's prescribed waste facility. Lubricants and additives used in drilling are biodegradable.

Suppliers of Goods and Services

A strict control of our suppliers is made to verify their compliance with the environmental legislation, standards and policies in force in Colombia. This includes ongoing audit of the practices of the drilling company, such as for the adequate use of protective gear by operators and auxiliary personnel, the use of biodegradable additives in the drilling, and adequate storage and handling of fuels, lubricants and different types of residues.

Training of Field Personnel

Personnel attend workshops on a regular basis in order to raise awareness and to minimise the impact on the natural resources, with special emphasis on the local flora and fauna.

20.2 Social and Community Engagement

20.2.1 Socio-economic Baseline Study

With the support of the town of Berlin and neighbouring areas, U3O8 Corp. staff undertook a socio-economic baseline study to identify the primary needs of the community to identify practical ways to work jointly with the local people to improve quality of life. Detailed data were obtained by observation and interviews with residents.

20.2.2 Social Initiatives

The areas around the Berlin Project have an agricultural subsistence economy. The socio-economic review resulted in the implementation of initiatives in the areas of nutrition, health, education, employment and encouraging agri-business development that could help achieve sustainable and long term benefits for the local community. These initiatives include:

Central vegetable garden

A community orchard has been established on company-owned land to promote healthy diet, self-sufficiency, training on nutritious food selection and preparation and cooking of the produce. At present, an area of 0.5 Ha is under cultivation, producing vegetables such as corn, beans, tomatoes, peppers, cabbage, spinach, onions, chard, cucumber and herbs using organic methods. Women from the community work in the garden on a rotating basis and share in the vegetables produced. They gain experience in sowing, fertilising and plant care, applying pest control measures (mainly by bio-control) and crop rotation. The community is encouraged to create their own home vegetable gardens and help is provided with set-up and maintenance.

Vermiculture and aquiculture

Residents are being encouraged to produce tilapia for a sustainable source of quality protein, which is also marketable in the region. A serving of tilapia provides about 26 g or 52% of the daily protein requirement. A vermiculture program uses organic waste from the camp and garden to generate compost. A proportion of the earthworms is fed to the fish and the compost is used on the vegetable garden.

Health initiatives

Public health initiatives include providing transport for a nurse to reach outlying communities, improving the quality of drinking water and outreach programs and education on the control of tropical-related health issues/diseases. Fitness classes for seniors and personal development workshops have also been implemented as well as sponsorship of local sports teams and tournaments.

Education initiatives

On education, U3O8 Corp. has worked with the community and government officials to improve school facilities and set up adult classes for grades 6-9. Nurseries have been started to support working families in the Berlin area. The Company is also working with the education authorities to hire teachers and trained staff, encourage ongoing training for teachers and promote Berlin as a pilot training centre for the region.

Members of the community and school children review environmental initiatives undertaken by U3O8 Corp., such as reforestation and erosion control, as a means of education and exposure to environmental stewardship. School children have helped reforest some areas and planted seeds in the indigenous tree nursery.

Workshops, information sessions and briefings with residents, community leaders and authorities as well as site visits and school trips provide information on U3O8 Corp's exploration activities, the Berlin Project and on uranium and associated radioactivity.

Agri-business development initiatives

The Company supports development of agri-businesses by facilitating access to instructors and courses from government learning institutes (e.g. Colombia's National Learning Service or Agricultural Institute). The Company is working with the government and private enterprise agronomists to assist with crop selection and production techniques suitable for the area to encourage development of sustainable small ventures and co-operatives. Areas of interest include the production of avocado or cacao for chocolate, which were historically grown in the region, as well as testing new crops such as teas, anti–oxidant berries, bamboo for furniture and dairy products. Help is also provided to residents in writing business plans to access government grants for community projects.

Local employment

The majority of U3O8 Corp's exploration teams – from geologists, technicians, field staff to administrative and camp support – come from within the country in which the project is located including nearby communities. In Colombia, our team is entirely Colombian, supplemented by people from the local villages.

20.3 Health and Safety

Sound policies and precautionary measures have been established to create a safe and healthy workplace that helps protect our personnel against potential adverse effects of radiation related to uranium and associated radionuclides. These measures include protective gear, strict personal hygiene standards and radiation monitoring devices for all employees.

21 CAPITAL AND OPERATING COSTS

21.1 Approach

Capital and operating costs have been estimated in accordance with standard industry practices for this level of study. The one exception to these practices is the estimate of cost credit from power generated from heat produced by the sulphuric acid plant. Reasonable estimates have been made by comparison with plants of similar size elsewhere and the cost of power contribution to the processing plant used in this study is considered conservative. Operating costs estimates are based on quoted reagent and international transport costs and estimates of local transport costs. Estimated mining costs are based on budget costs from contractors and suppliers as well as the mining consultant's experience from comparable mining projects elsewhere in South America. Capital and operating cost estimates are considered accurate within ±35%.

Capital and operating costs have been estimated on the basis of mining and processing nominal 1,430 tpd of mineralised material for an annual plant throughput of 500,000 t that at an average recovery of 96.1% would produce 1.17 Mlb of U_3O_8 per year. This production rate would be achieved in Year 2, after a ramp-up period in Year 1 in which throughput is planned at 250,000 t for production of approximately 0.58 Mlb of U_3O_8. Year 15 sees the processing of the final 210,000 t of the conceptually mineable resource for production of an estimated 0.49 Mlb of U_3O_8. Total uranium production over the 15 year mine life is 16.3 Mlb.

Capital and operating cost estimates have been made for two alternative processing scenarios:

- A base case, which uses acetic acid as a means of dissolving calcite, which constitutes over 50% of the mineral content of the material. Beneficiation with acetic acid reduces the mass of material entering the acidic ferric iron leach process at approximately double the grade of the ROM material and less sulphuric acid is used in the leach process. Due to the mass reduction in material being leached, smaller equipment is required for processing beyond the beneficiation step. The process of reconstituting spent acetic acid uses sulphuric acid and produces gypsum that is a potentially saleable product that adds to revenue; and

- An alternative case in which the mineralised material is not beneficiated before undergoing leaching with acidic ferric iron leach, termed the "non-acetic option".

21.2 Capital Cost Estimates

21.2.1 Summary

The pre-production capital is estimated at $407.0 million. Sustaining capital is estimated as $42.5 million including mine closure costs. The Life-of-Mine capital is estimated as $449.5 million. A contingency of $40.6 million was used for the capital cost estimate (Table 21-1).

Table 21-1: Capital cost estimate for the Berlin Project – base-case with acetic acid beneficiation ($ millions)

	Initial	Sustaining	Total
SUMMARY (Details below)			
Mining	62.5	11.4	73.9
Plant	182.2	12.5	194.7
Mine dewatering, environmental & closure	10.0	14.9	24.9
Infrastructure	42.8	3.7	46.5
Indirect costs (includes contingency)	109.5	-	109.5
TOTAL	**407.0**	**42.5**	**449.5**
Mining			
Exploration decline – 760 m (4.6 x 5.0 m)	3.8	-	3.8
Horizontal/Ramp development	6.1	-	6.1
Underground crushing and milling chamber	7.7	-	7.7
Underground loading and crushing section	6.8	-	6.8
Vertical development/Raise boring work	1.6	-	1.6
Secondary mine openings	2.3	-	2.3
Underground mobile equipment	27.8	9.3	37.1
Ventilation equipment	6.4	2.1	8.5
SUB TOTAL	62.5	11.4	73.9
Plant			
Crushing	1.8	0.4	2.2
Stockpile	1.9	0.4	2.3
Grinding	7.8	1.6	9.4
Acetic acid leaching	6.9	1.4	8.3
Acetic acid regeneration	10.5	2.1	12.6
Acidic ferric iron leaching	6.8	1.4	8.2
Ion exchange	4.4	0.9	5.3
Mo recovery	0.8	0.2	1.0
U recovery	13.6	2.7	16.3
V recovery	0.8	0.2	1.0
SX – Phosphates & REE	3.1	0.6	3.7
SX –Ni & Zn	1.7	0.3	2.0
REE	0.5	0.1	0.6
Ni & Zn recovery	0.3	0.1	0.4
Earthwork, Services, Reagents and Buildings	120.6	-	120.6
Waste treatment plants, Liquid	0.7	0.1	0.8
SUB TOTAL	182.2	12.5	194.7

	Initial	Sustaining	Total
Mining, Dewatering & Environmental			
Mine dewatering	6.4	0.2	6.6
Environmental	3.6	5.4	9.0
Mine Closure	-	9.3	9.3
SUB TOTAL	10.0	14.9	24.9
Infrastructure			
Land acquisition	3.0	-	3.0
Site preparation	4.3	-	4.3
Power line & main substation (12 MVA)	8.3	-	8.3
Emergency power generation	3.0	0.2	3.2
Overall site communication and control system	1.9	0.4	2.3
Administrative buildings	4.5	0.1	4.6
Utilities and facilities buildings	2.3	-	2.3
Workshop and support buildings	2.5	-	2.5
Plan mobile fleet	1.5	2.9	4.4
Earthwork and civil (concrete)	6.5	-	6.5
Plant site dam	2.5	0.1	2.6
Water treatment and decontamination	1.5	-	1.5
Other	1.0	-	1.0
SUB TOTAL	42.8	3.7	46.5
Indirect Costs			
Mine construction (surface installations)	1.1	-	1.1
EPCM/Owner costs	54.1	-	54.1
First fills/Reagents/Spares	8.2	-	8.2
Temporary facilities	5.5	-	5.5
Contingency	40.6	-	40.6
SUB TOTAL	109.5	-	109.5
TOTAL	**407.0**	**42.5**	**449.5**
Working Capital	15.6	(15.6)	-

21.2.2 Mine and Plant

21.2.2.1 Sustaining Capital Cost Estimate

All costs are estimated for owner-operation. Sustaining capital has been estimated at $42.5 million (Table 21-1). Key sustaining capital has been estimated based on Tenova and P&K practices, which are:

- Process Plant Equipment - 2% per annum of the Capital Equipment Costs commencing from the 5th year;
- Buildings - 1% per annum of the Building Costs every 5 years;
- Mobile Equipment - all light vehicles replaced at year 5, with all other mobile equipment after 10 years;
- Pipeline - 1% per annum of the Capital Pipeline Cost commencing from the 5th Year; and
- Tailing facilities – 2 % per annum from 5th year.

21.2.2.2 Direct Capital Cost of Underground Mine

The direct initial capital costs include all new equipment, new materials, and installation for all permanent facilities associated with:

- Underground crushing and milling, stockpiling and other processing facilities;
- Process building and earthwork, civil and drainage;
- Infrastructure roads and site preparation;
- Power supply and distribution;
- Pre-production development of the underground mine;
- Tailings storage;
- Warehousing;
- Administration;
- Truck shop;
- Yard services and other utilities;
- Control and communications systems;
- Plant mobile equipment;
- Fuel storage; and
- Explosives storage.

21.2.2.3 Underground Mine Development Capital Cost Estimate

A considerable amount of development work is planned for two pre-production years (Years -2 and -1) to provide a degree of flexibility in terms of access, which should facilitate mine production scheduling. The total initial capital for the mine is estimated as $62.5 million with mobile equipment as the main component at $27.8 million (Table 21-1).

The construction of a proposed exploration decline at a cost of $3.8 million was included as a mine cost for the purposes of this study (Table 21-1). The estimated capital required for the two years of pre-production development of horizontal and ramp development is $6.1 million. Horizontal development includes the access and production ramps ($3.8 million) and the excavations required to access the production drifts and other opening for underground infrastructure. This plan would provide enough material to start production. The capital required for an underground tailings storage facility is estimated at $1.2 million. Additional drift development would be completed at a capital cost of $1.1 million.

A crusher – mill complex is planned to be located in an underground chamber located adjacent to the main decline at a cost of $7.7 million. This location will reduce airborne emissions at surface. The underground loading and crushing section will be completed at a cost of $6.8 million (Table 21-1).

Pre-production vertical development and raise boring includes the exit ($0.4 million) and exhaust ($0.4 million) shafts and ventilation raises ($0.5 million) necessary to prepare the first stope for production. A storage facility for mineralised material is included at a cost of $0.3 million.

21.2.2.4 Secondary Mine Openings

Secondary mine openings include truck bypass ($0.8 million), side excavations ($0.8 million), underground electrical ($0.3 million) for one room and a further $0.3 million for a second room, and $0.1 million for an explosives magazine.

21.2.2.5 Underground Mining Equipment Capital Cost

The initial capital cost of the fleet of mine equipment required for underground mining is approximately $27.8 million (Table 21-2). It is expected that the raise-borer is likely to retain approximately 25% of its original value at the end of the mine life.

Table 21-2: Summary of initial capital cost estimate for underground mining equipment

Description	Quantity	Cost $ millions
Mine truck (28 t) 300 HP	5	3.9
LHD 3 m^3 (Development) 139 HP	4	2.4
LHD 5 m^3 (Production) 231 HP	4	3.5
Jumbo Twin Boom (88 - 104 kW)	3	2.7
Jumbo Single Boom (24.5 kW)	3	2.3
Roof bolter 57 HP	3	2.5
Shotcrete truck 210 HP	3	1.5
Scissor lift truck 82 HP	3	0.9
Service transport truck	3	0.9
Jackleg and stopper drill	10	0.1
Long-hole drill 40 HP	4	1.5
Raise boring machine 2.5 - 3.7 m diam. 350 HP	1	4.7
Surface FEL – Cat 966 type 180 HP	2	0.4
Grader – Underground 138 HP	1	0.3
Pick-up 150 HP	4	0.2
Total		**27.8**

21.2.2.6 Underground Ventilation Equipment Capital Cost

The cost of ventilation equipment, summarized in Table 21-3, is estimated at approximately $6.4 million.

Table 21-3: Capital cost estimate of ventilation equipment

Description	Quantity	Cost $ millions
Main fan (250 HP) and accessories	3	1.0
Underground fans (50 HP)	20	2.6
Compressors (300 HP) with installation	2	0.2
Pumps 300 HP	4	0.3
Pumps 150 HP	4	0.3
Install hardware and piping		1.5
Install ventilation system		0.5
Total		**6.4**

21.2.3 Mineral Processing Facility

21.2.3.1 Basis of Capital Cost Estimate of the Plant

The capital costs were developed from first principles with inputs based on budget consumables prices, feedback from contractors, experience and cost estimation services. The basis of the cost estimate, for both acetic and non-acetic options, was based on throughput of mineralised material of 1 Mtpa and extrapolated for 0.5 Mtpa throughput (Table 21-1). Each cost estimate itemised below includes associated concrete, structural, platework, piping and electrical costs are included as a percentage of the mechanical equipment costs:

- Crushing: Initial capital costs of $1.8 million include a front end loader, jaw crusher, conveyer for feed, dust suppression, platework, piping and electrical and instrumentation;

- Stockpile: Stockpile costs of $1.9 million relate to excavation of a chamber for stockpiling of mineralised material;
- Grinding: Grinding costs of $7.8 million relate to an apron feeder, SAG mill and stockpile area dust scrubber;
- Acetic acid leaching: Acetic acid leaching costs of $6.9 million relate to agitators, thickeners, leach tanks and wash tanks;
- Acetic acid regeneration: Acetic acid regeneration costs of $10.5 million relate to agitators, thickeners, filters, pumps, and feed, wash and filtrate tanks;
- Acidic ferric iron leaching: Acidic ferric iron leaching costs of $6.8 million relate to agitators, thickeners, pumps and tanks for leaching, iron reduction and PLS storage;
- IX: IX costs of $4.4 million relate to exchange columns, a maintenance crane and pumps;
- Molybdenum Recovery: Molybdenum recovery costs of $0.8 million relate to a precipitation tank and agitator, thickener and waste water storage tank with agitator;
- Uranium Recovery: Uranium recovery costs of $13.6 million relate to a calciner and final product treatment facility, precipitation tank and agitator, thickener, waste water storage tank with agitator and a transfer conveyor;
- Vanadium Recovery: Vanadium recovery costs of $0.8 million relate to a precipitation tank, a solid/liquid separator, an elution tank, a precipitation tank and thickener;
- SX – Phosphates and REEs: Costs of $3.1 million relate to supply tank agitators, a water storage tank, a phosphoric acid upgrade evaporator, a PLS supply tank and a phosphoric acid tank;
- SX – Nickel and Zinc: Costs of $1.7 million relate to an organic supply tank agitator, a mixer tank, mixer settlers, a mixed organic supply tank and a water storage tank;
- REE Recovery: REE recovery costs of $0.5 million relate to agitators and a precipitation tank, a centrifuge filtrate tank, a rare earth solid/liquid separator, a precipitation tank and a centrifuge filtrate tank;
- Nickel and Zinc Recovery: Nickel and zinc recovery costs of $0.3 million relate to agitators and carbonate precipitation tanks, a carbonate solid/liquid separator, a precipitation tank and a waste water storage tank;
- Earthwork, Services, Reagents and Buildings: Costs of $120.6 million relate mostly to a sulphuric acid plant ($49.0 million), a power plant ($24.5 million), piping ($16.9 million), concrete work ($15.0 million), and structural work ($9.6 million); and
- Waste Treatment Plants: Costs of $0.7 million relate to a waste treatment agitator, filter feed, pump and thickener mechanism, with piping and structural work.

21.2.4 Mine Dewatering

Mine dewatering would be planned to commence prior to and in conjunction with pre-development of the mine and is estimated to cost $6.4 million. The estimate includes the cost of drilling 10 dewatering wells of 250 m each ($3.2 million), lining and preparation of well filters, 12 vertical pumps ($2.7 million) and all necessary piping and accessories ($0.5 million) for the system.

21.2.5 Capital Costs Associated with Environmental Work

Initial capital costs associated with environmental studies and work is estimated at $3.6 million, including mine and mill. The preliminary environmental assessment is budgeted at $0.6 million. Pre-production monitoring is expected to cost $1.8 million and Environmental Studies are expected to cost a further $1.2 million. The estimate assumed that environmental monitoring will commence 2 years prior to production.

21.2.6 Capital Cost Estimate of Ongoing Environmental and Mine Closure

Environmental and closure costs for the end of the mine life are included in sustaining capital at a cost of $14.7 million. These costs include a series of expenditures that are assumed to commence three years before production shutdown. It is assumed that dismantling of all surface installations and associated cleanup, reclamation of tailings facilities and any surface construction, would continue six months after mining ceases. Thereafter, a monitoring period of five years is anticipated.

21.2.7 Infrastructure

The capital cost of required infrastructure is estimated to be $42.8 million. This cost includes installations for electricity from the high tension power grid into which the La Miel hydroelectric dam feeds, required to augment power generated on site from the sulphuric acid plant. It also includes site communications and administrative building costs.

Earthworks ($6.5 million) and plant site dam costs ($2.5 million) are low as the tailings are expected to be stored at surface until dry and then to be returned underground as backfill.

21.2.8 Indirect and Owner Costs

The total indirect and owner's capital costs are estimated at $109.5 million. Engineering and procurement, construction management, and owner's costs comprise $54.1 million. The $40.6 million of contingency costs is approximately 10% of the total initial capital expenditure. Indirect costs include the following:

- Temporary construction services including some construction equipment;
- Freight;
- Vendor representatives;
- First fills and capital spares;
- Engineering and procurement (EP);
- Construction management (CM);
- Services (including travel expenses);

- QA;
- Surveying;
- Owner's costs; and
- Start-up and commissioning allowance.

21.3 Operating Costs

21.3.1 Summary

Operating cost estimates include a contingency based on 10% of operating costs. Revenue-based royalties were not subject to a contingency adjustment. The operating contingency is independent of the capital spending contingency. On a nominal annual throughput of 500,000 t per year the expected operating cost is $116.6 million (Table 21-4).

The study was done on a pre-income tax basis. Colombian income taxes are 33% of taxable income. Revenue-based royalties assume compliance with contractual or legal agreements or statutes as outlined in Section 4.

Operating costs for the Berlin Project are expected to average $233 /t of mineralized material including royalties and a 10% contingency (Table 21-4). Note that mine operating costs are estimated at an annual nominal production rate of 500,000 tpa of mineralised material and 250,000 tpa of waste and are expressed as a cost per tonne of mineralised material.

Table 21-4: Summary of operating cost estimated for the base-case scenario of acetic acid beneficiation expressed on a per tonne of mineralised material basis

Items	Cost ($ millions per yr)	Cost per tonne
Revenue-based royalties	$ 9.4	$18
Mining	27.3	55
Processing	65.9	132
G&A	1.8	4
De watering	2.4	5
Contingency	9.8	19
TOTAL	**$116.6**	**$233**

21.3.2 Personnel Cost Estimate

Personnel for the Berlin Project are budgeted at $10.3 million, divided between General and Administration ($0.7 million), Mining ($4.7 million) and Plant ($4.9 million) (Table 21-5).

Table 21-5: Estimate of personnel costs

Description	#	Cost $ millions	Cost $ per tonne
General and Administration	21	0.7	1.45
Mining	130	4.7	9.45
Plant			
Service and infrastructure	66	1.5	
Process	81	2.4	
Maintenance and engineering	35	1.0	
	182	4.9	9.62
TOTAL	**333**	**10.3**	**20.52**

21.3.3 Mine

21.3.3.1 Underground Mine Operating Cost Estimate

Mine operating costs are estimated for the production of 500,000 tpa of mineralised material and approximately 250,000 tpa of waste. Operating cost estimates for both waste and mineralised material are expressed on a per tonne of mineralised material basis. Mine operating costs are estimated to be $54.69 /t of mineralised material (Table 21-6).

Table 21-6: Estimate of mine operating costs expressed on a per tonne of mineralised material basis for the Berlin Project

Description	$ millions	$ per tonne
Mine Costs		
Labour	$4.7	$9.45
Development	5.3	10.50
Stoping	4.2	8.32
Backfill (paste)	7.3	14.56
Truck haulage	2.4	4.85
Power	1.6	3.26
Ventilation	1.3	2.50
Mine drainage	0.6	1.25
Total	**$27.3**	**$54.69**

21.3.4 Processing Facility

21.3.4.1 Basis of Processing Facility Operating Cost Estimate

Annual plant operating costs of $65.9 million mostly comprise reagents ($42.0 million), consumables ($6.2 million), maintenance ($9.2 million) and personnel ($4.9 million). Power consumption costs of $11.4 million annually are significantly offset by a co-generation electrical plant which produces a consumption credit of $8.6 million annually, based on capture and conversion of heat generated from the sulphuric acid plant. Other annual milling costs include site vehicle maintenance ($0.1 million) and laboratory analytical costs ($0.7 million). Processing facility costs are estimated to be $131.72 /t of mineralised material (Table 21-7).

Table 21-7: Estimate of mine operating costs expressed on a per tonne of mineralised material basis for the Berlin Project

Description	$ millions	$ per tonne
Processing Facility Costs		
Reagents	$42.0	$84.01
Consumables	6.2	12.48
Maintenance	9.2	18.29
Personnel	4.9	9.62
Power	11.4	22.82
Power credit	(8.6)	(17.19)
Site vehicle maintenance	0.1	0.27
Laboratory analytical	0.7	1.42
Total	**$65.9**	**$131.72**

21.3.4.2 Reagents and Consumables Cost Estimate

Reagent costs are based on international prices, sourced internationally, with the exception of sulphur which is assumed to be sourced in Colombia at international prices. The annual reagent cost is $42 million (Table 21-8).

Table 21-8: List of annual reagent costs used for the operating expense model for the acetic case

Reagent	Annual Cost $ million
Acetic acid	11.3
Ammonium Hydroxide	2.3
Ammonium Carbonate	0.6
Hydrogen peroxide	0.1
Lime	0.3
Pyrolusite	0.3
Scrap iron	0.4
Sulphur	13.0
Diatomaceous earth	0.1
Lime (for H_2SO_4 plant)	0.0
IX - resin	0.5
SX – ShellSol	0.1
SX – TBP	0.0
SX – D2EHPA	0.0
SUB TOTAL	**29.0**
Transportation	13.0
TOTAL REAGENTS	**42.0**

21.3.4.3 Communition Media Cost Estimate

Processing plant operating costs are estimated at $132 /t of mineralised material. Crushing and milling costs were estimated on the basis of a Bond Work Index (BW_i) = 11 kW/t at the quoted power cost of $0.012 /kW-h. The most significant consumable items relate to grinding media ($3.6 million annually), and replacement parts for the crusher and mill liners ($1.9 million annually).

21.3.4.4 Power Cost Estimate

Power costs are based on annual plant needs of 14.3 MW at a cost of $9.4 million. The Phosphoric acid evaporation area consumes almost half of this power draw, while grinding and sulphuric acid leach tank heating are also large power consumers. Other camp areas are expected to draw 2.8 MW power annually at a cost of $2.0 million.

Heat generated by a sulphuric acid plant would power a 14.3 MW co-generation power plant and is expected to generate 10.7 MW electricity annually, offsetting the quantity of external power required for the project. The revenue saved by this on-site power production is estimated to be approximately $8.6 million.

21.3.5 General and Administration

General and administration ("G&A") includes all costs associated with administrative staff, office consumables, vehicles, communication, on-site kitchen and recreation area, facilities other than industrial, site security and general maintenance of the site. G&A costs are estimated at $1.8 million per annum of $3.60 /t of mineralised material (Table 21-9).

Table 21-9: Estimate of general and administration costs

Description	$ millions	$ /t
General and Administration		
Site operating costs	1.3	2.50
Offsite costs (transport, export)	0.5	1.10
TOTAL	**1.8**	**3.60**

21.4 Estimate of Revenue

The base case revenue projections for the Berlin Project are expected to average $429 /t of mineralised material and to generate annual revenue of $214.5 million. The current long-term uranium price of $60 /lb was used. Current uranium spot prices are in the order of about $45 /lb. Prices used for other commodities are based on average prices over the past year (Table 21-10).

Table 21-10: Commodity prices used for the revenue estimates for the Berlin Project

Compound	Price $	Sources:
U_3O_8	60 /lb	(1)
NH_4VO_3	9 /kg	(2)
H_3PO_4 (phosphoric acid)	1 /kg	(3)
$Y(OH)_3$	50 /kg	(4)
$NiCO_3$	9.25 /lb	(5)
$Nd(OH)_3$	50 /kg	(6)
Mo	12 /lb	(7)
Zn	0.89 /lb	(8)
Gypsum	30 /t	(9)

One of the benefits of the acidic ferric iron leach is that it generates a PLS from which metal salts are readily produced. Metal salts tend to command a price above that of the metal itself. The price of these salts varies widely with purity. In all cases, the lowest available price was used. Yttrium and neodymium hydroxide prices are not freely available and generally must be negotiated in an off-take agreement. However, the guidance used in pricing the hydroxides is that contracts are typically priced at a 15-25% discount to the price of the oxide, to allow for the additional processing that's required to produce the oxide from the hydroxide. For example, neodymium oxide averages approximately $120 /kg over the past year. Applying a discount of 25% to that price would give a guidance price for the hydroxide of approximately $95 /kg. However, a price of $50 /kg is used in the financial model to build in more conservatism. Similarly for yttrium, the oxide price averaged approximately $95 /kg over the last year. Applying a 25% discount generates a price of approximately $70 /kg and the financial model uses a price of $50 /kg. Where reliable pricing for the salt was not available, the price of the metal was used as a conservative estimate, e.g. Mo, Zn.

1. U_3O_8 – UxC Consulting, Tradetech
2. NH_4VO_3 – www.alibaba.com
3. H_3PO_4 (phosphoric acid) – www.ecvv.com
4. $Y(OH)_3$ – http://trade.e-to-china.com, www.asiametal.com, www.metal-pages.ca
5. $NiCO_3$ – www.alibaba.com, www.strem.com, www.shepchem.com
6. $Nd(OH)_3$ –www.sigmaaldrich.com, www.alfa.com, www.asiametal.com,www. metal-pages.ca
7. MO – www.strem.com
8. Zn – www.alibaba.com
9. Gypsum – http://minerals.usgs.gov

Revenue is dependent on metal content and overall recovery rates. These values for the metals in the study are shown in Table 21-11.

Table 21-11: Recovery rate of final products

Compounds	Yearly production (t)	Alternative compounds for sale	Yearly production (t)	Net recovery (%)
U_3O_8	5,32	U_3O_8	532	96.1
NH_4VO_3	2,137	V_2O_5	1,662	66.3
H_3PO_4	63,485	P_2O_5	45,988	98.9
$Y(OH)_3$	257	Y_2O_3	207	83.5
$NiCO_3$	1,482	Ni	733	63.7
$Nd(OH)_3$	33	Nd_2O_3	28	48.6
$CaMoO_4$	284	Mo	136	45.4
$ZnCO_3$	2,943	Zn	1,534	95.9
Gypsum	388,270	Gypsum	388,270	
		$MnCO_3$	1,807	

22 ECONOMIC ANALYSIS

This study is preliminary in nature and it includes Inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorised as mineral reserves. There is no certainty that the results on which this study is based will be realised. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

Tonnage and grade of mineralised material used in this discounted cash flow ("DCF") analysis is reported in Section 14. Approximately 7% of the material in the DCF analysis is from the Indicated category while 93% of the material is from the Inferred category. The study assumes that 20% of the resource would not be mined since it would be left in place to provide structural support for the underground mine.

The study uses a mining rate of 500,000 tpa because the balance between revenue generated and capital cost requirements maximised the rate of return for the available resource. The contemplated plant and mine plan could easily be scaled up in the event that more resources are defined.

This model generates revenue of $3.0 billion against operating costs (with contingency) of $1.6 billion over the 15 year mine life (Table 22-1).

Table 22-1: Summary of cash flow model for base case of beneficiation with acetic acid

	P-2	P-1	P1	P2	P3	P4	P5	P6	P7	P8	P9	P10	P11	P12	P13	P14	P15	TOTAL
MINE PLAN																		
Resource (open – 000's tonnes)	6,960	6,960	6,960	6,710	6,210	5,710	5,210	4,710	4,210	3,710	3,210	2,710	2,210	1,710	1,210	710	210	6,960
Mined (000's tonnes	-	-	250	500	500	500	500	500	500	500	500	500	500	500	500	500	210	6,960
Resource (close – 000's tonnes)	6,960	6,960	6,710	6,210	5,710	5,210	4,710	4,210	3,710	3,210	2,710	2,210	1,710	1,210	710	210	-	-

	P-2	P-1	P1	P2	P3	P4	P5	P6	P7	P8	P9	P10	P11	P12	P13	P14	P15	TOTAL
CASH FLOW																		
Revenue			107.3	214.5	214.5	214.5	214.5	214.5	214.5	214.5	214.5	214.5	214.5	214.5	214.5	214.5	90.1	2,986.5
Operating Costs:																		
Revenue-based royalties			4.7	9.4	9.4	9.4	9.4	9.4	9.4	9.4	9.4	9.4	9.4	9.4	9.4	9.4	3.9	130.7
Mining			13.7	27.3	27.3	27.3	27.3	27.3	27.3	27.3	27.3	27.3	27.3	27.3	27.4	27.3	11.5	380.6
Milling			32.9	65.9	65.9	65.9	65.9	65.9	65.9	65.9	65.9	65.9	65.9	65.9	66.6	66.6	27.7	916.8
G&A			0.9	1.8	1.8	1.8	1.8	1.8	1.8	1.8	1.8	1.8	1.8	1.8	1.8	1.8	0.8	25.1
Dewatering			1.2	2.4	2.4	2.4	2.4	2.4	2.4	2.4	2.4	2.4	2.4	2.4	2.4	2.4	1.0	33.7
Contingency			4.9	9.7	9.7	9.7	9.7	9.7	9.7	9.7	9.7	9.7	9.7	9.7	9.8	9.8	4.1	135.6
			58.3	116.6	116.6	116.6	116.6	116.6	116.6	116.6	116.6	116.6	116.6	116.6	116.6	116.6	49.0	1,622.5
			49.0	98.0	98.0	98.0	98.0	98.0	98.0	98.0	98.0	98.0	98.0	98.0	98.0	98.0	41.2	1,364.0
CAPEX	227.5	156.0	28.7	11.0	0.4	0.4	7.7	3.0	3.0	1.6	1.6	3.1	7.3	1.6	2.5	5.3	(10.8)	449.5

22.1 Valuation

The DCF valuation method was used in valuing the Berlin Project. The Cash Flow Approach relies on the "value in use" principle and requires determination of the present value of future cash flows over the useful life of the asset. The asset is valued using the free cash flow capitalisation, i.e. the DCF methodology.

The DCF model is aimed at assessing the economic feasibility of mining and processing the uranium and other metals contained in the mineral resource.

The DCF was calculated based on a range of uranium prices from \$40 /lb to \$80 /lb of U_3O_8 in \$10 increments. The DCF was calculated for each uranium price for a variety of discount rates from 0% to 15% in 5% increments. The resulting DCF matrix for each uranium price and discount rate is shown in Table 22-2.

Table 22-2: DCF matrix for various uranium prices and discount rates

DCF ($ millions, except for uranium prices)	Uranium Price				
	$40	$50	$60	$70	$80
0%	595	755	915	1,074	1,234
5%	246	346	447	546	647
10%	59	126	192	258	325
15%	(45)	1	47	94	140
IRR	12%	15%	17%	20%	22%
Payback (years)	6.3	5.5	4.9	4.4	4.0

22.2 Sensitivity Analysis

In the base-case scenario in which acetic acid is used for beneficiation of the mineralised material, uranium comprises 33% of the Berlin Project revenue while phosphates comprise 29%. Nickel (14%), yttrium (6%) and vanadium (9%) are also significant revenue generating metals (Table 22-3).

Table 22-3: Summary of revenue generated by each element extracted from mineralised material from Berlin. Figures are shown for the "Acetic" process in which acetic acid is used to beneficiate the mineralised material and also for the "Non-acetic" process in which beneficiation is not used in mineral processing

Element	Acetic		Non-acetic	
	$ million	%	$ million	%
U_3O_8	$ 977	33	$ 977	35
NH_4VO_3	268	9	268	9
H_3PO_4 (phosphoric acid)	866	29	866	31
$Y_2(OH)_3$	179	6	179	6
$NiCO_3$	420	14	420	15
$Nd(OH)_3$	23	1	23	1
Mo_2O	50	2	50	2
Zn	42	1	42	1
Gypsum	162	5	-	-
TOTAL	**$ 2,986**	**100**	**$ 2,824**	**100**

Sensitivity analysis scenarios for DCF were considered for changes in revenue, operating costs and capital spending. A sensitivity analysis was performed on the base-case numbers with variances of -20%, -10%, 0%, +10%, and +20% on each of revenue, operating costs and capital expenditure. The contingency in capital spending and in operating costs was included in the sensitivity analysis.

Revenue could change from the base case scenario from any combination of a change payable metal (different grades or recoveries from base case assumptions) and/or a change in prices. Changes in revenue would affect the operating costs and royalties would also be affected.

The DCF sensitivity matrix to changing the revenue assumptions is shown in Table 22-4.

Table 22-4: DCF sensitivity matrix for changes in revenue

DCF ($ millions)	-20%	-10%	0%	+10%	+20%
0%	321	618	915	1,211	1,508
5%	75	261	447	633	819
10%	(55)	68	192	315	439
15%	(125)	(39)	47	133	219
IRR	7%	13%	17%	22%	25%
Payback (years)	8.4	6.2	4.9	4.1	3.5

A change in operating costs (revenue-based costs not affected for this analysis) would result in different cash flow streams to the project. Note that the base-case operating expense includes a 10% contingency. Operating costs could change, for example, as a result of efficiencies in operating methods or metallurgical processes, or improved costs for reagents and supplies.

The DCF sensitivity matrix to changing operating cost assumptions is shown in Table 22-5.

Table 22-5: DCF sensitivity matrix for changes in operating costs

DCF ($ millions)	+20%	+10%	0%	-10%	-20%
0%	616	766	915	1,064	1,213
5%	259	353	447	540	634
10%	68	130	192	254	316
15%	(39)	4	47	91	134
IRR	13%	15%	17%	20%	22%
Payback (years)	6.1	5.5	4.9	4.4	4.1

The base-case capital spending includes a $41 million contingency, being 10% of the initial capital requirement. Capital spending could increase due to shortages in availability of materials or construction workers. Capital spending could decrease as more economic alternatives are found and implemented for the extraction and processing of metals and phosphate from the mineralised material, or the contingency might prove to be overly conservative.

The DCF sensitivity matrix to changes in capital expenditure assumptions is shown in Table 22-6.

Table 22-6: DCF sensitivity matrix for changes in capital expenditures

DCF ($ millions)	+20%	+10%	0%	-10%	-20%
0%	825	870	915	960	1,005
5%	365	406	447	487	528
10%	117	154	192	230	267
15%	(22)	12	47	82	117
IRR	14%	16%	17%	20%	22%
Payback (years)	5.8	5.3	4.9	4.4	4.0

22.3 Non-Acetic Mineral Processing Option

22.3.1 Capital Cost Estimates

An example of the effect that alternative mineral processing methods may have on the economics of the Berlin Project is provided by a processing route that excludes beneficiation of the mineralised material by acetic acid. In the non-acetic processing option, the milled ROM material undergoes acidic ferric iron leach directly without beneficiation by acetic acid. Capital costs are affected in the following ways:

- They are reduced because the acetic acid storage and regeneration components of the conceptual plant are eliminated;
- They increase to some extent because the remainder of the plant needs to be enlarged to handle the higher volumes of mineralised material. The alternative acetic acid beneficiation dissolves over half of the mineralised material, thereby reducing the volume of material requiring downstream processing; and

- A tailings facility is required for the long-term storage of tailings that are in excess of the volume required for backfill in the underground mine.

Capital cost estimates fall from $449.5 million in the acetic acid beneficiation option to $441.0 million in the non-acetic option (Table 22-7). A contingency of $40.6 million is maintained for both cases.

Table 22-7: Summary of estimated capital costs of processing of mineralised material from Berlin with acetic acid beneficiation compared with no beneficiation ("non-acetic")

	Acetic	Non-acetic
SUMMARY (Details below)		
Mining	73.9	73.9
Plant	194.7	177.4
Mine dewatering, environmental & closure	24.9	24.9
Infrastructure	46.5	59.0
Indirect costs (includes contingency)	109.5	105.8
TOTAL	**449.5**	**441.0**
Mining		
Unchanged items	73.9	73.9
Plant		
Unchanged items	38.7	38.7
Acetic acid leaching	8.3	-
Acetic acid regeneration	12.6	-
Acidic ferric iron leaching	8.2	10.4
SX – Phosphates & REE	3.7	3.8
SX –Ni & Zn	2.0	2.3
REE	0.6	1.1
Earthwork, Services, Reagents and Buildings	120.6	121.1
SUB TOTAL	194.7	177.4
Mine dewatering, environmental & closure		
Unchanged items	24.9	24.9
Infrastructure		
Unchanged items	43.9	43.9
Plant site dam	2.6	15.1
SUB TOTAL	46.5	59.0

	Acetic	Non-acetic
Indirect Costs		
Unchanged items	41.7	41.7
EPCM/Owner costs	54.1	51.5
First fills/Reagents/Spares	8.2	8.4
Temporary facilities	5.5	4.2
SUB TOTAL	109.5	105.8
TOTAL	**449.5**	**441.0**

22.3.2 Operating Cost Estimates

The non-acetic processing route would also affect revenue and operating costs in the following way (Table 22-8):

- Revenue would be lost from the potential by-product gypsum in the non-acetic process; and
- The cost of reagents is reduced because acetic acid is not required and a component of the sulphuric acid balance is no longer required for regeneration of spent acetic acid

Table 22-8: Summary of the annual reagent cost differences between acetic and non-acetic mineral processing routes

Reagents	Acetic $ million	Non-acetic $ million
Acetic acid	11.3	-
Ammonium Hydroxide	2.3	2.3
Ammonium Carbonate	0.6	0.6
Hydrogen peroxide	0.1	0.1
Lime	0.3	0.3
Pyrolusite	0.3	0.3
Scrap iron	0.4	0.4
Sulphur	13.0	14.1
Diatomaceous earth	0.1	0.1
Lime (for H_2SO_4 plant)	0.0	0.0
IX - resin	0.5	0.5
SX – ShellSol	0.1	0.1
SX – TBP	0.0	0.0
SX – D2EHPA	0.0	0.0
SUB TOTAL	29.0	18.7
Transportation	13.0	9.7
TOTAL REAGENTS	**42.0**	**28.4**

At a uranium price of $60 /lb, revenue per tonne of mineralised material would decrease approximately 5% from $429 at the base-case including acetic acid to $406 using the non-acetic acid alternative as gypsum revenue is removed from the process. Operating costs at a uranium price of $60 /lb would drop approximately 14% from $233 /t of mineralised material beneficiated with acetic acid to $201 /t in the non-acetic alternative.

A DCF matrix for the non-acetic option shows an improved IRR (Table 22-9) over the non-acetic option (Table 22-2). At a uranium price of $60 /lb and a discount rate of 10%, the non-acetic yields a marginally better IRR of 19% in comparison to 17% for the base-case in which acetic acid is used to beneficiate the mineralised material.

Table 22-9: DCF matrix for various uranium prices and discount rates for the non-acetic

DCF ($ million, except uranium prices)	Uranium Price				
	$40	$50	$60	$70	$80
0%	663	822	982	1,142	1,302
5%	291	391	491	591	691
10%	90	157	223	290	356
15%	(21)	24	71	117	163
IRR	14%	16%	19%	21%	23%
Payback (years)	5.9	5.2	4.6	4.2	3.8

22.4 Valuation Summary

The Berlin Project DCF is $192 million at a 10% discount rate for the base-case model in which acetic acid is used to beneficiate the mineralised material. The project is economically robust to changes in revenue, operating cost and capital expenditure.

The non-acetic alternative is modestly more economic than the acetic alternative with a DCF of $223 million at 10% discount rate. The economics of operating cost savings more than offset the gypsum revenue lost by choosing the non-acetic alternative.

Results of this PEA are preliminary and work continues on process optimisation, reducing reagent consumption and finding methods of improving the extraction and recovery of metals that are currently less than optimal. Since cost savings could change the relative merits of the processing alternatives considered in this study, both avenues continue to be explored in an effort to add economic benefit to the project. These preliminary results should not be used to indicate that one production alternative is a preferred method at this time.

23 ADJACENT PROPERTIES

There are no exploration projects or concessions adjacent to or in the immediate vicinity of the Berlin Project.

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24 OTHER RELEVANT DATA AND INFORMATION

24.1 Potential Sources of Reagent Supply

As described under Recovery Methods (Section 17), U3O8 Corp. has established an acidic ferric iron leach process to efficiently extract the various commodities from the mineralised material at Berlin, which uses a combination of ferric iron and sulphuric acid reagents. The Berlin deposit is located near possible sources of pyrite and sulphur within several hundred kilometres of the project, which could be used for on-site production of ferric iron and sulphuric acid for use in the leach process and provide a potentially low-cost, local source of reagents.

Sulphur for generating sulphuric acid could come from the Barrancabermejo oil refinery that is about 300 km downstream from the Berlin Project on Colombia's principal river, the Magdalena. Berlin is located about 60 km west of the Magdalena River, which is navigable by barge. The sulphur mine, El Vinagre, in southern Colombia could be another possible local source.

Local pyrite could be obtained from pyrite-rich tailings at nearby gold and base metal mines located in the Marmato and Cauca gold belts. This may be a mutually beneficial situation for U3O8 Corp. and these mine operators. Pyrite, a potentially problematic waste product for gold producers, could be converted into a stable, environmentally benign iron mineral to generate ferric iron and dilute sulphuric acid for U3O8 Corp's leach process.

Depending on transport costs, pyrite and sulphur sourced near the Berlin Project may be environmentally-friendly options as well as help to reduce reagent costs. U3O8 Corp. is initiating discussions with some of these operations.

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25 INTERPRETATION AND CONCLUSIONS

- This PEA is preliminary in nature. It is based largely on Inferred resources, a category of mineral resources that is considered too speculative geologically to have the economic considerations applied to them that would allow them to be categorised as mineral reserves. There is no certainty that the results of the preliminary economic assessment will be realised.
- Industry standard exploration, resource estimation, mining, process design, construction methods and economic evaluation practices have been employed in the preparation of this PEA on the Berlin deposit.
- In regard to the mineral resource on which this PEA is based:
  - Approximately 93% of the current resource at Berlin is in the Inferred category.
  - Geological information obtained from diamond drilling, supported by industry-standard assay and QAQC procedures, are considered adequate and appropriate for the estimation of the resource on which this PEA is based.
  - Independent data verification was undertaken and no areas of concern were identified. Independent sampling of duplicate core samples by Coffey Mining (2102) yielded results that were within acceptable limits of the original data.
  - Results of exploration conducted adjacent to, and along trend from the initial resource area on which this PEA is based, were consistent and comparable with those of the resource area. Nothing observed in the drill core and assay data from the exploration area was found to be inconsistent with parameters used in the mineral resource estimation undertaken by Coffey Mining (2102), on which this PEA is based.
  - In addition to the elements on which the mineral resource estimate was undertaken by Coffey Mining (2102), subsequent metallurgical test work indicated that both zinc and gypsum may constitute saleable by-products and, hence, Coffey Mining was requested to undertake a resource estimate for calcium and magnesium, from which the calcium content of the mineralised body could be calculated, which led to the mass of gypsum by-product being estimated. A resource was also estimated for zinc.
  - Manganese should be included in the list of elements included in the next mineral resource estimate.
- Drill results and assay data show remarkable continuity of the mineralised unit, not only within the 3 km long initial resource area, but also in the adjacent 3.5 km long area in which exploration drilling was undertaken in 2012. The northern 4 km of the mineralised trend at Berlin requires exploration drilling to establish the potential of that area.
- There is potential to significantly increase the size of the Berlin deposit through infill drilling of the area in which exploration drilling was undertaken in 2012 as well as through exploration of the northern part of the trend which has never been drilled.

- Mining:
  - The PEA is based on production of 500,000 tpa of mineralised material. Associated access and development would involve the mining of approximately 250,000 tpa of waste.
  - Cut and fill mining methods are considered appropriate for the steeply inclined parts of the deposit, while room and pillar methods are modelled for the shallowly inclined western limb. Approximately 30% of the deposit would be mined by room and pillar techniques. Approximately 20% of the resource is modelled to be left as structural pillars to ensure the structural integrity of the underground operations. Hence, only 16.3 Mlb of the 1.5 Mlb Indicated and 19.9 Mlb Inferred of contained uranium is modelled to be extracted and processed.
  - The principal mine layout is envisaged to consist of a principal access decline and a production decline. Two vertical ventilation shafts complete the basic structure of the envisioned mine.
- Safety:
  - Principal mine safety access routes would be provided by the two access declines and the two ventilation shafts. A proposed exploration decline, whose principal function would be to provide access for rock strength data for detailed mine design as well as bulk samples for a metallurgical pilot plant at the feasibility stage of the project, would be incorporated into the overall mine design and would provide an additional safety access route.
  - Standard safety refuge chambers and fire suppressant equipment would be installed underground.
  - The radioactivity of the mineralised material from the resource area at Berlin does not constitute a source of radiation levels that would necessitate robotic mining. The ventilation system would be designed such that air flow mitigates miners' exposure to potential sources of radon gas.
  - Crushing and milling would be undertaken in an underground chamber to minimise radioactive dust escaping into the environment.
  - In the two scenarios modelled, all tailings in one scenario would be used as backfill and stored permanently underground so residual radiation would be minimised at surface. In the other scenario modelled, in which mineralised material is not beneficiated prior to processing, excess tailings that do not fit underground would be stored in a tailings facility on appropriately impermeable sub-stratum. The tailings facility would be designed to the highest safety standards and would be covered with a layer of soil sufficiently thick to mask residual radiation in the tailings.
- Communition:
  - ROM mineralised material would be crushed and milled to a grain size approximately 100 µm in an underground chamber located adjacent to the main production decline.
  - Milling would be done in a SAG mill.

- Metallurgical test work:
  - Metallurgical test work for the beneficiation and leaching portions of the flow sheet has been extensive and comprehensive and is of an appropriate level of confidence to be used in a PEA.
  - Test work for the metals recovery portion of the flowsheet has yet to be completed to prove the conceptual flow sheet and a test work program culminating in an integrated continuous piloting will be required for further definition of the project. Sufficient prior experience with the processes selected is available to provide the basis for costing this part of the circuit to a PEA standard.
  - Test work has been conducted on composite samples from 25 of the 74 intersections (34%) that form the basis of the mineral resource estimate on which this PEA is based (Coffey Mining, 2012). These intersections are from throughout the mineral resource area. For these two reasons, the material used in the metallurgical test work is considered to be representative of the resource area, and therefore the metallurgical test work is considered representative of the resource area.
  - Acidic ferric iron leach has proved to be very effective at in extracting uranium, phosphate, zinc, vanadium, yttrium and neodymium from the mineralised material at Berlin. It is moderately effective in extraction of molybdenum and nickel, and less effective for rhenium and silver.
  - Mineralised material from Berlin is amenable to beneficiation (reduction of carbonate content) by acetic acid, and the economics of mineralised material beneficiated in this way are compared with unbeneficiated mineralised material in this PEA.
  - Flotation is another means of beneficiation in which progress is being made in on-going test work. An initial sulphide flotation step has proved effective at concentrating nickel sulphide, zinc sulphide and pyrite. The behaviour of molybdenum and associated rhenium still needs to be established. However, there is a possibility that extraction of these metals may be better from a sulphide concentrate than they are with acidic ferric iron leach. If this is the case, the revenue stream from mineralised material from Berlin may be improved from the current models.
  - A combined carbon pre-float with sulphide flotation results in a sulphide-carbon concentrate that contains the majority of the uranium from the mineralised material.
  - Test work is now focused on following up encouraging results that have been obtained on the separation of carbonate from the main phosphate-bearing mineral, apatite.
  - Beneficiation of the mineralised material by flotation provides the most promising route for the extraction of a significant proportion of the carbonate, which is largely responsible for elevated reagent consumption, from mineralised material from Berlin.

- Uranium extraction to the flotation concentrate increased to approximately 71% with up to 87% of the carbon and 84% of the sulphide, 76% of the vanadium and 57% of the apatite and only 41% of the calcite. A further advantage of flotation to form a sulphide concentrate is that metal extraction may be higher from a concentrate than it is with the current acidic ferric iron leach process which may make a positive contribution to project economics. Reagent consumption, with associated capital and operating costs, may be reduced with beneficiation. Based on current knowledge and assumptions, the results of this study show the project to be economically robust.

- Recovery methods:
  - The current recovery process does not include rhenium. Further work should be undertaken to determine the extent to which this metal can be recovered from the PLS.
  - Recovery of metals from solution must be done in a specific order to avoid fouling of the down-stream extraction process. Molybdenum is extracted from the PLS first using IX to produce calcium molybednate. Uranium and vanadium are then recovered from the PLS with IX and sulphuric acid and peroxide are used to separate these two elements. Uranium is calcined to form UO_3 and vanadium is produced as ammonium metavanadate. Phosphoric acid is produced by SX. Rare earth elements, yttrium and neodymium are produced by direct precipitation as hydroxides. Nickel, zinc and manganese carbonates are extracted last by SX.
  - Pre-feasibility level test work is required to more fully define the details of the process by which the various metals are extracted from the PLS. Such studies would require a bulk sample of several tonnes to provide sufficient PLS for the tests to be considered representative.

- Tailings:
  - Preliminary work indicates that a benign tailing is likely to be produced from the metallurgical process that has been established for mineralised material from Berlin.
  - Radioactivity from tailings is to be mitigated in the environment by the storage of large proportion of tailings underground, while any excess would be stored in a suitably designed long-term facility located some 14km from site.
  - Beneficiation of the mineralised material with acetic acid results in a reduction in volume of tailings to the extent that all of the tailings would be stored underground. With the alternative process of not beneficiating the mineralised material, an excess of tailings would be produced such that an external tailings facility would be required. These excess tailings would be stored in a facility built on metamorphic and igneous rocks that have minimal permeability.

- Extent of confidence in mining and metallurgical processes on which this PEA is based:
  - The mining methods contemplated for the Berlin Project are commonly used in the mining industry and the expertise required to implement these methods is widely available within the extractive industry.

- Beneficiation: two scenarios are modelled – one using ROM mineralised material without beneficiation and the other uses acetic acid as a means of beneficiating the mineralised material. The latter approach is a relatively novel concept and has not been applied on a commercial scale in the mineral industry. However, laboratory test work shows the reaction of acetic acid with carbonate to be a simple process. Furthermore, test work shows that the regeneration of acetic acid with sulphuric acid and the associated generation of gypsum to be straight forward. Further laboratory test work and a pilot-scale testing would be required to demonstrate the applicability of this method as a viable beneficiation process for mineralised material from Berlin. It is important to note that the economic viability of the Berlin Project is not dependent on beneficiation by acetic acid and, in fact, the more economically attractive method is direct processing of ROM material without beneficiation with acetic acid.

- Beneficiation by flotation, which ongoing test work suggests may be applicable to mineralised material from Berlin as an alternative to acetic acid, uses technology that is widely used in the mineral processing industry.

- Variants of acidic ferric iron leach have been used for the extraction of uranium and yttrium in the Elliot Lake district in northern Ontario. It is currently being used for the extraction of uranium at Rossing Uranium mine in Namibia and Buffelsfontein mine in South Africa. It is also used for the extraction of a suite of metals, including uranium, from the multi-commodity Talvivaara mine in Finland. Certain aspects of the acidic ferric iron leach process have been developed and modified to optimise recoveries from the mineralised material from Berlin. Test work is ongoing and the intention is to scale this test work up with laboratory tests using larger masses of mineralised material to a pilot plant as the project advances.

- Recovery of metals and phosphate from the PLS uses well known technology including SX, IX and direct precipitation using reagents that are commonly used in the extractive industry. Recovery methods envisaged are similar to those that were used in the Elliot Lake camp and are currently used in processing facilities at Rossing, Buffelsfontein and Talvivaara.

• A 15 year mine life is envisaged with a nominal 250,000 t of mineralised material mined in the first year, ramping up to 500,000 tpa thereafter, and 210,000 t of mineralised material being mined in Year 15 for a total production of 16.3 Mlb. Approximately 20% of the resource is modelled to be sterilised because it is required to be left in place as structural pillars in the underground mine.

• The capital cost of the acetic option is estimated at $449.5 million while for the non-acetic option is $441.0 million. This includes sustaining capital of $42.5 million and $40.6 million contingency in both cases (Table 25-1).

• Revenue for the case with acetic acid beneficiation is estimated at $3.0 billion and for the option in which there is no beneficiation of the mineralised material, revenue is estimated at $2.8 billion. Potential revenues from rhenium and silver are not included in the financial model because the flow sheet does not yet define how these metals nay be economically extracted from the PLS. Similarly, potential revenues from manganese are excluded because there is not yet a mineral resource estimate for this element.

- Operating costs estimate at a uranium price of $60 /lb:
  - The option with beneficiation by acetic generates revenue of $429 /t of mineralised material with operating costs of $233 /t, yielding a free cash flow of $196 /t of mineralised material (Table 25-2);
  - With no beneficiation, revenue generated by the processing of ROM mineralised material is $406 /t and operating costs are $201 /t for a free cash flow of $205 /t of mineralised material.

Table 25-1: Estimated capital cost for the acetic and non-acetic alternative processing

Area	Acetic acid beneficiation			Without beneficiation
	Initial (million)	Sustaining (million)	Total (million)	Total (million)
Mining	$62.5	$11.4	$73.9	$73.9
Processing plant	$182.2	$12.5	$194.7	$177.4
Mine dewatering, environmental & closure	$10.0	$14.9	$24.9	$24.9
Infrastructure	$42.8	$3.7	$46.5	$59.0
Indirect costs	$68.9	--	$68.5	$65.2
Contingency	$40.6	--	$40.6	$40.6
Total	**$407.0**	**$42.5**	**$449.5**	**$441.0**

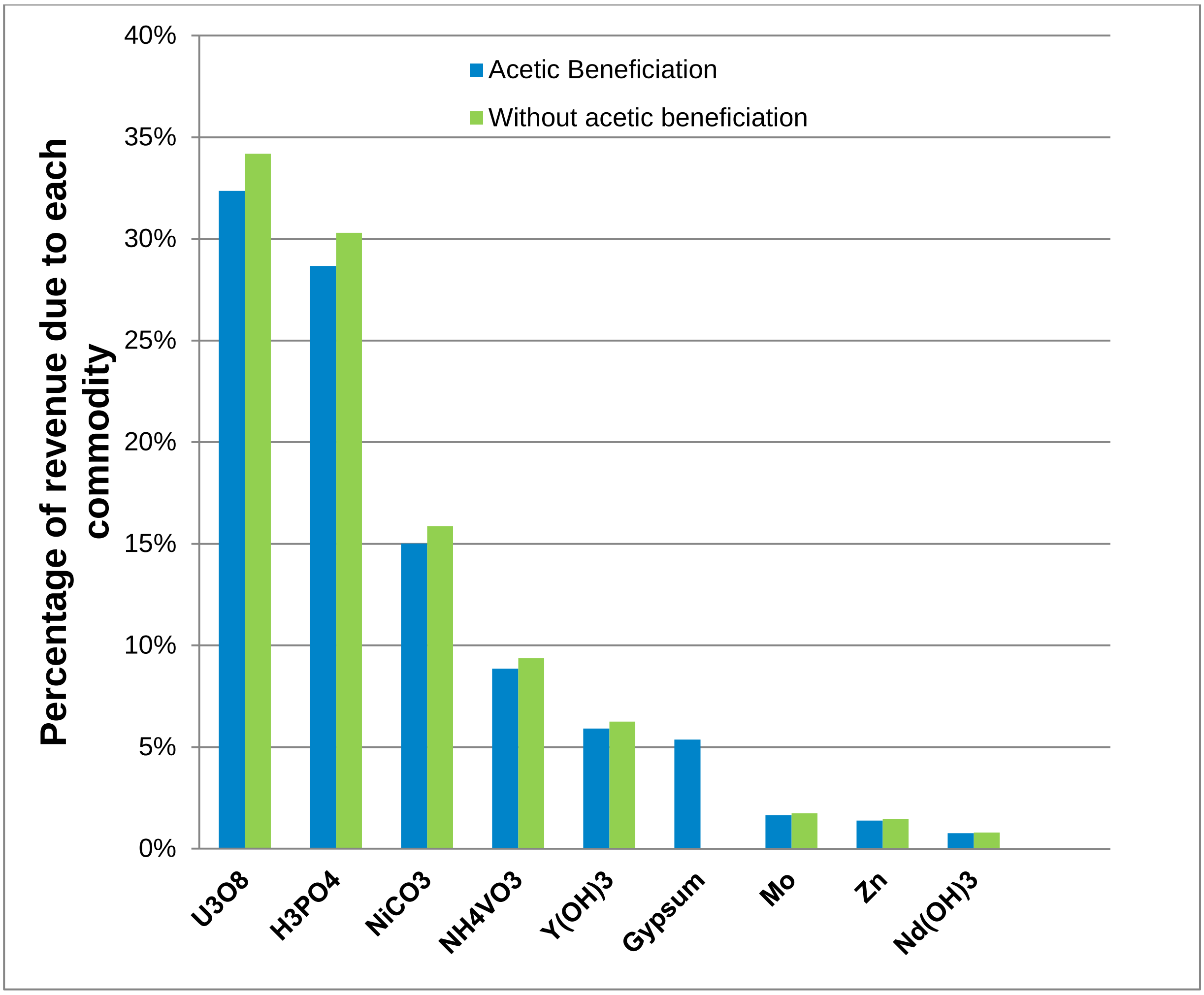

Figure 25-1: Percentage that each commodity contributes to total revenue for the case in which acetic acid is used to beneficiate the mineralised material versus the option of no beneficiation.

- At a discount rate of 10%, the DCF for the Berlin Project at a uranium price of $60 /lb is:
  - $192 million with an IRR of 17% and a payback period of 4.9 years for the option in which acetic acid is used for beneficiation; and
  - $223 million with an IRR of 19% and a payback period of 4.6 years.
- DCF models have shown that the project is viable at uranium prices down to $40 /lb at a discount rate of 10%. In addition, at a discount rate of 10%, DCF matrices show that the project remains robust with capital costs increasing over 20%, operating costs increasing over 20% and revenue decreasing as much as 15%.

Table 25-2: Estimated operating cost for the acetic and non-acetic alternative processing routes

	Acetic acid beneficiation		Without beneficiation	
	Cost per year (million)	Cost per tonne of mineralised material	Cost per year (million)	Cost per tonne of mineralised material
Revenue	$214.5	$429	$202.9	$406
Revenue-based royalties	($9.4)	($18)	($9.3)	($18)
Mining	($27.3)	($55)	($27.3)	($55)
Processing	($65.9)	($132)	($51.5)	($103)
G&A	($1.8)	($4)	($1.8)	($4)
Dewatering	($2.4)	($5)	($2.4)	($5)
Contingency	($9.7)	($19)	($8.3)	($16)
Total Operating Cost	($116.6)	($233)	($100.7)	($201)
	$97.9	$196	($101.7)	$205

26 RECOMMENDATIONS

26.1 Summary

Despite the completion of this PEA, which was undertaken mainly to establish the potential viability of a small part of what is evidently a deposit of large size potential, Berlin continues to be an exploration project. Typically, a PEA that demonstrates robust economics as this study does, recommends advancing to pre-feasibility. However, given the large size potential of the Berlin deposit, recommendations below concentrate firstly on expanding the size of the resource to cover the whole mineralised trend, and then to upgrading the resource from the Inferred to the Indicated category. Hand-in-hand with the exploration and resource drilling, ongoing metallurgical test work should be continued to prove the overall process flow sheet and to identify efficiencies may be obtained that may positively impact the economics of the project. Basic hydrological studies and geotechnical studies would contribute to the evaluation of conceptual mine designs.

In order to advance the Berlin Project towards a PFS, the following steps are recommended to be taken:

- Exploration and resource drilling should be undertaken in a modular, stepwise way as follows:
  - Wide-spaced exploration drilling to confirm the extent of continuity of mineralisation along trend to the northern extremity of the prospective host rocks (budget $3.3 million; Table 26-1);
  - Infill drilling of the areas adjacent to the current resource with the aim of increasing the current Inferred resource category through appropriately spaced section line drilling (budget $6.6 million);
  - Infill drilling to convert current and contiguous Inferred resources to the Indicated category (budget $11 million).
- On conversion of a significant part of the Inferred Resource to the Indicated category, a PFS should be undertaken on the potentially larger Berlin deposit;
- Ongoing metallurgical test work should continue with a focus on a means of efficiently beneficiating the mineralised material with the aim of reducing operating and capital costs while maintaining the revenue stream. The aim of this work would be to set up for larger-scale test work which would be required to support pre-feasibility stage studies;
- Initial hydrological studies should be undertaken using data from exploration drill holes;
- Comprehensive geotechnical studies should be undertaken of bore hole core from exploration and resource drilling programs; and
- Development of an exploration ramp to allow access to the mineralised zone at depth should be undertaken in conjunction with pre-feasibility level studies once the resource has been upgraded from dominantly Indicated to dominantly Inferred.

26.2 Process Development

26.2.1 Comminution

It is recommended that further investigations be carried out in the next stages of the project development into the effects of varying ore characteristics on comminution and leach.

The design of the milling circuit should be developed further in terms of:

- The type, power draw, diameter, grinding length and the ball charge of the SAG mill;
- AG vs. SAG grinding should be investigated;
- Should AG be viable, the ball milling operation could probably be removed from the process flow sheet. The process recycle water used for wet screening is a solution recycled from the various thickeners in the process. The effect of these on the grinding equipment and process should be determined by locked-cycle test work;
- Variability in mineralised material breakage characteristics is expected because the mineralised material will contain varying amounts of competent calcrete and apatite. Detailed investigation of ore characteristics should be investigated further;
- A detailed study of the underground layout of the mine and the space required for the crushing station is recommended with focus on providing enough space for a SAG mill; and
- The SAG mill could also be located at the plant, because SAG (or AG) and ball mills are wet processes and dust will not be a problem.

26.2.2 Leaching

26.2.2.1 Option A

The acetic acid leach residue is washed and transported to the acidic ferric iron leaching process. Further test work during the next phase of the project is required to determine the efficiency and the size of this washing circuit. The mass balance around the acetic acid leaching should also be investigated and developed in more detail. Further detail modelling of the circuit flow is required.

26.2.2.2 Option B

- The reaction of sulphuric acid with large quantities of calcite in the ROM mineralised material in the leaching section results in a volatile reaction. The effect on leaching efficiencies and other potential influences on design should be determined by pilot plant test work;
- Investigate the effects of higher flow rates on downstream processes; and
- Investigate the effect on downstream processes of large quantities of gypsum formed in the acidic ferric leach.

Test work on the characteristics of the different slurries is required to determine the size of the thickeners. The quantity of the thickeners required could increase to more than seven for the high washing efficiency assumed in Options A and B. Pilot test work is also required to determine the washing efficiency for the residues of Options A and B (that result in different slurries fed to the CCD). Current washing efficiency assumed is 100% for both cases. The use of belt filters instead of thickeners for washing should also be considered.

Test work is required to determine the design parameters of the pinned bed clarifier.

26.2.3 Metals Recovery Plant

It is emphasized that only an integrated pilot plant campaign for this complex separation system will confirm its technical viability. No test work results were available for the design of this part of the plant.

Difference in Mo recovery: The recovery system could be affected by the higher flow-rate of the PLS. A more detail design would determine if the system equipment size should be increased and if the recovery of Mo will stay the same. Test work to compare the two systems using the different precipitation reagents is required for the next stage.

The technical viability of REE recovery, especially for Option B, needs to be determined by test work.

The power requirements for the different sections should be designed taking into account future test work results. During the study, conservative assumptions were made in preparing the energy balance, including the heat and power available from the acid plant. Further more detailed design work is required to determine the heat and energy requirements. It may result in further lowering of the CAPEX and OPEX costs assumed.

The SX of Zn, Ni and Mn is complex and it is almost impossible to predict the extraction behaviour. The theoretical design of the plant must be followed up with extensive laboratory and pilot plant test work in the next stage of the project development.

26.3 Budget for recommendations

In Table 26-1 the budget associated with the recommendations for the Berlin Project is illustrated.

Table 26-1: Tabulation of budget associated with the recommendations for the Berlin Project

Exploration and Resource Growth				
Objective	Detail	Estimated metreage of drilling required (m)	Budget ($)	
Exploration	Exploration drilling of Northern and Western part of the Berlin syncline	8,500	All-in drilling budget	$2,800,000
			Assay	$80,000
			Contingency @ ~15%	$420,000
	Subtotal			**$3,300,000**
Proposal to increase Inferred Resource	Infill drilling in the northern extension of the Berlin trend. Drill spacing: Sections spaced at 400m intervals with intercepts within section at ~100m spacing	18,000	All-in drilling budget	$5,400,000
			Assay	$150,000
			Contingency @ ~15%	$850,000
	Resource estimation			$200,000
	Subtotal			**$6,600,000**
Proposal to increase Inferred Resource to Indicated category	Infill drilling throughout the Inferred resource area	35,000	All-in drilling budget	$9,000,000
			Assay	$300,000
			Contingency @ ~15%	$1,450,000
	Resource estimation			$250,000
	Subtotal			**$11,000,000**
Metallurgy:				
Ongoing testwork to define efficiencies in metal and phopshate extraction	Test material from exploration & resource drilling			$500,000
	Subtotal			**$500,000**
Hydrological & Assessment of Rock Quality:				
Initial hydrological studies on exploration and infill bore holes				$200,000
Geotechnical and strength tests on core from exploration and infill bore hole core				$100,000
	Subtotal			**$300,000**
Total				**$21,700,000**

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28 SIGNATURE PAGE

This technical report was written by the Qualified Persons listed in Section 2. The effective date of this technical report is January 18, 2013.

Reviewed by

Louis de Klerk, Pr. Eng., BSc(Eng) (Chemical), P Grad Dip in Advanced Process Design

All data used as source material plus the text, tables, figures, and attachments of this document have been reviewed and prepared in accordance with generally accepted professional engineering and environmental practices.

29 CERTIFICATES OF QUALIFIED PERSONS

- Paul Miller
- Pedro Veliz
- Louis de Klerk
- Pieter Niemann
- Doug Corley

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U3O8 CORP. PRELIMINARY ECONOMIC ASSESSMENT OF THE BERLIN DEPOSIT, COLOMBIA – 18 January 2013

CERTIFICATE OF QUALIFIED PERSON

As a qualified person responsible for preparing or supervising the preparation of all or part of the technical report ("the Report"), I hereby state:

1. My name is Louis William de Klerk and I am a Manager: Process Engineering with the firm Tenova Mining and Minerals Australia (Tenova) of Suite 2, 93 Francisco Street, Belmont, WA 6104, Australia.
2. This certificate applies to the "U3O8 CORP. PRELIMINARY ECONOMIC ASSESSMENT OF THE BERLIN DEPOSIT, COLOMBIA – 18 January 2013"
3. I am a member of the Engineering Council of South Africa and registered as a Professional Engineer. I graduated with a BSc (Eng) (Chemical) degree from the University of the Witwatersrand in 1978 and with a Post Graduate Diploma in Advanced Process Design from Monash University in 2012. I have been involved in metallurgical and petrochemical processing since 1979 and have practiced my profession continuously since then. I have been involved in process operations, process engineering, project development and project finance covering a wide range of mineral commodities for projects in South Africa, Zambia, Nigeria and Australia. As a result of my experience and qualifications, I am a Qualified Person ("QP") as defined in National Instrument 43-101 Standards of Disclosure of Mineral Projects (NI 43-101) (the "Instrument").
4. I have not visited the Berlin Project.
5. I certify that I am responsible for supervision of preparation of the technical report including Sections 1, 2, 3, 4, 5, 6, 7, 8, 9, 10, 15, 19, 20, 23, 24, 25, 26 and 27.
6. I am independent of U3O8 Corp. pursuant to section 1.5 of the Instrument.
7. I have had no prior involvement with the property that is the subject of the technical report. I do not have nor do I expect to receive a direct or indirect interest in the Berlin Project of U3O8 Corp. and I do not beneficially own, directly or indirectly, any securities of U3O8 Corp. or any associate or affiliate of such company.
8. I have read National Instrument 43-101 and the part of the report I have read has been prepared in compliance with that Instrument.
9. As of the date of this certificate, to the best of my knowledge, information and belief, the abovementioned chapters of the technical report contain all the scientific and technical information that is required to be disclosed to make the technical report not misleading.

Dated at Perth, Western Australia, on 18 January 2013.

Louis de Klerk
Pr. Eng., BSc (Eng) (Chemical), P Grad Dip in Advanced Process Design

Berlin Project, Colombia. NI-43-101 Technical Document – January 18th , 2013

Certificate of Qualified Person

As an author of the report entitled" U3O8 Corp Preliminary Economic Assessment on the Berlin Deposit Colombia " dated January 18th, 2013, on the Berlin Project of U3O8 Corp. (the "Study"), I hereby state:

1. My name is Paul Charles Miller and I am a Consulting Metallurgist and Managing Director with the firm of Sulphide Resource Processing Pty Ltd of 31 Mabena Place, Ocean Reef, WA 6027, Australia.

2. I am a practising Metallurgist and a Chartered Engineer (U.K.) and a Member of the Institute of Mining and Metallurgy London, England.

3. I am a graduate of Birmingham University, England with a BSc (Hons) in Minerals Processing in 1978 and an MSc in Biochemical Engineering in 1979 and a PhD in Chemical Engineering in 1982. In 1990, I graduated from the University of Cape Town, South Africa with a Master of Business Administration MBA.

4. I have practiced my profession continuously since 1982.

5. I am a "qualified person" as that term is defined in National Instrument 43-101 (Standards of Disclosure for Mineral Projects) (the "Instrument").

6. I have not visited the Berlin Project. I have performed consulting services and reviewed files and data supplied by U3O8 Corp., SGS Mineral Services in Lakefield, Ontario and SGS Oretest Minerals Services in Perth, Australia between June 2011 and March 2012. I have supervised the further test work conducted at SGS Oretest Perth and at Optimet Laboratories South Australia between April 2012 and December 2012.

7. I contributed to and am responsible for parts of Section 1, 13, 25, 26 and 29 of the Study.

8. As of the effective date of the Study, to the best of my knowledge, information and belief, the parts of the Study for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Study not misleading.

9. I am independent of U3O8 Corp. pursuant to section 1.5 of the Instrument.

10. I have read the National Instrument and Form 43-101F1 (the "Form") and the Study has been prepared in compliance with the Instrument and the Form.

11. I do not have nor do I expect to receive a direct or indirect interest in the Berlin Project of U3O8 Corp., and I do not beneficially own, directly or indirectly, any securities of U3O8 Corp. or any associate or affiliate of such company.

Dated at Perth, Western Australia, on January 18th,2013.

Paul Miller

Paul C. Miller
Managing Director (Consulting Metallurgist) Sulphide Resource Processing Pty Ltd
BSc (Hons); MSC; PhD (Chem. Eng.); MBA, C. Eng; MIMM

Certificate of Qualified Person

As an author of parts of the report entitled "U3O8 Corp. Preliminary Economic Assessment on the Berlin Deposit, Colombia" dated 18 January 2013, on the Berlin Project property of U3O8 Corp. (the "Study"), I hereby state:

1. My name is Pedro S. P. Véliz and I am a Principal Consultant and President of the firm, P&K Projetos e Consultoria Ltda, with head office at Avenida Rio Branco 277 – Conj. 1410 – Centro – Rio de Janeiro, RJ – 20040-009 – Brazil.

2. I am a Professional Engineer (P. E.) in Brazil, a Member of the Society for Mining, Metallurgy & Exploration – SME (#4165319), member of the American Chemical Society – ACS (#30135187) and member of the Society of Petroleum Engineers – SPE (#3491269).

3. I hold the title of Mining Engineer granted by the Federal University of Rio Grande do Sul, Brazil 1976. I graduated as Civil Mining Engineer from the Technical University of the State, Chile in 1974.

4. I have practiced my profession continuously since 1976 working with companies involved in the mining and exploration of metal and industrial mineral deposits in South America. As a consultant, I have worked on projects involving base metals, iron ore, manganese, bauxite, coal, scheelite-Bi-Mo skarn, gold, palladium, platinum, diamond, limestone, kaolin, and uranium.

5. I am a "qualified person" as that term is defined in National Instrument 43-101 (Standards of Disclosure for Mineral Projects) (the "Instrument").

6. I visited the Berlin Project between 23 through 26 April 2012.

7. I contributed to and I am responsible for parts of Sections: 1 (Summary), 16 (Mining), 18 (Infrastructure), 21 (Capex/Opex), 22 (Economics), 25 (Conclusions) and 26 (Recommendations) of the Study.

8. As of the effective date of the Study, to the best of my knowledge, information and belief, the parts of the Study for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Study not misleading.

9. I am independent of U3O8 Corp. pursuant to section 1.5 of the Instrument.

10. I have read the National Instrument and Form 43-101F1 (the "Form") and the Study has been prepared in compliance with the Instrument and the Form.

11. I do not have nor do I expect to receive a direct or indirect interest in the Berlin Project of U3O8 Corp., and I do not beneficially own, directly or indirectly, any securities of U3O8 Corp. or any associate or affiliate of such company.

Dated at Rio de Janeiro, Brazil, on January 18, 2013.

Pedro S. P. Véliz
President (Mining Consultant) - SME

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U3O8 CORP. PRELIMINARY ECONOMIC ASSESSMENT OF THE BERLIN DEPOSIT, COLOMBIA – 18 January 2013

CERTIFICATE OF QUALIFIED PERSON

As a qualified person responsible for preparing or supervising the preparation of all or part of the technical report ("the Report"), I hereby state:

1. My name is Pieter Jacobus Niemann and I am a Senior Process Engineer with the firm Tenova Mining and Minerals Australia (TMM) of Suite 2, 93 Francisco Street, Belmont, WA 6104, Australia.
2. This certificate applies to the "U3O8 CORP. PRELIMINARY ECONOMIC ASSESSMENT OF THE BERLIN DEPOSIT, COLOMBIA – 18 January 2013"
3. I am a member of the Engineering Council of South Africa and registered as a Professional Engineer. I graduated with a B.Eng. degree from the Potchefstroom University for Christian Higher Education in 1985 and with a Post Graduate MBA Degree from the same University later. I have been involved in metallurgical and petrochemical processing since 1985 and have practiced my profession continuously since then. I have been involved in process operations, process engineering, project development and project finance covering a wide range of mineral commodities for projects in South Africa, Namibia and Australia. As a result of my experience and qualifications, I am a Qualified Person ("QP") as defined in National Instrument 43-101 Standards of Disclosure of Mineral Projects (NI 43-101) (the "Instrument").
4. I have not visited the Berlin Project.
5. I certify that I have supervised preparation of the technical report and am responsible for Section 17. I further certify that I have contributed to Sections 18 and 21 and the parts of Sections 1, 2, 3, 24, 25 and 26 relevant to the recovery methods.
6. I am independent of U3O8 Corp. pursuant to section 1.5 of the Instrument.
7. I have had no prior involvement with the property that is the subject of the technical report. I do not have nor do I expect to receive a direct or indirect interest in the Berlin Project of U3O8 Corp. and I do not beneficially own, directly or indirectly, any securities of U3O8 Corp. or any associate or affiliate of such company.
8. I have read National Instrument 43-101 and the part of the report I have read has been prepared in compliance with that Instrument.
9. As of the date of this certificate, to the best of my knowledge, information and belief, the abovementioned chapters of the technical report contain all the scientific and technical information that is required to be disclosed to make the technical report not misleading.

Dated at Perth, Western Australia, on 18 January 2013.

PJ Niemann

Pieter Niemann
Pr. Eng., B.Eng. (Metallurgy), MBA

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coffey mining
SPECIALISTS FROM BOARDROOM TO MINE FACE

Berlin Project, Colombia. NI 43-101 Technical Report – January 18, 2013

Certificate of Qualified Person

As an author of the report titled "U3O8 Corp. Preliminary Economic Assessment on the Berlin Deposit, Colombia" dated January 18, 2013, on the Berlin Project of U3O8 Corp. (the "Study"), I hereby state:

1. My name is Doug Corley and I am a Principal Resource Geologist with the firm of Coffey Mining Pty Ltd of 1162 Hay Street, West Perth, Western Australia, Australia.
2. I am a practising geologist with 22 years of mining and resource estimation experience. I am a member of the Australian Institute of Geoscientists ("MAIG").
3. I am a graduate of James Cook University – Townsville, Queensland and hold a Bachelor of Science Degree (with Honours) in Geology (1991). I have practiced my profession continuously since 1991.
4. I am a "qualified person" as that term is defined in National Instrument 43-101 (Standards of Disclosure for Mineral Projects) (the "Instrument").
5. I visited the property that is the subject of this Report for the period from June 13 to June 19, 2012.
6. I contributed to and am responsible for parts of Sections 1, 11, 12, 14, 25, and 26 of the Study.
7. I am not aware of any material fact or material change with respect to the subject matter of the Report that is not reflected in the Report, or the omission to disclose which makes the Report misleading and that as of the date of this certificate, to the best of my knowledge, information and belief, the Report contains all scientific and technical information that is required to be disclosed to make the Report not misleading.
8. I am independent of the issuer applying all of the tests in section 1.5 of the National Instrument 43-101.
9. I have read the National Instrument and Form 43-101F1 (the "Form") and the Technical Report has been prepared in compliance with the Instrument and the Form.
10. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessibly by the public, of the Technical Report.

Coffey Mining Pty Ltd

Dated at Perth, Western Australia, on January 18, 2013.
For and on behalf of Coffey Mining Pty Ltd

This is a scanned signature held on file by Coffey Mining. The person and signatory consents to its use only for the purpose of this document.

Doug Corley, BSc (Hons); MAIG (RPGeo)
Principal Resource Geologist, Coffey Mining, West Perth

U3O8 CORP. PRELIMINARY ECONOMIC ASSESSMENT ON THE BERLIN DEPOSIT, COLOMBIA

Report Prepared for

U3O8 Corp.

APPENDIX A: ACRONYMS AND ABBREVIATIONS

Report Prepared by

Tenova Mining & Minerals (Australia) Pty Ltd

M6088.A

Abbreviation	Description
“	inches
%	percent
#	mesh
µm	microns
3D	three dimensional
AAS	atomic absorption spectrometry
Ag	Silver
AG	Autogenous grinding
amsl	Above mean sea level
ANSTO	Australian Nuclear Science and Technology Organisation
AusIMM	Australasian Institute of Mining and Metallurgy
BSE	Backscattered electron microscope
BTW	B-thinwall
BW_i	Bond Work index
C	Carbon
Ca	Calcium
CAPEX	Capital cost estimate
CCD	Counter current decantation
CIM	Canadian Institute of Mining, Metallurgy & Petroleum
CL	Clay (Universal Soil Classification System)
cm	centimetre
Co	cobalt
COG	Cutoff Grade
cps	Counts per second
CRM	certified reference material or certified standard
Cu	copper
D_2EHPA	di-(2-ethylhexyl)phosphoric acid
DCF	Discounted cash flow
DEM	Digital elevation model
EDS	Energy dispersive system
EGL	Effective grinding length
EIA	Environmental impact assessment
EPCM	Engineering, Procurement and Construction Management
$_eU_3O_8$	equivalent U_3O_8
Fe	iron
ft.	foot
FTZ	Free trade zone
FS	Feasibility study
g	gram

Abbreviation	Description
G&A	General and Administration
g/L	grams per litre
g/t	grams per tonne
GPS	Geographical positioning system
h or hr	hours
ha	hectare
HCl	hydrochloric acid
H_2SO_4	sulphuric acid
HDPE	High density polyethylene
HP	horsepower
HQ	size of diamond drill rod/bit/core
ICP-AES	inductively coupled plasma atomic emission spectroscopy
ICP-MS	inductively coupled plasma mass spectroscopy
INGEOMINAS	National Institute of Geology and Mining (Colombia)
IRR	Internal Rate of Return
ISO	International Standards Organisation
IX	Ion exchange
JORC	Joint Ore Reserves Committee
K	Potassium
Kd	Dissociation constant
kg	kilogram
km	kilometres
km^2	square kilometres
kV	kilovolt
kW	kilowatt
kWh	kilowatt hour
IAN	Instituto de Asuntos Nucleares (Colombia)
lb	pounds
LHD	Load haul dump
m	metres
M	million
m^3/h	cubic metres per hour
m^3/t	cubic metres per tonne
Ma	Million years
MCC	Motor control centre
Mg	Magnesium
ml	millilitre
ML	Silt (Universal Soil Classification System)
Mlb	Million pounds

Abbreviation	Description
mm	millimetres
Mn	Manganese
Mo	Molybdenum
MPa	Megapascals
Mt	Million tonnes
Mtpa	million tonnes per annum
MW	Megawatt
NATM	New Australian Tunnelling Method
Nb	Niobium
Nd	Neodymium
Ni	Nickel
NI 43-101	National Instrument 43-101 (Canada)
NORM	Naturally occurring radioactive material
NPV	net present value
NQ	size of diamond drill rod/bit/core
NSR	Net smelter return
NTW	N-thinwall
ºC	degrees Centigrade/Celsius
OPEX	Operating cost estimate
oz	ounce
P	Phosphorus
P_2O_5	Diphosphorus Pentoxide
P_{80}	80% of sample passing
PBC	Pinned Bed Clarifier
PEA	Preliminary economic assessment
PDC	Power distribution centre
PFS	Pre-feasibility study
PLS	Pregnant liquor solution
POX	Pressure oxygen leaching
ppb	parts per billion
ppm	parts per million
PSA	Particle size analyser
PVC	Polyvinyl chloride
QAQC	Quality assurance, quality control
QC	quality control
Re	Rhenium
REE	Rare earth element
ROM	Run-of-mine
RQD	rock quality designation

Abbreviation	Description
RMR	Rock mass rating
SAG	Semi-autogenous grinding
SEM	Scanning electron microscope
Si	Silica
Sr	Strontium
st	short ton
SX	Solvent extraction
t	metric tonne
t/m^3	tonnes per cubic metre
Ta	Tantalum
TBP	Tributyl phosphate
Th	Thorium
TOC	total organic carbon
tpa	tonnes per annum
tpd	Tonnes per day
TSX	Toronto Stock Exchange
U	Uranium
U_3O_8	tri uranium octoxide
UNDP	United Nations Development Program
UO_2	Uranium dioxide
UO_4	Uranium tetroxide
UTM	Universal Transverse Mercator
V	Vanadium
V_2O_5	Vanadium pentoxide
VAT	Value-added tax
WRF	Waste rock facility
XRD	X-Ray diffraction
XRF	X-Ray fluorescence
Y	Yttrium
Y_2O_3	Yttrium oxide
Zn	Zinc

U3O8 CORP. PRELIMINARY ECONOMIC ASSESSMENT ON THE BERLIN DEPOSIT, COLOMBIA

Report Prepared for

U3O8 Corp.

APPENDIX B: EXTRACTION COEFFICIENT DIAGRAMS

Report Prepared by

Tenova Mining & Minerals (Australia) Pty Ltd

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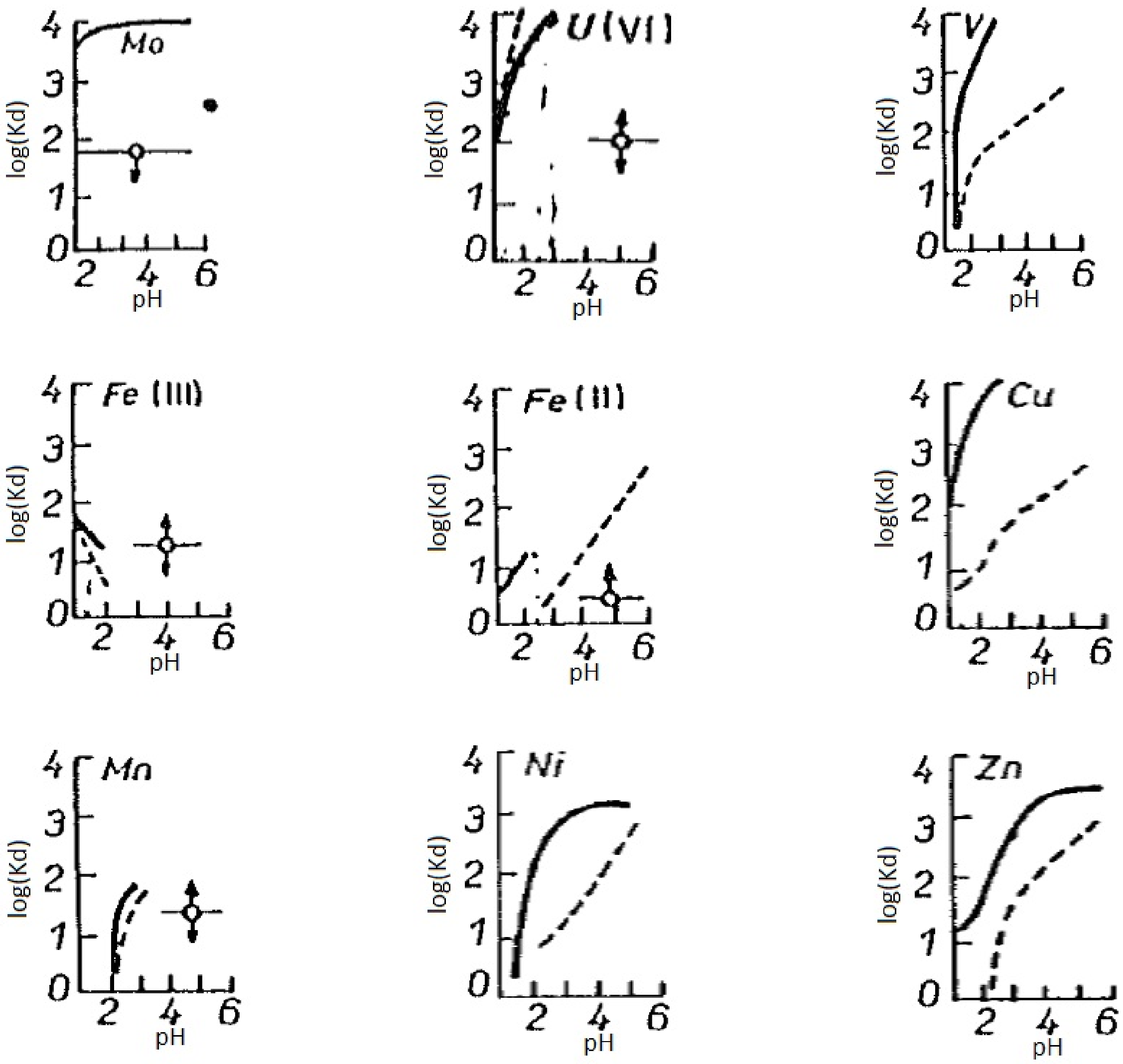

Figure 1: Kd for relevant ions as a function of liquor pH (Marhol M., 1982)

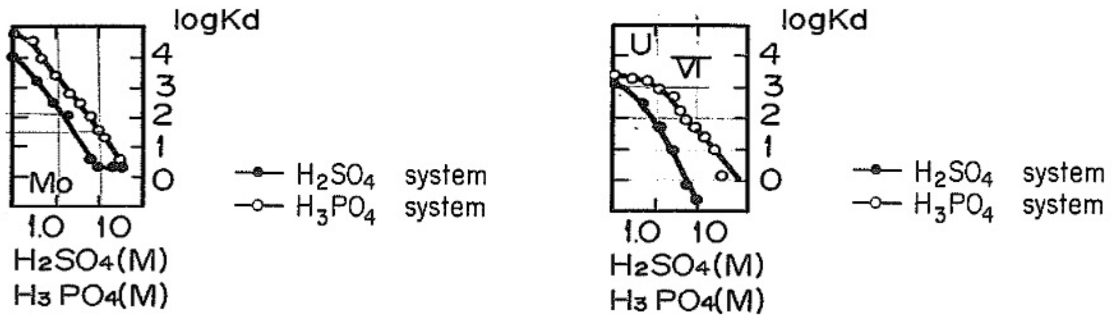

Figure 2: Kd for U, V and Mo in Phosphoric and Sulphuric acids at increasing acid concentration in industrial resins developed for this task (Akatsu E, 1977)

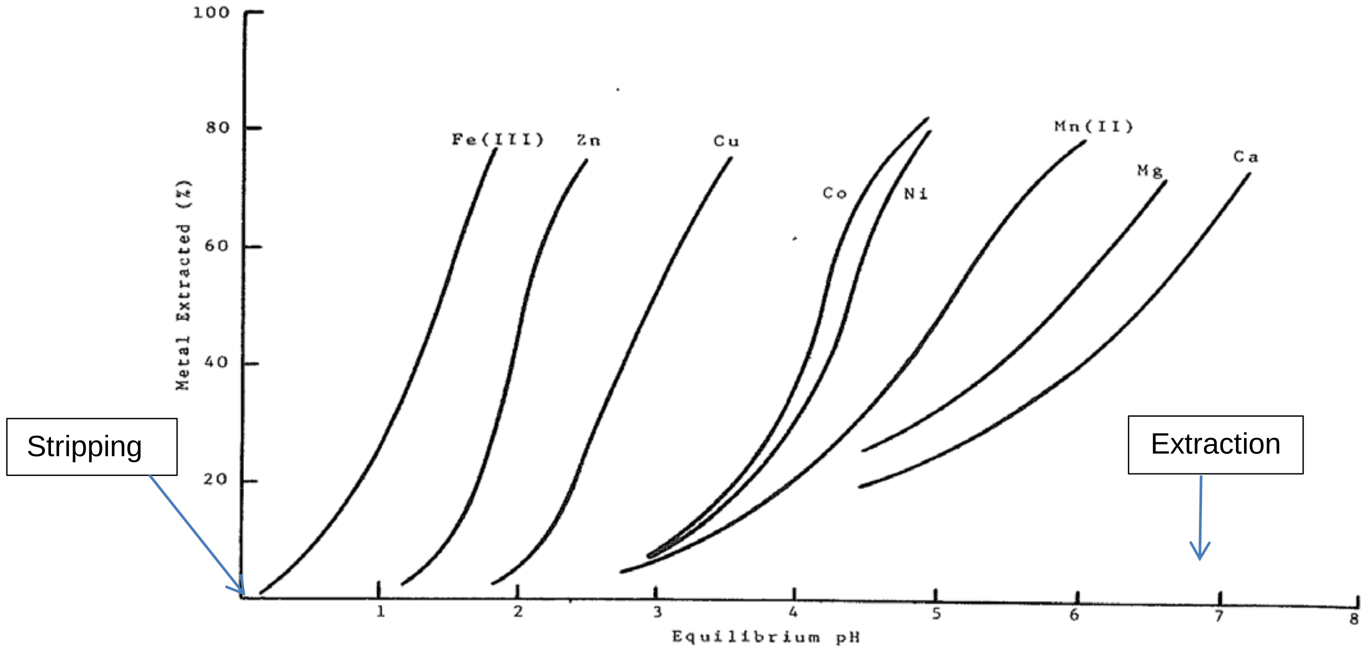

Figure 3: Extraction of metals using D2EHPA from a sulphate solution. (Gordon M. Ritcey, 2006)

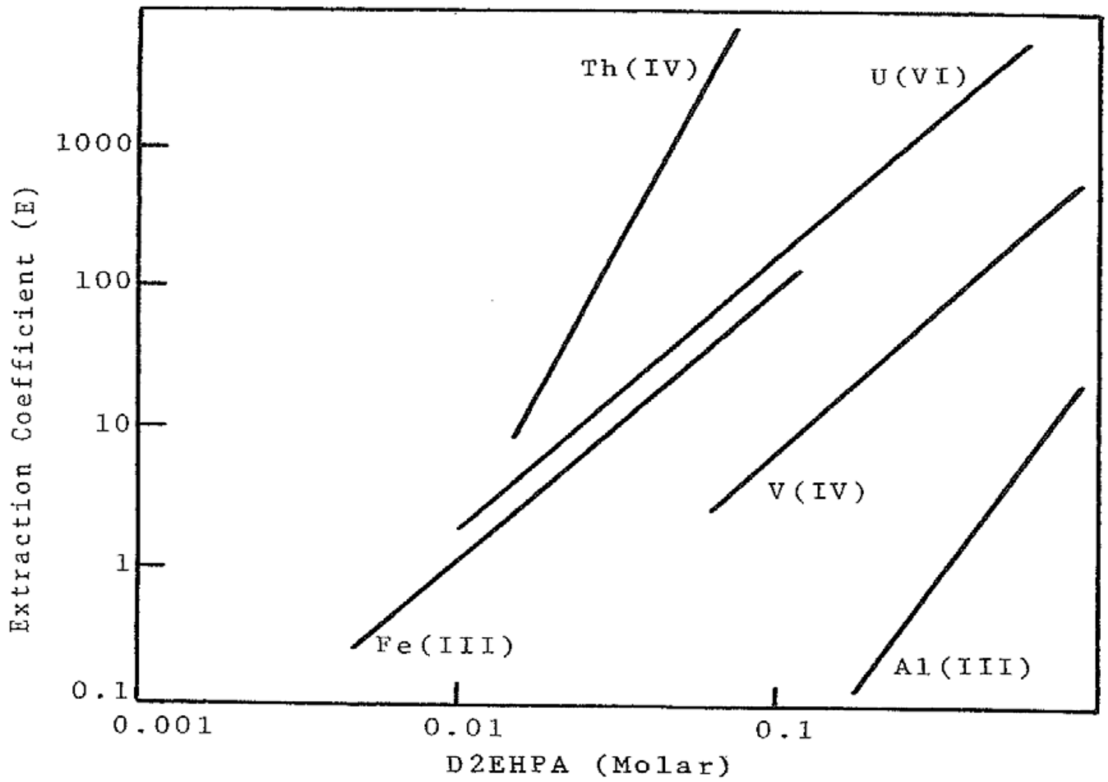

Figure 4: The effect of D2EHPA concentration on metal extraction from a 0.5 sulphate solution. (Gordon M. Ritcey, 2006)

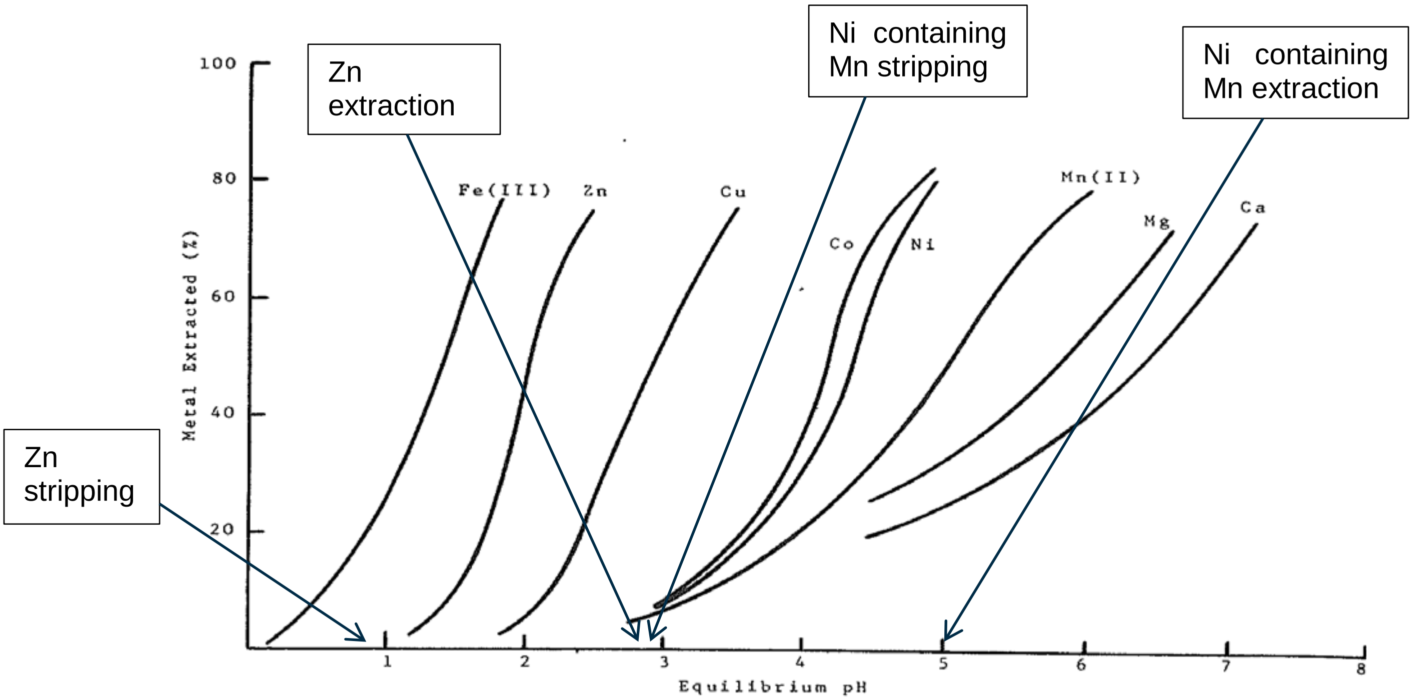

Figure 5: Separate Extraction of Zinc and Nickel Containing Manganese using a D2EHPA from a Sulphate Solution. (Gordon M. Ritcey, 2006)

U3O8 CORP. PRELIMINARY ECONOMIC ASSESSMENT ON THE BERLIN DEPOSIT, COLOMBIA

Report Prepared for

U3O8 Corp.

APPENDIX C: OPTION A
PROCESS PLANT MECHANICAL EQUIPMENT LIST

Report Prepared by

Tenova Mining & Minerals (Australia) Pty Ltd

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The Uranium Discovery Company

U3O8 CORPORATION LIMITED

BERLIN PROJECT - OPTION A

PEA STUDY

MECHANICAL EQUIPMENT LIST - 1 MTPA

M6088.A-M810-001

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REV	DATE	REVISION HISTORY	ORIGINATED	CHECKED	APPROVED	TENOVA APPROVAL
0	25/01/2013	ISSUED FOR USE	J.SCHLOFFER	P.NIEMANN	R.RAITER	A.WALKER
C	22/11/2012	ISSUED FOR ICR	J.SCHLOFFER	P.NIEMANN	R.RAITER	
B	7/11/2012	REVISION	J.SCHLOFFER	P.NIEMANN		
A	10/10/2012	ISSUED FOR IDR	J.SCHLOFFER	R.RAITER		

PLANT AREA DESCRIPTION

AREA NUMBER	AREA DESCRIPTION
100	CRUSHING
200	GRINDING
300	ACETIC ACID LEACHING
350	ACETIC ACID REGENERATION
400	SULPHURIC ACID LEACHING
500	ION EXCHANGE - Mo, V & U
540	Mo RECOVERY
560	U RECOVERY
580	V RECOVERY
600	SOLVENT EXTRACTION - Phosphoric Acid & Rare Earths
700	SOLVENT EXTRACTION - Mn, Ni & Zn
800	PHOSPHORIC ACID CONCENTRATION
830	RARE EARTH RECOVERY
860	MANGANESE, NICKEL & ZINC RECOVERY
900	REAGENTS
1000	WASTE WATER TREATMENT
1100	UTILITIES & SERVICES
1200	BUILDINGS

MECHANICAL EQUIPMENT LIST

Tag	Duty / Standby	Quantity	Equipment title	Capacity	Materials Of Construction	No of Drives	Variable speed drive	kW Installed	Rev
AREA 100	**-**		**CRUSHING**						
100-FL-001	Duty	1	FRONT END LOADER	5t			NO		C
100-SC-001	Duty	1	STATIC SCALPING SCREEN	175t/hr	Carbon Steel		NO		C
100-DT-001	Duty	1	METAL DETECTOR	11.0kW		1	NO	11.0	C
100-SC-002	Duty	1	VIBRATING SCALPING SCREEN	175t/hr, 30kW	Carbon Steel	1	YES	30.0	C
100-CR-001	Duty	1	JAW CRUSHER	175t/hr, 84.84kW	Carbon Steel/cast Manganese Steel	1	YES	84.84	C
100-CV-001	Duty	1	JAW CRUSHER DISCHARGE PRODUCT CONVEYOR	175t/hr		1	YES	2.0	C
100-CV-002	Duty	1	JAW CRUSHER DISCHARGE SPILLAGE CONVEYOR			1	YES	5.0	C
100-PU-001	Duty	2	DUST SUPPRESSION WATER PUMP	1.1kW	Carbon Steel	2	YES	1.1	C
100-PU-003	Duty	1	CRUSHING AREA SUMP PUMP	3kW	Carbon Steel	1	YES	3.0	C
100-SS-001	Duty	1	CRUSHING AREA SAFETY SHOWER/EYEWASH		Stainless Steel 316L		NO		C
AREA 140	**-**		**STOCKPILING**						
140-AF-001	Duty	2	APRON FEEDER	175t/hr	Carbon Steel/cast Manganese Steel	2	YES	55.0	C
140-CV-001	Duty	1	SAG MILL FEED CONVEYOR	135t/hr		1	YES	5.0	C
140-SR-001	Duty	1	STOCKPILE AREA DUST SCRUBBER	30kW	Carbon Steel	1	YES	30.0	C
AREA 200	**-**		**GRINDING**						
200-ML-001	Duty	1	SAG MILL	135t/hr	Carbon Steel/ Cr-Mo	1	YES	500.0	C
200-FD-001	Duty	1	SAG MILL DISCHARGE SCREEN VIBRATING FEEDER	305t/hr	Carbon Steel / Bisalloy liners	1	YES	5.0	C
200-SC-001	Duty	1	SAG MILL DISCHARGE SCREEN	305t/hr	Carbon Steel, rubber screens	1	YES	10.0	C
200-CV-001	Duty	1	SAG MILL RECYCLE CONVEYOR	202t/hr, 30kW		1	YES	30.0	C
200-CV-002	Duty	2	SAG MILL RECYCLE CONVEYOR	202t/hr, 30kW		1	YES	30.0	C
200-PU-001	Duty	2	BALL MILL HOPPER PUMP	436m^3/hr, 220kW	Carbon Steel	1	YES	220.0	C
200-CY-001	Duty	3	CYCLONE				NO		C
200-ML-002	Duty	1	BALL MILL		Carbon Steel	1	YES	2500.0	C
200-CR-001	Duty	1	PEBBLE CRUSHER	30t/hr (*to be confirmed by lead engineer)	Carbon Steel	1	YES	20.0	C
200-CV-004	Duty	1	PEBBLE CRUSHER TO SAG MILL RECYCLE CONVEYOR			1	YES	5.0	C
200-CN-001	Duty	2	MILL CRANE	15t	Carbon Steel	1	NO	11.0	C
200-PU-003	Duty	2	DUST SUPPRESSION WATER PUMP	1.1kW	Carbon Steel	1	YES	1.1	C
200-SR-001	Duty	1	GRINDING AREA DUST SCRUBBER	5kW	Carbon Steel	1	YES	5.0	C
200-PU-005	Duty	1	GRINDING AREA SUMP PUMP	22kW	Carbon Steel	1	YES	22.0	C
200-SS-001	Duty	1	GRINDING AREA SAFETY SHOWER/EYEWASH		Stainless Steel 316L		NO		C
AREA 300	**-**		**ACETIC ACID LEACHING**						
300-TK-001	Duty	5	LEACHING TANK	120m^3	Stainless Steel 316L		NO		C
300-AG-001	Duty	5	LEACHING TANK MIXER AGITATOR	5.5kW	Stainless Steel 316L	5	YES	5.5	C
300-PU-001	Duty	2	LEACHING TANK SLURRY DISCHARGE PUMP	944m^3/hr and 280kW	Stainless Steel 316L	2	YES	280.0	C
300-TH-001	Duty	2	LEACH RESIDUE THICKENER	200m^2 or 16m diameter		2	YES	3.0	C
300-PU-003	Duty	3	LEACH RESIDUE THICKENER UNDERFLOW PUMP	40m^3/hr and 22kW	Stainless Steel 316L	2	YES	22.0	C
300-TK-006	Duty	1	LEACH RESIDUE WATER WASH MIXING TANK	269m^3	Stainless Steel 316L		NO		C
300-AG-006	Duty	1	LEACH RESIDUE WATER WASH MIXING TANK AGITATOR	18.5kW	Stainless Steel 316L	1	YES	18.5	C
300-PU-006	Duty	2	LEACH RESIDUE WATER WASH MIXING TANK DISCHARGE PUMP	244m^3/hr and 90kW	Stainless Steel 316L	2	YES	90.0	C
300-TH-003	Duty	1	LEACH RESIDUE WASHING THICKENER	200m^2 or 16m diameter		1	YES	3.0	C
300-PU-008	Duty	2	LEACH RESIDUE WASHING THICKENER UNDERFLOW PUMP	150m^3/hr and 16kW	Stainless Steel 316L	1	YES	16.0	C
300-PU-010	Duty	1	CALCIUM ACETATE PUMP	25m^3/hr and 11kW	Stainless Steel 316L	1	YES	11.0	C
300-SR-001	Duty	1	LEACHING AREA SCRUBBER	5kW		1	YES	5.0	C
300-PU-011	Duty	1	LEACHING AREA SUMP PUMP	30kW		1	YES	30.0	C
300-SS-001	Duty	1	LEACHING AREA SAFETY SHOWER/EYEWASH		Stainless Steel		NO		C
AREA 350	**-**		**ACETIC ACID REGENERATION**						
350-TK-001	Duty	6	GYPSUM PRECIPITATION TANK	173m^3	Carbon Steel, rubber lined		NO		C
350-AG-001	Duty	6	GYPSUM PRECIPITATION TANK AGITATOR	7.5kW	Stainless Steel 316L	6	YES	7.5	C
350-PU-001	Duty	2	GYPSUM PRECIPITATION TANK SLURRY DISCHARGE PUMP	842m^3/hr and 280kW	Stainless Steel 316L	1	YES	280.0	C
350-TH-001	Duty	1	GYPSUM PRECIPITATE THICKENER	200m^2 or 16m diameter, 5.5kW	Carbon Steel, rubber lined	1	YES	5.5	C
350-PU-003	Duty	2	GYPSUM PRECIPITATE THICKENER UNDERFLOW PUMP	136m^3/hr and 55kW	Stainless Steel 316L	1	YES	55.0	C
350-TK-007	Duty	2	BELT FILTER FEED TANK	628m^3	Stainless Steel 316L		NO		C
350-AG-007	Duty	1	BELT FILTER FEED TANK AGITATOR	45Kw	Stainless Steel 316L	1	YES	45.0	C
350-PU-005	Duty	3	BELT FILTER FEED PUMP	136m^3/hr and 55kW		1	YES	55.0	C
350-FL-001	Duty	1	BELT FILTER	136m^3/hr at 50% gypsum solids	Stainless Steel 316L	1	YES	8.0	C
350-TK-009	Duty	1	BELT FILTER WASH WATER TANK	255m^3	Stainless Steel 316L		NO		C
350-PU-008	Duty	1	BELT FILTER WASH WATER PUMP	100m^3/hr and 30kW	Stainless Steel 316L	1	YES	30.0	C
350-TK-010	Duty	1	BELT FILTER FILTRATE TANK	332m^3	Stainless Steel 316L		NO		C
350-PU-009	Duty	1	BELT FILTER FILTRATE TANK PUMP	95m^3/hr and 30kW	Stainless Steel 316L	1	YES	30.0	C
350-PU-010	Duty	1	BELT FILTER VACUUM PUMP	50m^3/hr and 6kW	Stainless Steel 316L	1	YES	6.0	C
350-PU-011	Duty	1	BELT FILTER BELT CLEANER SPRAY PUMP	120m^3/hr and 37kW	Stainless Steel 316L	1	YES	37.0	C
350-TK-011	Duty	2	FILTRATE RECEIVER		Stainless Steel 316L		NO		C
350-PU-012	Duty	1	FILTRATE RECEIVER TRANSFER PUMP	95m^3/hr and 30kW	Stainless Steel 316L	1	YES	30.0	C
350-CI-001	Duty	1	BELT FILTER CHILLERS	5kW	Carbon Steel	1	YES	5.0	C
350-CV-001	Duty	1	BELT FILTER PRODUCT CONVEYOR	5kW	Carbon Steel	1	YES	5.0	C
350-TK-012	Duty	1	ACETIC ACID SURGE TANK	812m^3	Stainless Steel 316L		NO		C
350-AG-008	Duty	1	ACETIC ACID SURGE TANK AGITATOR	18.5kW	Stainless Steel 316L	1	YES	18.5	C
350-PU-013	Duty	2	ACETIC ACID SURGE TANK DISCHARGE PUMP	693m^3/hr and 220kW	Stainless Steel 316L	1	YES	220.0	C
350-PU-019	Duty	1	ACETIC ACID REGENERATION AREA SUMP PUMP	22kW	Stainless Steel 316L	1	YES	22.0	C
350-SS-001	Duty	1	ACETIC ACID REGENERATION AREA SAFETY SHOWER/EYEWASH		Stainless Steel 316L		NO		C

MECHANICAL EQUIPMENT LIST

Tag	Duty / Standby	Quantity	Equipment title	Capacity	Materials Of Construction	No of Drives	Variable speed drive	kW Installed	Rev
AREA 400	**-**		**SULPHURIC ACID LEACHING**						
400-TK-001	Duty	14	LEACHING TANK	$189m^3$	Carbon Steel, rubber lined		NO		C
400-AG-001	Duty	14	LEACHING TANK AGITATOR	11kW	Carbon Steel, rubber lined	14	YES	11	C
400-PU-001	Duty	4	LEACHING TANK SLURRY DISCHARGE PUMP	$151m^3$/hr and 75kW	Stainless Steel 316L	2	YES	75	C
400-TH-001	Duty	2	CCD WASHING THICKENER	$200m^2$ or 16m diameter	Carbon Steel	2	YES	3	C
400-PU-005	Duty	4	CCD WASHING THICKENER UNDERFLOW PUMP	$139m^3$/hr and 75kW	Stainless Steel 316L	2	YES	75	C
400-PU-009	Duty	1	PINNED BED CLARIFIER PUMP	$180m^3$/hr and 75kW	Stainless Steel 316L	1	YES	75	C
400-CL-001	Duty	1	PINNED BED CLARIFIER	$230m^3$	Carbon Steel, rubber lined		NO		C
400-TK-015	Duty	2	IRON REDUCTION TANK	$14m^3$	Carbon Steel, rubber lined		NO		C
400-PU-010	Duty	1	IRON REDUCTION TANK PUMP	$180m^3$/hr and 75kW	Stainless Steel 316L	1	YES	75	C
400-TK-017	Duty	1	PLS STORAGE TANK	$255m^3$	Carbon Steel, rubber lined		NO		C
400-AG-015	Duty	1	PLS STORAGE TANK AGITATOR	11kW	Stainless Steel 316L	1	YES	11	C
400-PU-011	Duty	2	PLS STORAGE TANK PUMP	$180m^3$/hr and 75kW	Stainless Steel 316L	1	YES	75	C
400-PU-012	Duty	1	SULPHURIC ACID LEACHING AREA SUMP PUMP	22kW	Stainless Steel 316L	1	YES	22	C
400-SS-001	Duty	1	SULPHURIC ACID LEACHING AREA SAFETY SHOWER/EYEWASH		Stainless Steel 316L		NO		C
AREA 500	**-**		**ION EXCHANGE - Mo, V & U**						
500-CO-001	Duty	3	Mo ION EXCHANGE COLUMN	$20m^3$	Conductive FRP		NO		A
500-CO-004	Duty	9	U & V ION EXCHANGE COLUMN	$20m^3$	Conductive FRP		NO		A
500-PU-001	Duty	6	Mo ION EXCHANGE COLUMNS DISCHARGE PUMP	$180m^3$/hr and 75kW	Stainless Steel 316L	3	YES	75.0	A
500-PU-007	Duty	18	U & V ION EXCHANGE COLUMNS DISCHARGE PUMP	$180m^3$/hr and 75kW	Stainless Steel 316L	9	YES	75.0	A
500-PU-025	Duty	1	ION EXCHANGE AREA SUMP PUMP	7.5kW	Stainless Steel 316L	1	YES	7.5	A
500-SS-001	Duty	1	ION EXCHANGE AREA SAFETY SHOWER/EYEWASH		Stainless Steel 316L		NO		A
500-HO-001	Duty	1	ION EXCHANGE PLANT MAINTENANCE CRANE	2kW	Carbon Steel, painted.	1	YES	2.0	A
AREA 540	**-**		**Mo RECOVERY**						
540-TK-001	Duty	1	MOLYBDENUM ELUTION LIQUOR STORAGE TANK	$299m^3$	Carbon Steel, rubber lined		NO		C
540-PU-001	Duty	1	MOLYBDENUM ELUTION STORAGE TANK DISCHARGE PUMP	$11m^3$/hr and 4kW	Stainless Steel 316L	1	YES	4.0	C
540-TK-002	Duty	1	MOLYBDENUM PRECIPITATION TANK	$202m^3$	Carbon Steel, lined FRP		NO		C
540-AG-001	Duty	1	MOLYBDENUM PRECIPITATION TANK AGITATOR	7.5kW	Stainless Steel 316L	1	YES	7.5	C
540-PU-002	Duty	1	MOLYBDENUM PRECIPITATION TANK PUMP	$37m^3$/hr and 15kW	Stainless Steel 316L	1	YES	15.0	C
540-TH-001	Duty	1	MOLYBDENUM PRECIPITATION THICKENER	$20m^2$ or 5m diameter,	Carbon Steel	1	YES	1.0	C
540-PU-003	Duty	1	MOLYBDENUM PRECIPITATION THICKENER UNDERFLOW PUMP	$20m^3$/hr and 15kW	Carbon Steel	1	YES	15.0	C
540-VP-001	Duty	1	MOLYBDENUM PRECIPITATION SOLID/LIQUID SEPARATION VENDOR PACKAGE		Stainless Steel 316L	1	YES	25.0	C
540-TK-003	Duty	1	FILTRATE & WASTE WATER STORAGE TANK	$147m^3$	Carbon Steel		NO		C
540-AG-002	Duty	1	FILTRATE & WASTE WATER STORAGE TANK AGITATOR	1.5kW	Carbon Steel	1	YES	1.5	C
540-PU-004	Duty	1	FILTRATE & WASTE WATER STORAGE TANK PUMP	$29m^3$/hr and 11kW	Carbon Steel	1	YES	11.0	C
540-PU-005	Duty	1	MOLYBDENUM RECOVERY AREA SUMP PUMP	7.5Kw	Stainless Steel 316L	1	YES	7.5	C
540-SS-001	Duty	1	MOLYBDENUM RECOVERY AREA SAFETY SHOWER/EYEWASH		Stainless Steel 316L		NO		C
AREA 560	**-**		**U RECOVERY**						
560-TK-001	Duty	1	URANIUM & VANADIUM ELUTION LIQUOR STORAGE TANK	$603m^3$	Carbon Steel, rubber lined		NO		C
560-PU-001	Duty	1	U & V ELUTION LIQUOR STORAGE TANK PUMP	$21m^3$/hr and 7.5kW	Stainless Steel 316L	1	YES	7.5	C
560-TK-002	Duty	1	URANIUM PRECIPITATION TANK	$208m^3$	Carbon Steel, rubber lined		NO		C
560-AG-001	Duty	1	URANIUM PRECIPITATION TANK AGITATOR	7.5kW	Stainless Steel 316L	1	YES	7.5	C
560-PU-002	Duty	1	URANIUM PRECIPITATION TANK PUMP	$21.7m^3$/hr and 7.5kW	Stainless Steel 316L	1	YES	7.5	C
560-TH-001	Duty	1	URANIUM PRECIPITATION THICKENER	$20m^2$ or 5m diameter	Carbon Steel	1	YES	1.0	C
560-PU-003	Duty	1	URANIUM PRECIPITATION THICKENER UNDERFLOW PUMP	$22m^3$/hr and 12kW	Carbon Steel	1	YES	12.0	C
560-VP-001	Duty	1	URANIUM PRECIPITATION SOLID/LIQUID SEPARATION - VENDOR PACKAGE		Stainless Steel 316L	1	YES	25.0	C
560-TK-003	Duty	1	FILTRATE & WASTE WATER TANK	$111.3m^3$	Carbon Steel, rubber lined		NO		C
560-PU-004	Duty	1	FILTRATE & WASTE WATER TANK DISCHARGE PUMP	$21.2m^3$/hr and 7.5kW	Stainless Steel 316L	1	YES	7.5	C
560-CV-001	Duty	1	URANIUM PRODUCT TRANSFER CONVEYOR	1t/hr, 3kW		1	YES	3.0	C
560-VP-002	Duty	1	URANIUM PRODUCT CALCINER & FINAL PRODUCT TREATMENT - VENDOR PKG	10t/hr, 500kW	Stainless Steel 316L	1	YES	500.0	C
560-PU-005	Duty	1	URANIUM RECOVERY AREA SUMP PUMP	1.1kW	Stainless Steel 316L	1	YES	1.1	C
560-SS-001	Duty	1	URANIUM RECOVERY AREA SAFETY SHOWER/EYEWASH		Stainless Steel 316L		NO		C
AREA 580	**-**		**V RECOVERY**						
580-TK-001	Duty	1	VANADIUM IX ELUTION STORAGE TANK	$111m^3$	Carbon Steel, lined FRP		NO		C
580-PU-001	Duty	1	VANADIUM IX ELUTION STORAGE TANK DISCHARGE PUMP	$21.2m^3$/hr and 7.5kW	Stainless Steel 316L	1	YES	7.5	C
580-TK-002	Duty	1	VANADIUM PRECIPITATION TANK	$208m^3$	Carbon Steel, lined FRP		NO		C
580-AG-001	Duty	1	VANADIUM PRECIPITATION TANK AGITATOR	15kW	Stainless Steel 316L	1	YES	15.0	C
580-PU-002	Duty	1	VANADIUM PRECIPITATION TANK PUMP	$21.8m^3$/hr and 7.5kW	Stainless Steel 316L	1	YES	7.5	C
580-TH-001	Duty	1	VANADIUM PRECIPITATION THICKENER	$20m^2$ or 5m diameter	Carbon Steel	1	YES	3.0	C
580-PU-003	Duty	1	VANADIUM PRECIPITATION THICKENER UNDERFLOW PUMP	$22m^3$/hr and 12kW	Carbon Steel	1	YES	12.0	C
580-VP-001	Duty	1	VANADIUM PRECIPITATION SOLID/LIQUID SEPARATION - VENDOR PACKAGE		Stainless Steel 316L	1	YES	500.0	C
580-TK-003	Duty	1	VANADIUM PRECIPITATION THICKENER OVERFLOW STORAGE TANK (WWST 2)	$147m^3$	Carbon Steel		NO		C
580-PU-004	Duty	1	VANADIUM RECOVERY AREA SUMP PUMP	1.1kW	Stainless Steel 316L	1	YES	1.1	C
580-SS-001	Duty	1	VANADIUM RECOVERY AREA SAFETY SHOWER/EYEWASH		Stainless Steel 316L		NO		C

MECHANICAL EQUIPMENT LIST

Tag	Duty / Standby	Quantity	Equipment title	Capacity	Materials Of Construction	No of Drives	Variable speed drive	kW Installed	Rev
AREA 600	**-**		**SOLVENT EXTRACTION - Phosphoric Acid & Rare Earths**						
600-TK-001	Duty	1	MIXED ORGANIC SUPPLY TANK 1	226m^3	Stainless Steel 316L		NO		C
600-AG-001	Duty	1	MIXED ORGANIC SUPPLY TANK AGITATOR	4kW	Stainless Steel 316L	1	YES	4.0	C
600-PU-001	Duty	2	MIXED ORGANIC SUPPLY TANK PUMP	180m^3/hr and 75kW	Stainless Steel 316L	1	YES	75.0	C
600-TK-002	Duty	1	PLS SUPPLY TANK	269m^3	Stainless Steel 316L		NO		C
600-AG-002	Duty	1	PLS SUPPLY TANK AGITATOR	7.5kW	Stainless Steel 316L	1	YES	7.5	C
600-PU-003	Duty	2	PLS SUPPLY TANK PUMP	180m^3/hr and 75kW	Stainless Steel 316L	1	YES	75.0	C
600-MS-001	Duty	15	MIXER SETTLER	12 m^3 live capacity + settler capacity	FRP + Concrete FRP Lined		NO		C
600-AG-003	Duty	15	MIXER TANK AGITATOR	1kW	Stainless Steel 316L	15	YES	1.0	C
600-TK-004	Duty	1	18% PHOSPHORIC ACID TANK	423m^3	Carbon Steel, rubber lined.		NO		C
600-PU-005	Duty	1	18% PHOSPHORIC ACID TANK PUMP	46m^3/hr and 18.5kW	Carbon Steel, PTFE	1	YES	18.5	C
600-TK-009	Duty	1	WASTE WATER STORAGE TANK 3	202m^3	Stainless Steel 316L		NO		C
600-AG-018	Duty	1	WASTE WATER STORAGE TANK AGITATOR	2.2kW	Stainless Steel 316L	1	YES	2.2	C
600-PU-006	Duty	1	WASTE WATER STORAGE TANK PUMP	37.1m^3/hr and 10kW	Stainless Steel 316L	1	YES	10.0	C
600-VP-001	Duty	1	PHOSPHORIC ACID UPGRADE EVAPORATOR - VENDOR PACKAGE	500kW	Stainless Steel 316L	1	YES	500.0	C
600-PU-007	Duty	1	PHOSPHORIC ACID UPGRADE EVAPORATOR DISCHARGE PUMP	9.1m^3/hr and 4kW	Stainless Steel 316L	1	YES	4.0	C
600-TK-010	Duty	1	CONCENTRATED PHOSPHORIC ACID STORAGE TANK	1817m^3	Carbon Steel, rubber lined.		NO		C
600-PU-008	Duty	1	CONCENTRATED PHOSPHORIC ACID STORAGE TANK PUMP	10.9m^3/hr and 18.5kW	Stainless Steel 316L	1	YES	18.5	C
600-PU-009	Duty	1	SX AREA SUMP PUMP	7.5kW	Stainless Steel 316/316L wetted parts	1	YES	7.5	C
600-PU-010	Duty	2	MIXED ORGANIC & PHOSPHORIC ACID AREA SUMP PUMP	7.5kW	Stainless Steel 316/316L wetted parts	1	YES	7.5	C
600-SS-001	Duty	4	SOLVENT EXTRACTION (PA & RE) AREA SAFETY SHOWER/EYEWASH		Stainless Steel 316L		NO		C
AREA 700	**-**		**SOLVENT EXTRACTION - Mn, Ni & Zn**						
700-TK-001	Duty	1	MIXED ORGANIC SUPPLY TANK 2	226m^3	Stainless Steel 316L		NO		C
700-AG-001	Duty	1	MIXED ORGANIC SUPPLY TANK AGITATOR	5.5kW	Stainless Steel 316L	1	YES	5.5	C
700-PU-001	Duty	2	MIXED ORGANIC SUPPLY TANK PUMP	180m^3/hr and 75kW	Stainless Steel 316L	1	YES	75.0	C
700-MS-001	Duty	15	MIXER SETTLER	12 m^3 live capacity + settler capacity	FRP + Concrete FRP Lined		NO		C
700-AG-002	Duty	15	MIXER TANK AGITATOR	1kW	Stainless Steel 316L	15	YES	1.0	C
700-TK-002	Duty	1	WASTE WATER (RAFFINATE) STORAGE TANK 4	859m^3	Carbon Steel		NO		C
700-PU-003	Duty	2	WASTE WATER (RAFFINATE) STORAGE TANK DISCHARGE PUMP	175m^3/hr and 75kW	Stainless Steel 316L	1	YES	75.0	C
700-PU-005	Duty	1	SX AREA SUMP PUMP	7.5kW	Stainless Steel 316/316L wetted parts	1	YES	7.5	C
700-PU-006	Duty	1	MIXED ORGANIC AREA SUMP PUMP	7.5kW	Stainless Steel 316/316L wetted parts	1	YES	7.5	C
700-SS-001	Duty	3	SOLVENT EXTRACTION (PA & RE) AREA SAFETY SHOWER/EYEWASH		Stainless Steel		NO		C
AREA 830	**-**		**RARE EARTH RECOVERY**						
830-TK-001	Duty	1	RARE EARTH PRECIPITATION TANK	454m^3	Stainless Steel 316L		NO		C
830-AG-001	Duty	1	RARE EARTH PRECIPITATION TANK AGITATOR	18.5kW	Stainless Steel 316L	1	YES	18.5	C
830-PU-001	Duty	1	RARE EARTH PRECIPITATION TANK PUMP	50m^3/hr and 7kW	Stainless Steel 316L	1	YES	7.0	C
830-VP-001	Duty	1	RARE EARTH PRECIPITATION SOLID/LIQUID SEPARATION VENDOR PACKAGE	500kW	Stainless Steel 316L	1	YES	500.0	C
700-TK-002	Duty	1	CENTRIFUGE FILTRATE TANK	423m^3	Carbon Steel		NO		C
700-AG-002	Duty	1	CENTRIFUGE FILTRATE TANK AGITATOR	15kW	Carbon Steel	1	YES	15.0	C
700-PU-002	Duty	1	CENTRIFUGE FILTRATE TANK PUMP	180m^3/hr and 75kW	Carbon Steel	1	YES	75.0	C
830-PU-003	Duty	1	RARE EARTH RECOVERY AREA SUMP PUMP	7.5kW	Stainless Steel 316/316L wetted parts	1	YES	7.5	C
830-SS-001	Duty	1	RARE EARTH RECOVERY AREA SAFETY SHOWER/EYEWASH		Stainless Steel 316L		NO		C
AREA 860	**-**		**MANGANESE, NICKEL & ZINC RECOVERY**						
860-TK-001	Duty	1	MANGANESE, NICKEL & ZINC CARBONATE PRECIPITATION TANK	58m^3	Carbon Steel		NO		C
860-AG-001	Duty	1	MN, NICKEL & ZINC CARBONATE PRECIPITATION TANK AGITATOR	3.0kW	Carbon Steel	1	YES	3.0	C
860-PU-001	Duty	1	MN, NICKEL & ZINC CARBONATE PRECIPITATION TANK PUMP	24.5m^3/hr and 11kW	Carbon Steel	1	YES	11.0	C
860-VP-001	Duty	1	MN, NICKEL & ZINC CARBONATE PRECIPITATION SOLID/LIQUID SEPARATION VENDOR PACKAGE	500kW		1	YES	500.0	C
860-TK-002	Duty	1	WASTE WATER STORAGE TANK 5	111m^3	Carbon Steel		NO		C
860-AG-002	Duty	1	WASTE WATER STORAGE TANK AGITATOR	1.5kW	Carbon Steel	1	YES	1.5	C
860-PU-002	Duty	1	WASTE WATER STORAGE TANK PUMP	22.3m^3/hr and 11kW	Carbon Steel	1	YES	11.0	C
860-PU-003	Duty	1	MN, NICKEL & ZINC RECOVERY AREA SUMP PUMP	1.1kW	316/316L wetted parts	1	YES	1.1	C
860-SS-001	Duty	1	MN, NICKEL & ZINC RECOVERY AREA SAFETY SHOWER/EYEWASH		Stainless Steel 316L		NO		C

MECHANICAL EQUIPMENT LIST

Tag	Duty / Standby	Quantity	Equipment title	Capacity	Materials Of Construction	No of Drives	Variable speed drive	kW Installed	Rev
AREA 900	**-**		**REAGENTS**						
900-TK-001	Duty	1	ACETIC ACID (GLACIAL) SUPPLY TANK	1279m^3	Stainless Steel 316L		NO		C
900-PU-001	Duty	2	ACETIC ACID (GLACIAL) SUPPLY TANK PUMP	6.67m^3/hr and 3.0kW	Stainless Steel 316L	1	YES	3.0	C
900-SI-001	Duty	1	AMMONIUM CARBONATE SILO	50m^3	Stainless Steel 316L		NO		C
900-CV-001	Duty	1	AMMONIUM CARBONATE SCREW CONVEYOR AND WEIGHTOMETER	2kW	Stainless Steel 316L	1	YES	2.0	C
900-TK-002	Duty	1	AMMONIUM CARBONATE 50% SLURRY MIXING TANK	111m^3	Stainless Steel 316L		NO		C
900-AG-001	Duty	1	AMMONIUM CARBONATE SLURRY TANK AGITATOR	5.5kW	Stainless Steel 316L	1	YES	5.5	C
900-PU-003	Duty	2	AMMONIUM CARBONATE SLURRY TANK PUMP	1.7m^3/hr and 1.1kW	Stainless Steel 316L	1	YES	1.1	C
900-TK-003	Duty	1	AMMONIUM HYDROXIDE (60%) STORAGE TANK	891m^3	Carbon Steel		NO		C
900-PU-005	Duty	4	AMMONIUM HYDROXIDE (60%) STORAGE TANK PUMP	3.1m^3/hr and 2.2kW	Carbon Steel	4	YES	2.2	C
900-PU-009	Duty	2	AMMONIUM HYDROXIDE (60%) STORAGE TANK PUMP	2.1m^3/hr and 1.1kW	Carbon Steel	1	YES	1.1	C
900-TK-004	Duty	1	AMMONIUM HYDROXIDE (4%) STORAGE TANK	137m^3	Carbon Steel		NO		C
900-PU-011	Duty	4	AMMONIUM HYDROXIDE (4%) STORAGE TANK PUMP	2.3m^3/hr and 2.2kW	Carbon Steel	4	YES	2.2	C
900-TK-005	Duty	1	DILUENT STORAGE (SHELL SOL) TANK	53m^3	Carbon Steel		NO		C
900-PU-015	Duty	1	DILUENT STORAGE TANK PUMP	0.3m^3/hr and 1.1kW	Carbon Steel	1	YES	1.1	C
900-TK-006	Duty	1	FERRIC SULPHATE LIQUOR (60%) SUPPLY TANK	119m^3	Stainless Steel 316L		NO		C
900-AG-002	Duty	1	FERRIC SULPHATE LIQUOR (60%) SUPPLY TANK AGITATOR	5.5kW	Stainless Steel 316L	1	YES	5.5	C
900-PU-016	Duty	2	FERRIC SULPHATE LIQUOR (60%) SUPPLY TANK PUMP	2m^3/hr and 1.1kW	Stainless Steel 316L	1	YES	1.1	C
900-TK-007	Duty	1	HYDRATED LIME (40%) STORAGE TANK	81.7m^3	Carbon Steel, rubber lined		NO		C
900-PU-018	Duty	2	HYDRATED LIME (40%) STORAGE TANK PUMP	2.8m^3/hr and 1.1kW	Carbon Steel, rubber lined	1	YES	1.1	C
900-SI-002	Duty	1	HYDRATED LIME (HIGH PURITY) SILO	100m^3	Carbon Steel		NO		C
900-CV-002	Duty	1	HYDRATED LIME (HIGH PURITY) SCREW CONVEYOR AND WEIGHTOMETER	4kW	Carbon Steel	1	YES	4.0	C
900-AG-003	Duty	1	HYDRATED LIME (HIGH PURITY) (20%) STORAGE TANK AGITATOR	3kW	Carbon Steel, rubber lined	1	YES	3.0	C
900-TK-009	Duty	1	HYDRATED LIME (HIGH PURITY) (20%) STORAGE TANK	81.7m^3	Carbon Steel, rubber lined		NO		C
900-PU-020	Duty	2	HYDRATED LIME (HIGH PURITY) (20%) STORAGE TANK PUMP	2.9m^3/hr and 1.1kW	Carbon Steel, rubber lined	1	YES	1.1	C
900-TK-010	Duty	1	HYDROGEN PEROXIDE (35%) STORAGE TANK	11m^3	Stainless Steel 316L		NO		C
900-PU-022	Duty	2	HYDROGEN PEROXIDE (35%) STORAGE TANK PUMP	0.05m^3/hr and 1.1kW	Stainless Steel 316L	1	YES	1.1	C
900-TK-011	Duty	1	ORGANIC SOLVENT STORAGE (TBP) TANK	234m^3	Stainless Steel 316L		NO		C
900-PU-024	Duty	1	ORGANIC SOLVENT STORAGE (TBP) TANK PUMP	0.05m^3/hr and 1.1kW	Stainless Steel 316L	1	YES	1.1	C
900-TK-012	Duty	1	ORGANIC SOLVENT STORAGE (D2EHPA) TANK	234m^3	Stainless Steel 316L		NO		C
900-PU-025	Duty	1	ORGANIC SOLVENT STORAGE (D2EHPA) TANK PUMP	0.1m^3/hr and 1.1kW	Stainless Steel 316L	1	YES	1.1	C
900-SI-003	Duty	1	PYROLUSITE SILO	100m^3	Carbon Steel		NO		C
900-CV-003	Duty	1	PYROLUSITE SCREW CONVEYOR AND WEIGHTOMETER	5kW	Carbon Steel	1	YES	5.0	C
900-TK-013	Duty	1	PYROLUSITE 50% SLURRY MIXING TANK	75.4m^3	Carbon Steel		NO		C
900-AG-004	Duty	1	PYROLUSITE SLURRY TANK AGITATOR	4kW	Carbon Steel	1	YES	4.0	C
900-PU-026	Duty	2	PYROLUSITE SLURRY TANK PUMP	0.3m^3/hr and 1.1kW	Carbon Steel	1	YES	1.1	C
900-BN-001	Duty	1	PYRITE STORAGE BIN	1000m^3	Carbon Steel		NO		C
900-VP-001	Duty	1	PYRITE BURNING PLANT - VENDOR PACKAGE	500kW	Stainless Steel 316L	1	YES	500.0	C
900-BN-001	Duty	1	SULFUR STORAGE BIN	1000m^3	Carbon Steel		NO		C
900-VP-002	Duty	1	SULPHURIC ACID PLANT - VENDOR PACKAGE	500kW	Stainless Steel 316L	1	YES	500.0	C
900-TK-014	Duty	3	SULPHURIC ACID (98%) STORAGE TANK	1218m^3	Carbon Steel		NO		C
900-PU-028	Duty	2	SULPHURIC ACID (98%) STORAGE TANK PUMP	27m^3/hr and 15kW	Carbon Steel	2	YES	15.0	C
900-PU-030	Duty	2	SULPHURIC ACID (98%) STORAGE TANK PUMP	10.8m^3/hr and 5.5kW	Carbon Steel	1	YES	5.5	C
900-TK-017	Duty	6	SULPHURIC ACID (10%) MIXING TANK	891m^3	Stainless Steel 316L		NO		C
900-AG-005	Duty	6	SULPHURIC ACID (10%) MIXING TANK AGITATOR	7.5kW	Stainless Steel 316L	6	YES	7.5	C
900-PU-032	Duty	8	SULPHURIC ACID (10%) MIXING TANK PUMP	90.9m^3/hr and 37kW	Stainless Steel 316L	6	YES	37.0	C
900-BN-002	Duty	1	WASHED SCRAP IRON BIN	200m^3	Carbon Steel		NO		C
900-HO-001	Duty	2	REAGENT HOIST	1.6 t	Carbon Steel, painted to spec.		NO		C
900-PU-040	Duty	2	REAGENTS AREA SUMP PUMP	37kW	316/316L wetted parts	2	YES	37.0	C
900-SS-001	Duty	2	REAGENTS AREA SAFETY SHOWER/EYEWASH		Stainless Steel 316L		NO		C
AREA 1000	**-**		**WASTE WATER TREATMENT**						
AREA 1100	**-**		**UTILITIES & SERVICES**						
AREA 1200	**-**		**BUILDINGS**						

U3O8 CORP. PRELIMINARY ECONOMIC ASSESSMENT ON THE BERLIN DEPOSIT, COLOMBIA

Report Prepared for

U3O8 Corp.

APPENDIX D: OPTION B PROCESS PLANT MECHANICAL EQUIPMENT LIST

Report Prepared by

Tenova Mining & Minerals (Australia) Pty Ltd

M6088.A

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MINING & MINERALS

The Uranium Discovery Company
U3O8 CORP

U3O8 CORPORATION LIMITED

BERLIN PROJECT - OPTION B

PEA STUDY

MECHANICAL EQUIPMENT LIST - 1 MTPA

M6088.A-M810-002

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0	25/01/2013	ISSUED FOR USE	J.SCHLOFFER	P.NIEMANN	L.DE KLERK	A.WALKER
B	11/01/2013	ISSUED FOR ICR	J.SCHLOFFER	P.NIEMANN	L.DE KLERK	
A	18/12/2012	ISSUED FOR IDR	J.SCHLOFFER	P.NIEMANN		
REV	**DATE**	**REVISION HISTORY**	**ORIGINATED**	**CHECKED**	**APPROVED**	**TENOVA APPROVAL**

PLANT AREA DESCRIPTION

AREA NUMBER	AREA DESCRIPTION
100	CRUSHING
200	GRINDING
400	SULPHURIC ACID LEACHING
500	ION EXCHANGE - Mo, V & U
540	Mo RECOVERY
560	U RECOVERY
580	V RECOVERY
600	SOLVENT EXTRACTION - Phosphoric Acid & Rare Earths
700	SOLVENT EXTRACTION - Mn, Ni & Zn
800	PHOSPHORIC ACID CONCENTRATION
830	RARE EARTH RECOVERY
860	MANGANESE, NICKEL & ZINC RECOVERY
1100	UTILITIES & SERVICES
1000	WASTE WATER TREATMENT
1100	UTILITIES & SERVICES
1200	BUILDINGS

MECHANICAL EQUIPMENT LIST

Tag	Duty / Standby	Quantity	Equipment title	Capacity	Materials Of Construction	No of Drives	Variable speed drive	kW Installed	Rev
AREA 100	**-**		**CRUSHING**						
100-FL-001	Duty	1	FRONT END LOADER	5t			NO		A
100-SC-001	Duty	1	STATIC SCALPING SCREEN	175t/hr	Carbon Steel		NO		A
100-DT-001	Duty	1	METAL DETECTOR	11.0kW		1	NO	11.0	A
100-SC-002	Duty	1	VIBRATING SCALPING SCREEN	175t/hr, 30kW	Carbon Steel	1	YES	30.0	A
100-CR-001	Duty	1	JAW CRUSHER	175t/hr, 84.84kW	Carbon Steel/cast Manganese Steel	1	YES	84.84	A
100-CV-001	Duty	1	JAW CRUSHER DISCHARGE PRODUCT CONVEYOR	175t/hr		1	YES	2.0	A
100-CV-002	Duty	1	JAW CRUSHER DISCHARGE SPILLAGE CONVEYOR			1	YES	5.0	A
100-PU-001	Duty	2	DUST SUPPRESSION WATER PUMP	1.1kW	Carbon Steel	2	YES	1.1	A
100-PU-003	Duty	1	CRUSHING AREA SUMP PUMP	3kW	Carbon Steel	1	YES	3.0	A
100-SS-001	Duty	1	CRUSHING AREA SAFETY SHOWER/EYEWASH		Stainless Steel 316L		NO		A
AREA 140	**-**		**STOCKPILING**						
140-AF-001	Duty	2	APRON FEEDER	175t/hr	Carbon Steel/cast Manganese Steel	2	YES	55.0	A
140-CV-001	Duty	1	SAG MILL FEED CONVEYOR	135t/hr, 5.0kW		1	YES	5.0	A
140-SR-001	Duty	1	STOCKPILE AREA DUST SCRUBBER	30kW	Carbon Steel	1	YES	30.0	A
AREA 200	**-**		**GRINDING**						
200-ML-001	Duty	1	SAG MILL	135t/hr	Carbon Steel/ Cr-Mo	1	YES	500.0	A
200-FD-001	Duty	1	SAG MILL DISCHARGE SCREEN VIBRATING FEEDER	305t/hr	Carbon Steel / Bisalloy liners	1	YES	5.0	A
200-SC-001	Duty	1	SAG MILL DISCHARGE SCREEN	305t/hr	Carbon Steel, rubber screens	1	YES	10.0	A
200-CV-001	Duty	1	SAG MILL RECYCLE CONVEYOR	202t/hr, 30kW		1	YES	30.0	A
200-CV-002	Duty	2	SAG MILL RECYCLE CONVEYOR	202t/hr, 30kW		1	YES	30.0	A
200-PU-001	Duty	2	BALL MILL HOPPER PUMP	436m^3/hr, 220kW	Carbon Steel	1	YES	220.0	A
200-CY-001	Duty	3	CYCLONE				NO		A
200-ML-002	Duty	1	BALL MILL		Carbon Steel	1	YES	2500.0	A
200-CR-001	Duty	1	PEBBLE CRUSHER	30t/hr (*to be confirmed by lead engineer)	Carbon Steel	1	YES	20.0	A
200-CV-004	Duty	1	PEBBLE CRUSHER TO SAG MILL RECYCLE CONVEYOR			1	YES	5.0	A
200-CN-001	Duty	2	MILL CRANE	15t	Carbon Steel	1	NO	11.0	A
200-PU-003	Duty	2	DUST SUPPRESSION WATER PUMP	1.1kW	Carbon Steel	1	YES	1.1	A
200-SR-001	Duty	1	GRINDING AREA DUST SCRUBBER	5kW	Carbon Steel	1	YES	5.0	A
200-PU-005	Duty	1	GRINDING AREA SUMP PUMP	22kW	Carbon Steel	1	YES	22.0	A
200-SS-001	Duty	1	GRINDING AREA SAFETY SHOWER/EYEWASH		Stainless Steel 316L		NO		A
AREA 400	**-**		**SULPHURIC ACID LEACHING**						
400-TK-001	Duty	14	LEACHING TANK	394m^3	Carbon Steel, rubber lined		NO		A
400-AG-001	Duty	14	LEACHING TANK AGITATOR	22kW	Carbon Steel, rubber lined	14	YES	22	A
400-PU-001	Duty	4	LEACHING TANK SLURRY DISCHARGE PUMP	340m^3/hr and 132kW	Stainless Steel 316L	2	YES	132	A
400-TH-001	Duty	3	CCD WASHING THICKENER	415m^2 or 23m diameter	Carbon Steel	3	YES	3	A
400-PU-005	Duty	4	CCD WASHING THICKENER UNDERFLOW PUMP	208m^3/hr and 110kW	Stainless Steel 316L	2	YES	110	A
400-PU-009	Duty	1	PINNED BED CLARIFIER PUMP	245m^3/hr and 75kW	Stainless Steel 316L	1	YES	75	A
400-CL-001	Duty	1	PINNED BED CLARIFIER	300m^3	Carbon Steel, rubber lined		NO		A
400-TK-015	Duty	2	IRON REDUCTION TANK	14m^3	Carbon Steel, rubber lined		NO		A
400-PU-010	Duty	1	IRON REDUCTION TANK PUMP	245m^3/hr and 75kW	Stainless Steel 316L	1	YES	75	A
400-TK-017	Duty	1	PLS STORAGE TANK	348m^3	Carbon Steel, rubber lined		NO		A
400-AG-015	Duty	1	PLS STORAGE TANK AGITATOR	11kW	Stainless Steel 316L	1	YES	11	A
400-PU-011	Duty	2	PLS STORAGE TANK PUMP	245m^3/hr and 75kW	Stainless Steel 316L	1	YES	75	A
400-PU-012	Duty	1	SULPHURIC ACID LEACHING AREA SUMP PUMP	22kW	Stainless Steel 316L	1	YES	22	A
400-SS-001	Duty	1	SULPHURIC ACID LEACHING AREA SAFETY SHOWER/EYEWASH		Stainless Steel 316L		NO		A
AREA 500	**-**		**ION EXCHANGE - Mo, V & U**						
500-CO-001	Duty	3	Mo ION EXCHANGE COLUMN	20m^3	Conductive FRP		NO		A
500-CO-004	Duty	9	U & V ION EXCHANGE COLUMN	20m^3	Conductive FRP		NO		A
500-PU-001	Duty	6	Mo ION EXCHANGE COLUMNS DISCHARGE PUMP	245m^3/hr and 75kW	Stainless Steel 316L	3	YES	75.0	A
500-PU-007	Duty	18	U & V ION EXCHANGE COLUMNS DISCHARGE PUMP	245m^3/hr and 75kW	Stainless Steel 316L	9	YES	75.0	A
500-PU-025	Duty	1	ION EXCHANGE AREA SUMP PUMP	7.5kW	Stainless Steel 316L	1	YES	7.5	A
500-SS-001	Duty	1	ION EXCHANGE AREA SAFETY SHOWER/EYEWASH		Stainless Steel 316L		NO		A
500-HO-001	Duty	1	ION EXCHANGE PLANT MAINTENANCE CRANE	2kW	Carbon Steel, painted.	1	YES	2.0	A
AREA 540	**-**		**Mo RECOVERY**						
540-TK-001	Duty	1	MOLYBDENUM ELUTION LIQUOR STORAGE TANK	299m^3	Carbon Steel, rubber lined		NO		A
540-PU-001	Duty	1	MOLYBDENUM ELUTION STORAGE TANK DISCHARGE PUMP	11m^3/hr and 4kW	Stainless Steel 316L	1	YES	4.0	A
540-TK-002	Duty	1	MOLYBDENUM PRECIPITATION TANK	202m^3	Carbon Steel, lined FRP		NO		A
540-AG-001	Duty	1	MOLYBDENUM PRECIPITATION TANK AGITATOR	7.5kW	Stainless Steel 316L	1	YES	7.5	A
540-PU-002	Duty	1	MOLYBDENUM PRECIPITATION TANK PUMP	37m^3/hr and 15kW	Stainless Steel 316L	1	YES	15.0	A
540-TH-001	Duty	1	MOLYBDENUM PRECIPITATION THICKENER	20m^2 or 5m diameter,	Carbon Steel	1	YES	1.0	A
540-PU-003	Duty	1	MOLYBDENUM PRECIPITATION THICKENER UNDERFLOW PUMP	20m^3/hr and 15kW	Carbon Steel	1	YES	15.0	A
540-VP-001	Duty	1	MOLYBDENUM PRECIPITATION SOLID/LIQUID SEPARATION VENDOR PACKAGE		Stainless Steel 316L	1	YES	25.0	A
540-TK-003	Duty	1	FILTRATE & WASTE WATER STORAGE TANK	147m^3	Carbon Steel		NO		A
540-AG-002	Duty	1	FILTRATE & WASTE WATER STORAGE TANK AGITATOR	1.5kW	Carbon Steel	1	YES	1.5	A
540-PU-004	Duty	1	FILTRATE & WASTE WATER STORAGE TANK PUMP	29m^3/hr and 11kW	Carbon Steel	1	YES	11.0	A
540-PU-005	Duty	1	MOLYBDENUM RECOVERY AREA SUMP PUMP	7.5Kw	Stainless Steel 316L	1	YES	7.5	A
540-SS-001	Duty	1	MOLYBDENUM RECOVERY AREA SAFETY SHOWER/EYEWASH		Stainless Steel 316L		NO		A

MECHANICAL EQUIPMENT LIST

Tag	Duty / Standby	Quantity	Equipment title	Capacity	Materials Of Construction	No of Drives	Variable speed drive	kW Installed	Rev
AREA 560	-		**U RECOVERY**						
560-TK-001	Duty	1	URANIUM & VANADIUM ELUTION LIQUOR STORAGE TANK	603m^3	Carbon Steel, rubber lined		NO		A
560-PU-001	Duty	1	U & V ELUTION LIQUOR STORAGE TANK PUMP	21m^3/hr and 7.5kW	Stainless Steel 316L	1	YES	7.5	A
560-TK-002	Duty	1	URANIUM PRECIPITATION TANK	208m^3	Carbon Steel, rubber lined		NO		A
560-AG-001	Duty	1	URANIUM PRECIPITATION TANK AGITATOR	7.5kW	Stainless Steel 316L	1	YES	7.5	A
560-PU-002	Duty	1	URANIUM PRECIPITATION TANK PUMP	21.7m^3/hr and 7.5kW	Stainless Steel 316L	1	YES	7.5	A
560-TH-001	Duty	1	URANIUM PRECIPITATION THICKENER	20m^2 or 5m diameter	Carbon Steel	1	YES	1.0	A
560-PU-003	Duty	1	URANIUM PRECIPITATION THICKENER UNDERFLOW PUMP	22m^3/hr and 12kW	Carbon Steel	1	YES	12.0	A
560-VP-001	Duty	1	URANIUM PRECIPITATION SOLID/LIQUID SEPARATION - VENDOR PACKAGE		Stainless Steel 316L	1	YES	25.0	A
560-TK-003	Duty	1	FILTRATE & WASTE WATER TANK	111.3m^3	Carbon Steel, rubber lined		NO		A
560-PU-004	Duty	1	FILTRATE & WASTE WATER TANK DISCHARGE PUMP	21.2m^3/hr and 7.5kW	Stainless Steel 316L	1	YES	7.5	A
560-CV-001	Duty	1	URANIUM PRODUCT TRANSFER CONVEYOR	1t/hr, 3kW		1	YES	3.0	A
560-VP-002	Duty	1	URANIUM PRODUCT CALCINER & FINAL PRODUCT TREATMENT - VENDOR PKG	10t/hr, 500kW	Stainless Steel 316L	1	YES	500.0	A
560-PU-005	Duty	1	URANIUM RECOVERY AREA SUMP PUMP	1.1kW	Stainless Steel 316L	1	YES	1.1	A
560-SS-001	Duty	1	URANIUM RECOVERY AREA SAFETY SHOWER/EYEWASH		Stainless Steel 316L		NO		A
AREA 580	-		**V RECOVERY**						
580-TK-001	Duty	1	VANADIUM IX ELUTION STORAGE TANK	111m^3	Carbon Steel, lined FRP		NO		A
580-PU-001	Duty	1	VANADIUM IX ELUTION STORAGE TANK DISCHARGE PUMP	21.2m^3/hr and 7.5kW	Stainless Steel 316L	1	YES	7.5	A
580-TK-002	Duty	1	VANADIUM PRECIPITATION TANK	208m^3	Carbon Steel, lined FRP		NO		A
580-AG-001	Duty	1	VANADIUM PRECIPITATION TANK AGITATOR	15kW	Stainless Steel 316L	1	YES	15.0	A
580-PU-002	Duty	1	VANADIUM PRECIPITATION TANK PUMP	21.8m^3/hr and 7.5kW	Stainless Steel 316L	1	YES	7.5	A
580-TH-001	Duty	1	VANADIUM PRECIPITATION THICKENER	20m^2 or 5m diameter	Carbon Steel	1	YES	3.0	A
580-PU-003	Duty	1	VANADIUM PRECIPITATION THICKENER UNDERFLOW PUMP	22m^3/hr and 12kW	Carbon Steel	1	YES	12.0	A
580-VP-001	Duty	1	VANADIUM PRECIPITATION SOLID/LIQUID SEPARATION - VENDOR PACKAGE		Stainless Steel 316L	1	YES	500.0	A
580-TK-003	Duty	1	VANADIUM PRECIPITATION THICKENER OVERFLOW STORAGE TANK (WWST 2)	147m^3	Carbon Steel		NO		A
580-PU-004	Duty	1	VANADIUM RECOVERY AREA SUMP PUMP	1.1kW	Stainless Steel 316L	1	YES	1.1	A
580-SS-001	Duty	1	VANADIUM RECOVERY AREA SAFETY SHOWER/EYEWASH		Stainless Steel 316L		NO		A
AREA 600	-		**SOLVENT EXTRACTION - Phosphoric Acid & Rare Earths**						
600-TK-001	Duty	1	MIXED ORGANIC SUPPLY TANK 1	307m^3	Stainless Steel 316L		NO		A
600-AG-001	Duty	1	MIXED ORGANIC SUPPLY TANK AGITATOR	7.5kW	Stainless Steel 316L	1	YES	7.5	A
600-PU-001	Duty	2	MIXED ORGANIC SUPPLY TANK PUMP	240m^3/hr and 75kW	Stainless Steel 316L	1	YES	75.0	A
600-TK-002	Duty	1	PLS SUPPLY TANK	367m^3	Stainless Steel 316L		NO		A
600-AG-002	Duty	1	PLS SUPPLY TANK AGITATOR	11kW	Stainless Steel 316L	1	YES	11.0	A
600-PU-003	Duty	2	PLS SUPPLY TANK PUMP	245m^3/hr and 90kW	Stainless Steel 316L	1	YES	90.0	A
600-MS-001	Duty	15	MIXER SETTLER	12 m^3 live capacity + settler capacity	FRP + Concrete FRP Lined		NO		A
600-AG-003	Duty	15	MIXER TANK AGITATOR	1kW	Stainless Steel 316L	15	YES	1.0	A
600-TK-004	Duty	1	18% PHOSPHORIC ACID TANK	423m^3	Carbon Steel, rubber lined.		NO		A
600-PU-005	Duty	1	18% PHOSPHORIC ACID TANK PUMP	46m^3/hr and 18.5kW	Carbon Steel, PTFE	1	YES	18.5	A
600-TK-009	Duty	1	WASTE WATER STORAGE TANK 3	202m^3	Stainless Steel 316L		NO		A
600-AG-018	Duty	1	WASTE WATER STORAGE TANK AGITATOR	2.2kW	Stainless Steel 316L	1	YES	2.2	A
600-PU-006	Duty	1	WASTE WATER STORAGE TANK PUMP	37.1m^3/hr and 10kW	Stainless Steel 316L	1	YES	10.0	A
600-VP-001	Duty	1	PHOSPHORIC ACID UPGRADE EVAPORATOR - VENDOR PACKAGE	500kW	Stainless Steel 316L	1	YES	500.0	A
600-PU-007	Duty	1	PHOSPHORIC ACID UPGRADE EVAPORATOR DISCHARGE PUMP	9.1m^3/hr and 4kW	Stainless Steel 316L	1	YES	4.0	A
600-TK-010	Duty	1	CONCENTRATED PHOSPHORIC ACID STORAGE TANK	1817m^3	Carbon Steel, rubber lined.		NO		A
600-PU-008	Duty	1	CONCENTRATED PHOSPHORIC ACID STORAGE TANK PUMP	10.9m^3/hr and 18.5kW	Stainless Steel 316L	1	YES	18.5	A
600-PU-009	Duty	1	SX AREA SUMP PUMP	7.5kW	Stainless Steel 316/316L wetted parts	1	YES	7.5	A
600-PU-010	Duty	2	MIXED ORGANIC & PHOSPHORIC ACID AREA SUMP PUMP	7.5kW	Stainless Steel 316/316L wetted parts	1	YES	7.5	A
600-SS-001	Duty	4	SOLVENT EXTRACTION (PA & RE) AREA SAFETY SHOWER/EYEWASH		Stainless Steel 316L		NO		A
AREA 700	-		**SOLVENT EXTRACTION - Mn, Ni & Zn**						
700-TK-001	Duty	1	MIXED ORGANIC SUPPLY TANK 2	307m^3	Stainless Steel 316L		NO		A
700-AG-001	Duty	1	MIXED ORGANIC SUPPLY TANK AGITATOR	7.5kW	Stainless Steel 316L	1	YES	7.5	A
700-PU-001	Duty	2	MIXED ORGANIC SUPPLY TANK PUMP	240m^3/hr and 90kW	Stainless Steel 316L	1	YES	90.0	A
700-MS-001	Duty	15	MIXER SETTLER	12 m^3 live capacity + settler capacity	FRP + Concrete FRP Lined		NO		A
700-AG-002	Duty	15	MIXER TANK AGITATOR	1kW	Stainless Steel 316L	15	YES	1.0	A
700-TK-002	Duty	1	WASTE WATER (RAFFINATE) STORAGE TANK 4	1120m^3	Carbon Steel		NO		A
700-PU-003	Duty	2	WASTE WATER (RAFFINATE) STORAGE TANK DISCHARGE PUMP	243m^3/hr and 90kW	Stainless Steel 316L	1	YES	90.0	A
700-PU-005	Duty	1	SX AREA SUMP PUMP	7.5kW	Stainless Steel 316/316L wetted parts	1	YES	7.5	A
700-PU-006	Duty	1	MIXED ORGANIC AREA SUMP PUMP	7.5kW	Stainless Steel 316/316L wetted parts	1	YES	7.5	A
700-SS-001	Duty	3	SOLVENT EXTRACTION (PA & RE) AREA SAFETY SHOWER/EYEWASH		Stainless Steel		NO		A
AREA 830	-		**RARE EARTH RECOVERY**						
830-TK-001	Duty	1	RARE EARTH PRECIPITATION TANK	566m^3	Stainless Steel 316L		NO		A
830-AG-001	Duty	1	RARE EARTH PRECIPITATION TANK AGITATOR	22kW	Stainless Steel 316L	1	YES	22.0	A
830-PU-001	Duty	1	RARE EARTH PRECIPITATION TANK PUMP	245m^3/hr and 90kW	Stainless Steel 316L	1	YES	90.0	A
830-VP-001	Duty	1	RARE EARTH PRECIPITATION SOLID/LIQUID SEPARATION VENDOR PACKAGE	500kW	Stainless Steel 316L	1	YES	500.0	A
700-TK-002	Duty	1	CENTRIFUGE FILTRATE TANK	566m^3	Carbon Steel		NO		A
700-AG-002	Duty	1	CENTRIFUGE FILTRATE TANK AGITATOR	22kW	Carbon Steel	1	YES	22.0	A
700-PU-002	Duty	1	CENTRIFUGE FILTRATE TANK PUMP	245m^3/hr and 90kW	Carbon Steel	1	YES	90.0	A
830-PU-003	Duty	1	RARE EARTH RECOVERY AREA SUMP PUMP	7.5kW	Stainless Steel 316/316L wetted parts	1	YES	7.5	A
830-SS-001	Duty	1	RARE EARTH RECOVERY AREA SAFETY SHOWER/EYEWASH		Stainless Steel 316L		NO		A

MECHANICAL EQUIPMENT LIST

Tag	Duty / Standby	Quantity	Equipment title	Capacity	Materials Of Construction	No of Drives	Variable speed drive	kW Installed	Rev
AREA 860	**-**		**MANGANESE, NICKEL & ZINC RECOVERY**						
860-TK-001	Duty	1	MANGANESE, NICKEL & ZINC CARBONATE PRECIPITATION TANK	58m^3	Carbon Steel		NO		A
860-AG-001	Duty	1	MN, NICKEL & ZINC CARBONATE PRECIPITATION TANK AGITATOR	3.0kW	Carbon Steel	1	YES	3.0	A
860-PU-001	Duty	1	MN, NICKEL & ZINC CARBONATE PRECIPITATION TANK PUMP	24.5m^3/hr and 11kW	Carbon Steel	1	YES	11.0	A
860-VP-001	Duty	1	MN, NICKEL & ZINC CARBONATE PRECIPITATION SOLID/LIQUID SEPARATION VENDOR PACKAGE	500kW		1	YES	500.0	A
860-TK-002	Duty	1	WASTE WATER STORAGE TANK 5	111m^3	Carbon Steel		NO		A
860-AG-002	Duty	1	WASTE WATER STORAGE TANK AGITATOR	1.5kW	Carbon Steel	1	YES	1.5	A
860-PU-002	Duty	1	WASTE WATER STORAGE TANK PUMP	22.3m^3/hr and 11kW	Carbon Steel	1	YES	11.0	A
860-PU-003	Duty	1	MN, NICKEL & ZINC RECOVERY AREA SUMP PUMP	1.1kW	316/316L wetted parts	1	YES	1.1	A
860-SS-001	Duty	1	MN, NICKEL & ZINC RECOVERY AREA SAFETY SHOWER/EYEWASH		Stainless Steel 316L		NO		A
AREA 1100	**-**		**REAGENTS**						
1100-SI-001	Duty	1	AMMONIUM CARBONATE SILO	50m^3	Stainless Steel 316L		NO		A
1100-CV-001	Duty	1	AMMONIUM CARBONATE SCREW CONVEYOR AND WEIGHTOMETER	2kW	Stainless Steel 316L	1	YES	2.0	A
1100-TK-001	Duty	1	AMMONIUM CARBONATE 50% SLURRY MIXING TANK	111m^3	Stainless Steel 316L		NO		A
1100-AG-001	Duty	1	AMMONIUM CARBONATE SLURRY TANK AGITATOR	5.5kW	Stainless Steel 316L	1	YES	5.5	A
1100-PU-001	Duty	2	AMMONIUM CARBONATE SLURRY TANK PUMP	1.7m^3/hr and 1.1kW	Stainless Steel 316L	1	YES	1.1	A
1100-TK-002	Duty	1	AMMONIUM HYDROXIDE (60%) STORAGE TANK	891m^3	Carbon Steel		NO		A
1100-PU-003	Duty	4	AMMONIUM HYDROXIDE (60%) STORAGE TANK PUMP	3.1m^3/hr and 2.2kW	Carbon Steel	4	YES	2.2	A
1100-PU-007	Duty	2	AMMONIUM HYDROXIDE (60%) STORAGE TANK PUMP	2.1m^3/hr and 1.1kW	Carbon Steel	1	YES	1.1	A
1100-TK-003	Duty	1	AMMONIUM HYDROXIDE (4%) STORAGE TANK	137m^3	Carbon Steel		NO		A
1100-PU-009	Duty	4	AMMONIUM HYDROXIDE (4%) STORAGE TANK PUMP	2.3m^3/hr and 2.2kW	Carbon Steel	4	YES	2.2	A
1100-TK-004	Duty	1	DILUENT STORAGE (SHELL SOL) TANK	53m^3	Carbon Steel		NO		A
1100-PU-013	Duty	1	DILUENT STORAGE TANK PUMP	0.3m^3/hr and 1.1kW	Carbon Steel	1	YES	1.1	A
1100-TK-005	Duty	1	FERRIC SULPHATE LIQUOR (60%) SUPPLY TANK	119m^3	Stainless Steel 316L		NO		A
1100-AG-002	Duty	1	FERRIC SULPHATE LIQUOR (60%) SUPPLY TANK AGITATOR	5.5kW	Stainless Steel 316L	1	YES	5.5	A
1100-PU-014	Duty	2	FERRIC SULPHATE LIQUOR (60%) SUPPLY TANK PUMP	2m^3/hr and 1.1kW	Stainless Steel 316L	1	YES	1.1	A
1100-TK-006	Duty	1	HYDRATED LIME (40%) STORAGE TANK	81.7m^3	Carbon Steel, rubber lined		NO		A
1100-PU-016	Duty	2	HYDRATED LIME (40%) STORAGE TANK PUMP	2.8m^3/hr and 1.1kW	Carbon Steel, rubber lined	1	YES	1.1	A
1100-SI-002	Duty	1	HYDRATED LIME (HIGH PURITY) SILO	100m^3	Carbon Steel		NO		A
1100-CV-002	Duty	1	HYDRATED LIME (HIGH PURITY) SCREW CONVEYOR AND WEIGHTOMETER	4kW	Carbon Steel	1	YES	4.0	A
1100-AG-003	Duty	1	HYDRATED LIME (HIGH PURITY) (20%) STORAGE TANK AGITATOR	3kW	Carbon Steel, rubber lined	1	YES	3.0	A
1100-TK-007	Duty	1	HYDRATED LIME (HIGH PURITY) (20%) STORAGE TANK	81.7m^3	Carbon Steel, rubber lined		NO		A
1100-PU-018	Duty	2	HYDRATED LIME (HIGH PURITY) (20%) STORAGE TANK PUMP	2.9m^3/hr and 1.1kW	Carbon Steel, rubber lined	1	YES	1.1	A
1100-TK-010	Duty	1	HYDROGEN PEROXIDE (35%) STORAGE TANK	11m^3	Stainless Steel 316L		NO		A
1100-PU-020	Duty	2	HYDROGEN PEROXIDE (35%) STORAGE TANK PUMP	0.05m^3/hr and 1.1kW	Stainless Steel 316L	1	YES	1.1	A
1100-TK-011	Duty	1	ORGANIC SOLVENT STORAGE (TBP) TANK	234m^3	Stainless Steel 316L		NO		A
1100-PU-022	Duty	1	ORGANIC SOLVENT STORAGE (TBP) TANK PUMP	0.05m^3/hr and 1.1kW	Stainless Steel 316L	1	YES	1.1	A
1100-TK-012	Duty	1	ORGANIC SOLVENT STORAGE (D2EHPA) TANK	234m^3	Stainless Steel 316L		NO		A
1100-PU-023	Duty	1	ORGANIC SOLVENT STORAGE (D2EHPA) TANK PUMP	0.1m^3/hr and 1.1kW	Stainless Steel 316L	1	YES	1.1	A
1100-SI-003	Duty	1	PYROLUSITE SILO	100m^3	Carbon Steel		NO		A
1100-CV-003	Duty	1	PYROLUSITE SCREW CONVEYOR AND WEIGHTOMETER	5kW	Carbon Steel	1	YES	5.0	A
1100-TK-013	Duty	1	PYROLUSITE 50% SLURRY MIXING TANK	75.4m^3	Carbon Steel		NO		A
1100-AG-004	Duty	1	PYROLUSITE SLURRY TANK AGITATOR	4kW	Carbon Steel	1	YES	4.0	A
1100-PU-024	Duty	2	PYROLUSITE SLURRY TANK PUMP	0.3m^3/hr and 1.1kW	Carbon Steel	1	YES	1.1	A
1100-BN-001	Duty	1	PYRITE STORAGE BIN	1000m^3	Carbon Steel		NO		A
1100-VP-001	Duty	1	PYRITE BURNING PLANT - VENDOR PACKAGE	500kW	Stainless Steel 316L	1	YES	500.0	A
1100-BN-001	Duty	1	SULPHUR STORAGE BIN	1000m^3	Carbon Steel		NO		A
1100-VP-002	Duty	1	SULPHURIC ACID PLANT - VENDOR PACKAGE	500kW	Stainless Steel 316L	1	YES	500.0	A
1100-TK-014	Duty	3	SULPHURIC ACID (98%) STORAGE TANK	1218m^3	Carbon Steel		NO		A
1100-PU-026	Duty	2	SULPHURIC ACID (98%) STORAGE TANK PUMP	27m^3/hr and 15kW	Carbon Steel	2	YES	15.0	A
1100-PU-028	Duty	2	SULPHURIC ACID (98%) STORAGE TANK PUMP	10.8m^3/hr and 5.5kW	Carbon Steel	1	YES	5.5	A
1100-TK-017	Duty	6	SULPHURIC ACID (10%) MIXING TANK	891m^3	Stainless Steel 316L		NO		A
1100-AG-005	Duty	6	SULPHURIC ACID (10%) MIXING TANK AGITATOR	7.5kW	Stainless Steel 316L	6	YES	7.5	A
1100-PU-030	Duty	8	SULPHURIC ACID (10%) MIXING TANK PUMP	90.9m^3/hr and 37kW	Stainless Steel 316L	6	YES	37.0	A
1100-BN-002	Duty	1	WASHED SCRAP IRON BIN	200m^3	Carbon Steel		NO		A
1100-HO-001	Duty	2	REAGENT HOIST	1.6 t	Carbon Steel, painted to spec.		NO		A
1100-PU-038	Duty	2	REAGENTS AREA SUMP PUMP	37kW	316/316L wetted parts	2	YES	37.0	A
1100-SS-001	Duty	2	REAGENTS AREA SAFETY SHOWER/EYEWASH		Stainless Steel 316L		NO		A
AREA 1000	**-**		**WASTE WATER TREATMENT**						
AREA 1100	**-**		**UTILITIES & SERVICES**						
AREA 1200	**-**		**BUILDINGS**						

The Uranium Discovery Company

U3O8 CORP. PRELIMINARY ECONOMIC ASSESSMENT ON THE BERLIN DEPOSIT, COLOMBIA

Report Prepared for

U3O8 Corp.

APPENDIX E: OPTION A PROCESS CONCEPTUAL FLOWSHEET

Report Prepared by

Tenova Mining & Minerals (Australia) Pty Ltd

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U3O8 CORPORATION

BERLIN PROJECT - OPTION A

PEA STUDY

PROCESS CONCEPTUAL FLOWSHEET

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U3O8 Corp – Berlin Project Option A Conceptual Flow Sheet

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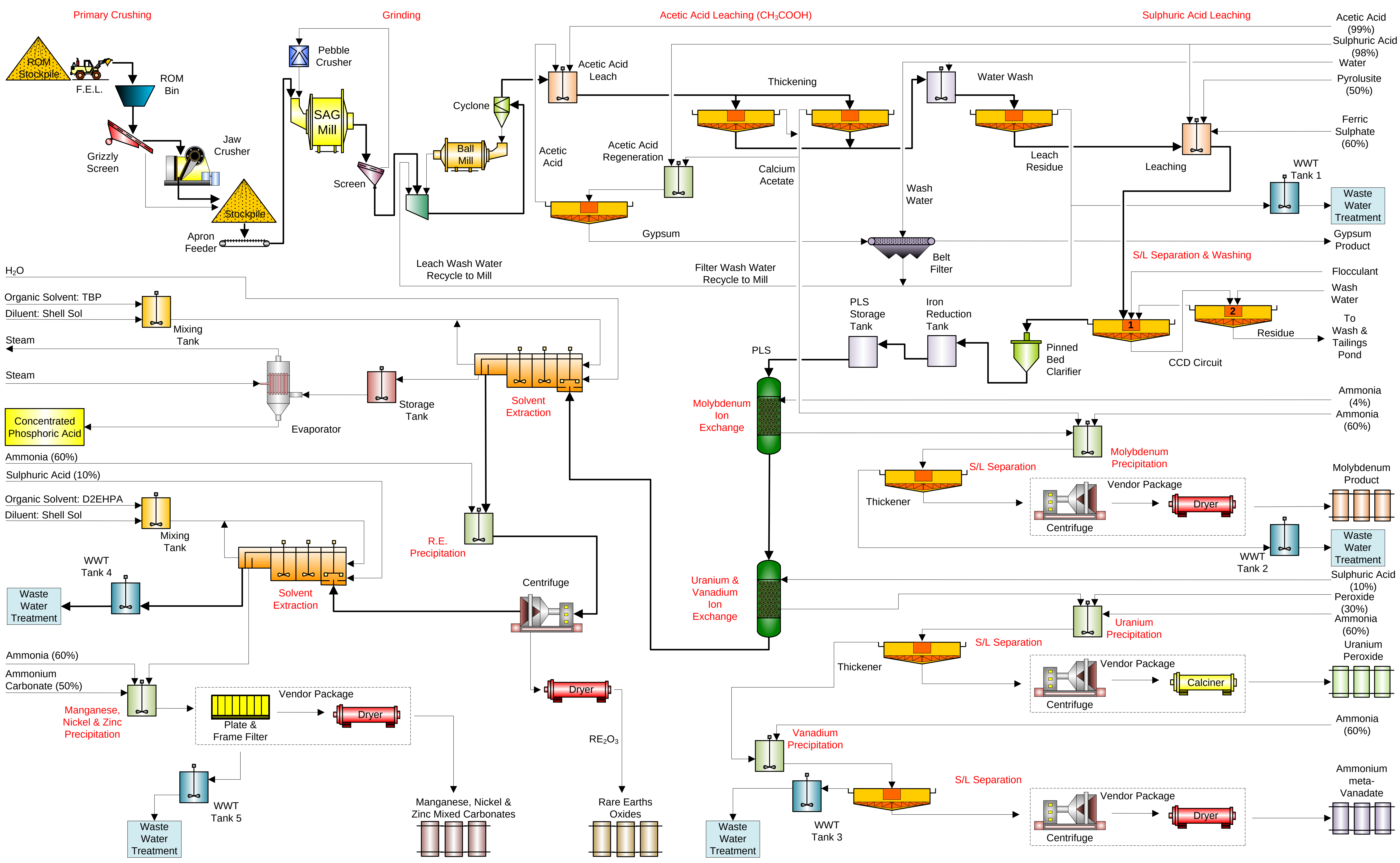

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U3O8 CORP. PRELIMINARY ECONOMIC ASSESSMENT ON THE BERLIN DEPOSIT, COLOMBIA

Report Prepared for

U3O8 Corp.

APPENDIX F: OPTION B PROCESS CONCEPTUAL FLOW SHEET

Report Prepared by

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BERLIN PROJECT - OPTION B

PEA STUDY

PROCESS CONCEPTUAL FLOWSHEET

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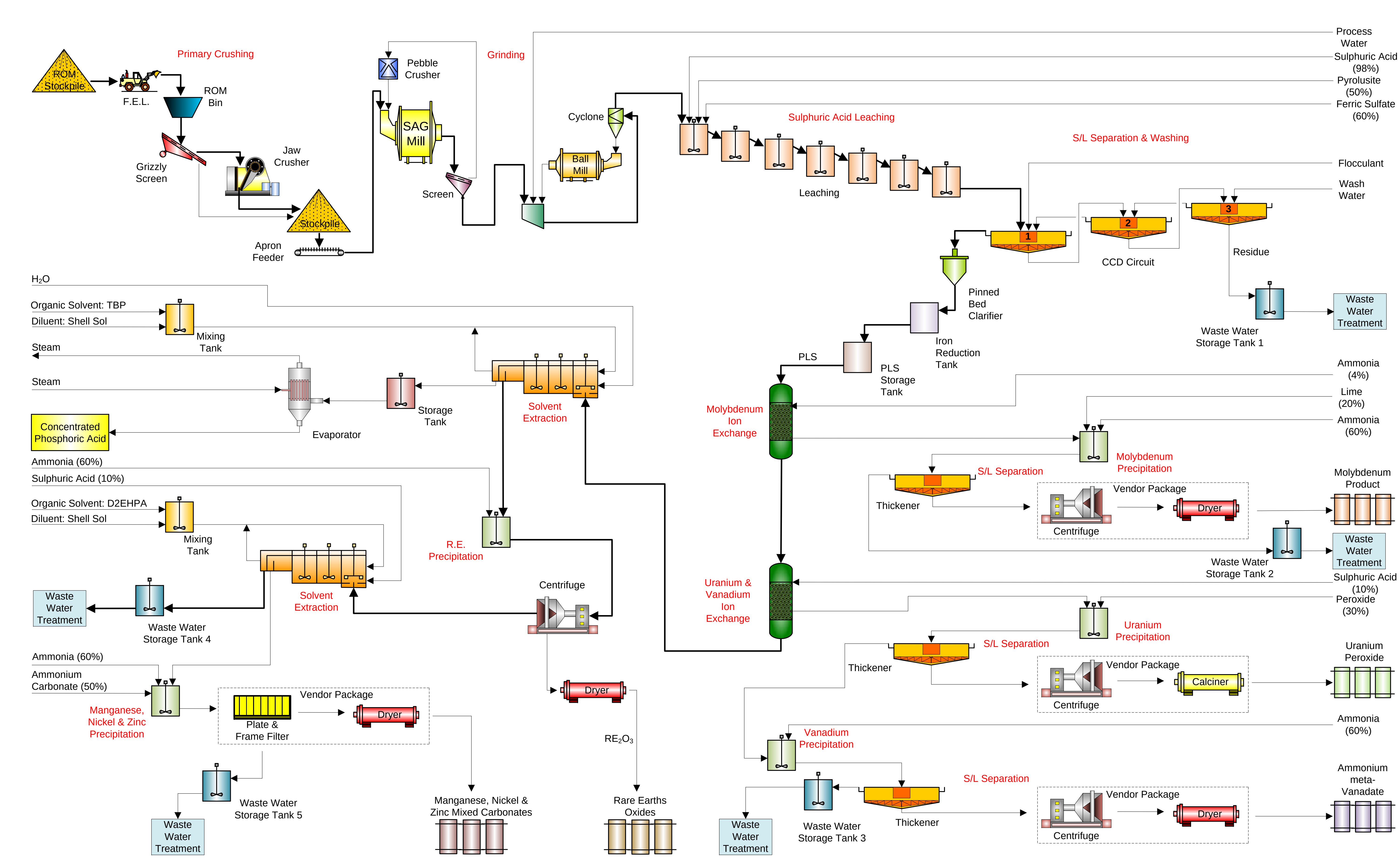

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Report Prepared for

U3O8 Corp.

APPENDIX G: OPTION A PROCESSING PLANT MASS BALANCE

Report Prepared by

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BERLIN PROJECT - OPTION A

PEA STUDY

PROCESSING PLANT MASS BALANCE

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MASS BALANCE OVERVIEW

Stream Description	Stream Units	ROM Material (Grizzly Screen Feed)	Grizzly Screen U/S	Grizzly Screen O/S (Primary Crusher Feed)	Primary Crusher Discharge	SAG Mill Feed (Fresh + Recycle)	SAG Mill Discharge (Screen Feed)	Screen U/S	Screen O/S (SAG Mill Recycle)	TOTAL Ball Mill Sump Feed	Leach Wash Water Recycle to Sump	Ball Mill Sump Slurry Dilution Water	Ball Mill Sump Discharge (Cyclone Feed)	Cyclone U/F (Ball Mill Feed)
Stream Number		**001**	**002**	**003**	**004**	**005**	**006**	**007**	**008**	**009**	**010**	**011**	**012**	**013**
DRY SOLIDS MASS FLOW RATE	t/h	122.0	122.0	122.0	122.0	305.0	305.0	122.0	183.0	366.0	-	-	366.0	244.0
DRY SOLIDS VOL FLOW RATE	m3/h	34.9	34.9	34.9	34.9	87.1	87.1	34.9	52.3	104.6	-	-	104.6	69.7
TOTAL														
TOTAL (PULP) MASS FLOW RATE	t/h	135.6	135.6	135.6	135.6	338.89	338.9	135.6	203.3	697.1	114.4	-	697.1	348.6
TOTAL (PULP) VOL FLOW RATE	m3/h	48.4	48.4	48.4	48.4	121.03	121.0	48.4	72.6	435.7	114.4	-	435.7	174.3
PULP DENSITY (%w/w Solids)	% w/w	90.00%	90.00%	90.00%	90.00%	90.00%	90.00%	90.00%	90.00%	52.50%	0.00%	0.00%	52.50%	70.00%
LIQUIDS														
URANIUM	t/h	-	-	-	-	-	-	-	-	-	-	-	-	-
VANADIUM	t/h	-	-	-	-	-	-	-	-	-	-	-	-	-
MOLYBDENUM	t/h	-	-	-	-	-	-	-	-	-	-	-	-	-
PHOSPHORUS	t/h	-	-	-	-	-	-	-	-	-	-	-	-	-
NICKEL	t/h	-	-	-	-	-	-	-	-	-	-	-	-	-
ZINC	t/h	-	-	-	-	-	-	-	-	-	-	-	-	-
MANGANESE	t/h	-	-	-	-	-	-	-	-	-	-	-	-	-
RE (Y & Nd)	t/h	-	-	-	-	-	-	-	-	-	-	-	-	-
ALL OTHERS	t/h	-	-	-	-	-	-	-	-	214.4	114.4	-	214.4	-
SOLIDS														
URANIUM	t/h	0.122	0.122	0.122	0.122	0.305	0.305	0.122	0.183	0.366	-	-	0.366	0.244
VANADIUM	t/h	0.279	0.279	0.279	0.279	0.697	0.697	0.279	0.418	0.836	-	-	0.836	0.557
MOLYBDENUM	t/h	0.065	0.065	0.065	0.065	0.162	0.162	0.065	0.097	0.194	-	-	0.194	0.130
PHOSPHORUS	t/h	3.945	3.945	3.945	3.945	9.861	9.861	3.945	5.917	11.834	-	-	11.834	7.889
NICKEL	t/h	0.303	0.303	0.303	0.303	0.758	0.758	0.303	0.455	0.910	-	-	0.910	0.606
ZINC	t/h	0.409	0.409	0.409	0.409	1.023	1.023	0.409	0.614	1.228	-	-	1.228	0.819
MANGANESE	t/h	-	-	-	-	-	-	-	-	-	-	-	-	-
RE (Y & Nd)	t/h	0.054	0.054	0.054	0.054	0.135	0.135	0.054	0.081	0.162	-	-	0.162	0.108
ALL OTHERS	t/h	116.8	116.8	116.8	116.8	292.1	292.1	116.8	175.2	350.5	-	-	350.5	233.6
WATER														
WATER MASS FLOW RATE	t/h	13.6	13.6	13.6	13.6	33.89	33.9	13.6	20.3	116.8	-	-	116.8	104.6
WATER VOL FLOW RATE	m3/h	13.6	13.6	13.6	13.6	33.89	33.9	13.6	20.3	116.8	-	-	116.8	104.6
DRY SOLIDS S.G.	t/m3	3.5	3.5	3.5	3.5	3.5	3.5	3.5	3.5	3.5	3.5	3.5	3.5	3.5
WATER S.G.	t/m3	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0
PULP S.G.	t/m3	3.25	3.25	3.25	3.25	3.25	3.25	3.25	3.25	2.31	1.00	-	2.31	2.75

Stream Description	Stream Units	Cyclone O/F (Acetic Acid Leach Feed)	Ball Mill Discharge	Glacial Acetic Acid to AA Leach Tank	Recycled Acetic Acid to AA Leach Tank	AA Leach Tank Discharge Slurry (to 2 x Thickeners)	AA Leach Tank Discharge Residue Thickener O/F (to AA Regeneration Tank)	Residue Thickener O/F Bleed (Ca Acetate to Mo Precipitation Tank)	AA Leach Tank Discharge Residue Thickener U/F (to Water Wash Tank)	Sulphuric Acid (to AA Regeneration Tank)	AA Regeneration Tank Discharge	AA Regeneration Thickener U/F (to Gypsum Belt Filter)	AA Regeneration Thickener O/F (recycled Acetic Acid stream)	Gypsum Belt Filter Cake Product
Stream Number		**014**	**015**	**016**	**017**	**018**	**019**	**020**	**021**	**022**	**023**	**024**	**025**	**026**
DRY SOLIDS MASS FLOW RATE	t/h	122.0	244.0	-	-	61.2	-	-	61.2	-	94.7	94.7	-	94.7
DRY SOLIDS VOL FLOW RATE	m3/h	34.9	69.7	-	-	16.1	-	-	16.1	-	40.8	40.8	-	40.8
TOTAL														
TOTAL (PULP) MASS FLOW RATE	t/h	348.6	348.6	2.0	706.1	989.9	867.6	26.0	122.4	54.0	895.5	189.4	706.1	105.2
TOTAL (PULP) VOL FLOW RATE	m3/h	261.4	174.3	1.9	672.4	876.1	845.9	25.1	75.8	29.3	824.0	133.2	688.9	51.3
PULP DENSITY (%w/w Solids)	% w/w	35.00%	70.00%	0.00%	0.00%	6.18%	0.00%	0.00%	50.00%	0.00%	10.58%	50.00%	0.00%	90.00%
LIQUIDS														
URANIUM	t/h	-	-	-	-	0.01	0.01	-	-	-	-	-	0.01	-
VANADIUM	t/h	-	-	-	-	-	-	-	-	-	-	-	-	-
MOLYBDENUM	t/h	-	-	-	-	-	-	-	-	-	-	-	-	-
PHOSPHORUS	t/h	-	-	-	-	-	-	-	-	-	-	-	-	-
NICKEL	t/h	-	-	-	-	0.03	0.03	-	-	-	-	-	0.03	-
ZINC	t/h	-	-	-	-	-	-	-	-	-	-	-	-	-
MANGANESE	t/h	-	-	-	-	-	-	-	-	-	-	-	-	-
RE (Y & Nd)	t/h	-	-	-	-	-	-	-	-	-	-	-	-	-
ALL OTHERS	t/h	-	-	2.0	231.1	312.4	291.83	8.8	20.6	-	262.1	31.0	231.1	-
SOLIDS														
URANIUM	t/h	0.122	0.244	-	-	0.116	-	-	0.116	-	-	-	-	-
VANADIUM	t/h	0.279	0.557	-	-	0.279	-	-	0.279	-	-	-	-	-
MOLYBDENUM	t/h	0.065	0.130	-	-	0.065	-	-	0.065	-	-	-	-	-
PHOSPHORUS	t/h	3.945	7.889	-	-	3.945	-	-	3.945	-	-	-	-	-
NICKEL	t/h	0.303	0.606	-	-	0.275	-	-	0.275	-	-	-	-	-
ZINC	t/h	0.409	0.819	-	-	0.409	-	-	0.409	-	-	-	-	-
MANGANESE	t/h	-	-	-	-	-	-	-	-	-	-	-	-	-
RE (Y & Nd)	t/h	0.054	0.108	-	-	0.054	-	-	0.054	-	-	-	-	-
ALL OTHERS	t/h	116.82	233.6	-	-	56.0	-	-	56.0	-	94.7	94.7	-	94.7
WATER														
WATER MASS FLOW RATE	t/h	226.6	104.6	-	475.0	616.3	575.7	17.3	40.6	-	538.6	63.7	474.9	10.5
WATER VOL FLOW RATE	m3/h	226.6	104.6	-	475.0	616.3	575.7	17.3	40.6	-	538.6	63.7	474.9	10.5
DRY SOLIDS S.G.	t/m3	3.5	3.5	3.5	3.5	3.8	3.8	3.8	3.8	3.8	2.3	2.3	2.3	2.3
WATER S.G.	t/m3	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0
PULP S.G.	t/m3	1.88	2.75	1.05	1.05	1.13	1.03	1.04	1.62	1.84	1.09	1.42	1.02	2.05

MASS BALANCE OVERVIEW

Stream Description	Gypsum Belt Filter Filtrate	Gypsum Belt Filter Wash Water	Water to Leach Residue Water Wash Tank	Leach Residue Water Wash Tank Discharge Slurry (to Thickener)	Thickener O/F (Recycle water to Ball Mill Sump)	Thickener U/F (to Sulphuric Acid Leach)	Sulphuric Acid to Sulphuric Acid Leach	Pyrolusite to Sulphuric Acid Leach	Ferric Sulphate to Sulphuric Acid Leach	Sulphuric Acid Leach Discharge (CCD Feed)	CCD Wash Water	CCD Overflow (PLS to IX - Molybdenum)	CCD Underflow (Sulphuric Acid Leach Residue to Tails)
Stream Number	027	028	029	030	031	032	033	034	035	036	037	038	039
DRY SOLIDS MASS FLOW RATE	-	-	-	61.2	-	61.2	-	0.8	2.1	68.7	-	-	68.7
DRY SOLIDS VOL FLOW RATE	-	-	-	16.1	-	16.1	-	0.2	0.67	23.1	-	-	18.1
TOTAL													
TOTAL (PULP) MASS FLOW RATE	94.7	100.0	166.8	289.2	114.4	174.8	25.0	1.7	3.4	202.0	113.0	194.4	120.6
TOTAL (PULP) VOL FLOW RATE	94.7	100.0	166.8	223.5	113.6	129.0	13.6	1.0	2.04	164.8	113.0	180.0	66.1
PULP DENSITY (%w/w Solids)	0.00%	0.00%	0.00%	21.16%	0.00%	35.00%	0.00%	50.00%	60.00%	34.00%	0.00%	0.00%	56.96%
LIQUIDS													
URANIUM	-	-	-	-	-	-	-	-	-	0.114	-	0.107	-
VANADIUM	-	-	-	-	-	-	-	-	-	0.203	-	0.250	-
MOLYBDENUM	-	-	-	-	-	-	-	-	-	0.033	-	0.033	-
PHOSPHORUS	-	-	-	-	-	-	-	-	-	3.945	-	3.945	-
NICKEL	-	-	-	-	-	-	-	-	-	0.164	-	0.154	-
ZINC	-	-	-	-	-	-	-	-	-	0.401	-	0.325	-
MANGANESE	-	-	-	-	-	-	-	-	-	0.026	-	0.026	-
RE (Y & Nd)	-	-	-	-	-	-	-	-	-	0.047	-	0.043	-
ALL OTHERS	-	-	-	-	-	10.3	25.0	-	-	125.7	-	125.8	-
SOLIDS													
URANIUM	-	-	-	0.116	-	0.116	-	-	-	0.003	-	-	0.003
VANADIUM	-	-	-	0.279	-	0.279	-	-	-	0.075	-	-	0.075
MOLYBDENUM	-	-	-	0.065	-	0.065	-	-	-	0.032	-	-	0.032
PHOSPHORUS	-	-	-	3.945	-	3.945	-	-	-	-	-	-	-
NICKEL	-	-	-	0.275	-	0.275	-	-	-	0.111	-	-	0.111
ZINC	-	-	-	0.409	-	0.409	-	-	-	0.008	-	-	0.008
MANGANESE	-	-	-	-	-	-	-	0.528	-	-	-	-	-
RE (Y & Nd)	-	-	-	0.054	-	0.054	-	-	-	0.007	-	-	0.007
ALL OTHERS	-	-	-	56.0	-	56.0	-	-	-	68.4	-	-	68.4
WATER													
WATER MASS FLOW RATE	94.7	100.0	166.8	207.4	104.0	103.4	-	0.8	1.4	2.6	113.0	63.7	51.9
WATER VOL FLOW RATE	94.7	100.0	166.8	207.4	104.0	103.4	-	0.8	1.4	2.6	113.0	63.7	51.9
DRY SOLIDS S.G.	2.3	2.3	3.8	3.8	3.8	3.8	3.8	3.8	3.8	3.8	3.8	3.8	3.8
WATER S.G.	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0
PULP S.G.	1.00	1.00	1.00	1.29	1.01	1.36	1.84	1.67	1.68	1.23	1.00	1.08	1.82

Stream Description	IX - Mo Discharge (to IX -U & V)	IX - Mo Water Wash Feed	IX - Mo Water Wash Discharge	IX - Mo 4% NH4OH Elution Feed	IX - Mo 4% NH4OH Elution Discharge	IX - Mo Water Wash 2 Feed	IX - Mo Water Wash 2 Discharge	Mo IX Elution Liquor Storage Tank Feed	Mo IX Elution Liquor Storage Tank Discharge (to Mo Precipitation Tank)	NH4OH to Mo Precipitation Tank	Mo Precipitation - Discharge Slurry (Filter Feed Slurry)	Mo S/L Separation - Solids (Mo Product)	Mo S/L Separation - Liquids (to WWT)	IX -U & V Discharge (to Phosphoric Acid SX)
Stream Number	040	041	042	043	044	045	046	047	048	049	050	051	052	053
DRY SOLIDS MASS FLOW RATE	-	-	-	-	-	-	-	-	-	-	0.069	0.069	-	-
DRY SOLIDS VOL FLOW RATE	-	-	-	-	-	-	-	-	-	-	0.02	0.016	-	-
TOTAL														
TOTAL (PULP) MASS FLOW RATE	194.37	4.5	5.5	2.2	2.4	4.5	4.5	11.3	11.3	-	37.2	0.070	28.5	193.6
TOTAL (PULP) VOL FLOW RATE	180.0	4.5	5.5	2.25	2.25	4.5	4.5	10.4	10.4	-	28.47	0.016	28.45	179.2
PULP DENSITY (%w/w Solids)	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.18%	98.00%	0.00%	0.00%
LIQUIDS														
URANIUM	0.107	-	-	-	-	-	-	-	-	-	-	-	-	-
VANADIUM	0.250	-	-	-	-	-	-	-	-	-	-	-	-	-
MOLYBDENUM	-	-	-	-	0.033	-	-	0.033	0.0	-	-	-	-	-
PHOSPHORUS	3.945	-	-	-	-	-	-	-	-	-	-	-	-	3.945
NICKEL	0.154	-	-	-	-	-	-	-	-	-	-	-	-	0.154
ZINC	0.325	-	-	-	-	-	-	-	-	-	-	-	-	0.325
MANGANESE	0.026	-	-	-	-	-	-	-	-	-	-	-	-	0.026
RE (Y & Nd)	0.043	-	-	-	-	-	-	-	-	-	-	-	-	0.043
ALL OTHERS	125.8	-	-	0.090	0.087	-	-	-	-	-	0.054	-	0.054	125.2
SOLIDS														
URANIUM	-	-	-	-	-	-	-	-	-	-	-	-	-	-
VANADIUM	-	-	-	-	-	-	-	-	-	-	-	-	-	-
MOLYBDENUM	-	-	-	-	-	-	-	-	-	-	0.033	0.033	-	-
PHOSPHORUS	-	-	-	-	-	-	-	-	-	-	-	-	-	-
NICKEL	-	-	-	-	-	-	-	-	-	-	-	-	-	-
ZINC	-	-	-	-	-	-	-	-	-	-	-	-	-	-
MANGANESE	-	-	-	-	-	-	-	-	-	-	-	-	-	-
RE (Y & Nd)	-	-	-	-	-	-	-	-	-	-	-	-	-	-
ALL OTHERS	-	-	-	-	-	-	-	-	-	-	-	-	-	-
WATER														
WATER MASS FLOW RATE	63.7	4.5	5.5	2.01	2.16	4.5	4.5	11.1	11.1	-	28.420	0.001	28.419	63.9
WATER VOL FLOW RATE	63.7	4.5	5.5	2.01	2.16	4.5	4.5	11.1	11.1	-	28.420	0.001	28.419	63.9
DRY SOLIDS S.G.	3.8	3.8	3.8	3.8	3.8	3.8	3.8	3.8	3.8	3.8	4.4	4.4	4.4	4.4
WATER S.G.	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0
PULP S.G.	1.08	1.00	1.00	0.99	0.99	1.00	1.00	1.08	1.08	1.77	1.01	4.35	1.00	1.08

MASS BALANCE OVERVIEW

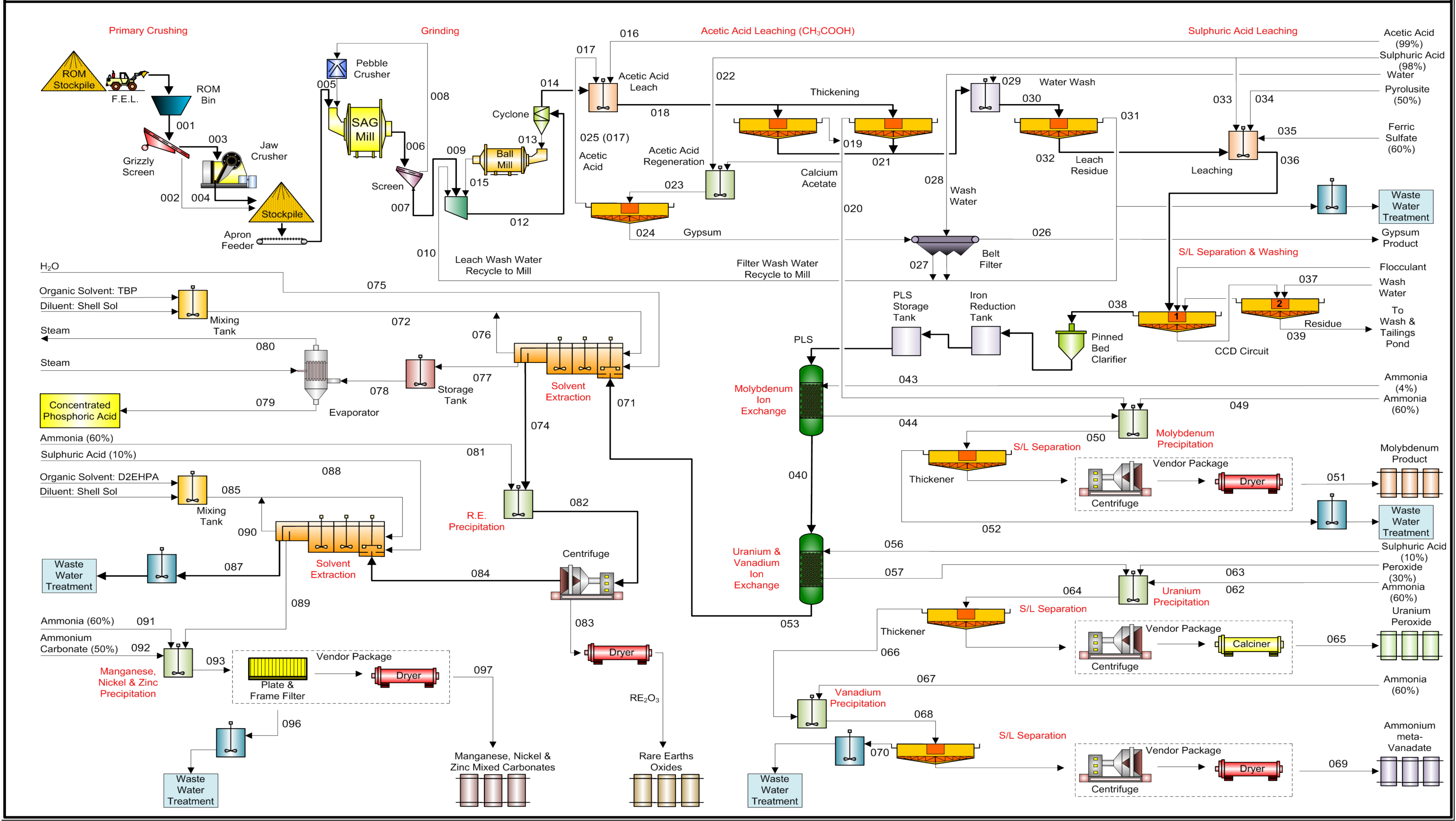
Stream Description	IX -U & V Water Wash Feed	IX -U & V Water Wash Discharge	IX -U & V Sulphuric Acid Elution Feed	IX -U & V Sulphuric Acid Elution Discharge	IX -U & V Water Wash 2 Feed	IX -U & V Water Wash 2 Discharge	U & V Elution Liquor Storage Tank Feed	U & V IX Elution Liquor Storage Tank Discharge (to U Precipitation Tank)	NH4OH to U Precipitation Tank	H2O2 to U Precipitation Tank	U Precipitation Discharge Slurry (Filter Feed)	Uranium S/L Separation - Solids (U Product)	Uranium S/L Separation - Liquids (to V Precipitation)
Stream Number	**054**	**055**	**056**	**057**	**058**	**059**	**060**	**061**	**062**	**063**	**064**	**065**	**066**
DRY SOLIDS MASS FLOW RATE	-	-	-	-	-	-	-	-	-	-	0.152	0.152	-
DRY SOLIDS VOL FLOW RATE	-	-	-	-	-	-	-	-	-	-	0.033	0.046	-
TOTAL													
TOTAL (PULP) MASS FLOW RATE	13.5	13.5	7.1	7.1	13.5	13.5	21.13	21.13	0.84	0.051	22.0	0.169	21.84
TOTAL (PULP) VOL FLOW RATE	13.5	13.5	6.8	6.8	13.5	13.5	20.3	20.3	0.90	0.046	21.2	0.088	21.18
PULP DENSITY (%w/w Solids)	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.69%	90.00%	0.00%
LIQUIDS													
URANIUM	-	-	-	-	-	-	0.107	0.107	-	-	-	-	-
VANADIUM	-	-	-	-	-	-	0.250	0.250	-	-	0.3	-	0.25
MOLYBDENUM	-	-	-	-	-	-	-	-	-	-	-	-	-
PHOSPHORUS	-	-	-	-	-	-	-	-	-	-	-	-	-
NICKEL	-	-	-	-	-	-	-	-	-	-	-	-	-
ZINC	-	-	-	-	-	-	-	-	-	-	-	-	-
MANGANESE	-	-	-	-	-	-	-	-	-	-	-	-	-
RE (Y & Nd)	-	-	-	-	-	-	-	-	-	-	-	-	-
ALL OTHERS	-	-	0.7	0.7	-	-	0.7	0.7	0.5	0.015	1.4	0.046	1.13
SOLIDS													
URANIUM	-	-	-	-	-	-	-	-	-	-	0.107	0.107	-
VANADIUM	-	-	-	-	-	-	-	-	-	-	-	-	-
MOLYBDENUM	-	-	-	-	-	-	-	-	-	-	-	-	-
PHOSPHORUS	-	-	-	-	-	-	-	-	-	-	-	-	-
NICKEL	-	-	-	-	-	-	-	-	-	-	-	-	-
ZINC	-	-	-	-	-	-	-	-	-	-	-	-	-
MANGANESE	-	-	-	-	-	-	-	-	-	-	-	-	-
RE (Y & Nd)	-	-	-	-	-	-	-	-	-	-	-	-	-
ALL OTHERS	-	-	-	-	-	-	-	-	-	-	-	-	-
WATER													
WATER MASS FLOW RATE	13.5	13.5	6.37	6.37	13.5	13.5	19.9	19.9	0.3	0.036	20.3	0.016	20.46
WATER VOL FLOW RATE	13.5	13.5	6.37	6.37	13.5	13.5	19.9	19.9	0.3	0.0	20.3	0.016	20.46
DRY SOLIDS S.G.	4.4	4.4	4.4	4.4	4.4	4.4	4.4	4.4	4.4	4.4	3.3	3.3	3.3
WATER S.G.	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0
PULP S.G.	1.00	1.00	1.05	1.05	1.00	1.00	1.04	1.04	0.92	1.10	1.04	3.05	1.03

Stream Description	NH4OH to V Precipitation Tank	V Precipitation Discharge Slurry (Filter Feed)	Vanadium S/L Separation - Solids (V Product)	Vanadium S/L Separation - Liquids (to WWT)	Phosphoric Acid SX - Aqueous Feed (PLS)	Phosphoric Acid SX - Mixed Organic Solvent	Phosphoric Acid SX - Loaded Organic Discharge	Phosphoric Acid SX - Raffinate Discharge (to RE Precipitation Tank)	Phosphoric Acid SX - Organic Stripping Agent (H2O)	Phosphoric Acid SX - Barren Organic Discharge (recycled for re-use)	Phosphoric Acid SX - Organic Stripping Agent (H2O + H3PO4 to Storage Tank)	H3PO4 Storage Tank Discharge	H3PO4 Evaporator Discharge Product	H3PO4 Evaporator Discharge Steam
Stream Number	**067**	**068**	**069**	**070**	**071**	**072**	**073**	**074**	**075**	**076**	**077**	**078**	**079**	**080**
DRY SOLIDS MASS FLOW RATE	-	0.575	0.575	-	-	-	-	-	-	-	-	-	-	-
DRY SOLIDS VOL FLOW RATE	-	0.247	0.247	-	-	-	-	-	-	-	-	-	-	-
TOTAL														
TOTAL (PULP) MASS FLOW RATE	0.287	22.13	0.64	21.49	193.6	180.0	192.5	193.6	52.0	180.0	52.0	52.0	14.9	37.1
TOTAL (PULP) VOL FLOW RATE	0.310	21.40	0.31	21.09	179.2	180.0	186.6	185.1	52.0	180.0	46.2	46.2	9.1	-
PULP DENSITY (%w/w Solids)	0.00%	2.60%	90.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%
LIQUIDS														
URANIUM	-	-	-	-	-	-	-	-	-	-	-	-	-	-
VANADIUM	-	-	-	-	-	-	-	-	-	-	-	-	-	-
MOLYBDENUM	-	-	-	-	-	-	-	-	-	-	-	-	-	-
PHOSPHORUS	-	-	-	-	3.945	-	-	-	-	-	3.945	3.945	3.945	-
NICKEL	-	-	-	-	0.164	-	-	0.164	-	-	-	-	-	-
ZINC	-	-	-	-	0.401	-	-	0.401	-	-	-	-	-	-
MANGANESE	-	-	-	-	0.028	-	-	0.028	-	-	-	-	-	-
RE (Y & Nd)	-	-	-	-	0.047	-	-	0.047	-	-	-	-	-	-
ALL OTHERS	0.172	0.933	0.003	0.9	115.6	180.0	192.5	116.6	-	180.0	8.537	8.537	8.537	-
SOLIDS														
URANIUM	-	-	-	-	-	-	-	-	-	-	-	-	-	-
VANADIUM	-	0.25	0.250	-	-	-	-	-	-	-	-	-	-	-
MOLYBDENUM	-	-	-	-	-	-	-	-	-	-	-	-	-	-
PHOSPHORUS	-	-	-	-	-	-	-	-	-	-	-	-	-	-
NICKEL	-	-	-	-	-	-	-	-	-	-	-	-	-	-
ZINC	-	-	-	-	-	-	-	-	-	-	-	-	-	-
MANGANESE	-	-	-	-	-	-	-	-	-	-	-	-	-	-
RE (Y & Nd)	-	-	-	-	-	-	-	-	-	-	-	-	-	-
ALL OTHERS	-	-	-	-	-	-	-	-	-	-	-	-	-	-
WATER														
WATER MASS FLOW RATE	0.115	20.6	0.061	20.6	63.9	-	-	76.4	52.0	-	39.5	39.5	2.4	37.1
WATER VOL FLOW RATE	0.115	20.6	0.061	20.6	63.9	-	-	76.4	52.0	-	39.5	39.5	2.4	37.1
DRY SOLIDS S.G.	3.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3
WATER S.G.	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0
PULP S.G.	0.92	1.03	2.06	1.02	1.08	1.00	1.03	1.05	1.00	1.00	1.13	1.13	1.64	1.00

MASS BALANCE OVERVIEW

Stream Description	RE Precipitation Reagent - NH4OH	RE Precipitation Tank Discharge (to S/L Separation)	RE Precipitation - S/L Separation (Solids Discharge to Dryer)	RE Precipitation - S/L Separation (Liquids Discharge) (Mn, Ni & Zn SX Aqueous Feed (PLS))	Mn, Ni & Zn SX - Mixed Organic Solvent	Mn, Ni & Zn SX - Loaded Organic Discharge	Mn, Ni & Zn SX - Raffinate Discharge (to WWT Tank)	Mn, Ni & Zn SX - Organic Stripping Agent (H2SO4)	Mn, Ni & Zn SX - Loaded Organic Stripping Agent (H2SO4) (to Mn, Ni, Zn Precipitation Tank)	Mn, Ni & Zn SX - Barren Organic Discharge (recycled for re-use)	Mn, Ni & Zn Neutralizing Agent -NH4OH (Ammonium Hydroxide)	Mn, Ni & Zn Precipitation Reagent -(NH4)2 CO3 (Ammonium Carbonate)	Mn, Ni & Zn Precipitation Tank - Discharge Slurry	Mn, Ni & Zn Precipitation Tank - Filter Feed Slurry	Mn, Ni & Zn Precipitation - S/L Separation (Solids Discharge) to Dryer	Mn, Ni & Zn Precipitation - S/L Separation (Liquids Discharge) to WWT	Mn, Ni & Zn Precipitation - Dryer Solids Discharge	Mn, Ni & Zn Precipitation - Dryer Steam Discharge	NET PROCESS WATER REQUIREMENT
Stream Number	081	082	083	084	085	086	087	088	089	090	091	092	093	094	095	096	097	098	099
DRY SOLIDS MASS FLOW RATE	-	0.072	0.072	-	-	-	-	-	-	-	-	-	1.16	1.16	1.16	-	1.158	-	-
DRY SOLIDS VOL FLOW RATE	-	0.015	0.015	-	-	-	-	-	-	-	-	-	0.27	0.27	0.27	-	0.267	-	-
TOTAL																			
TOTAL (PULP) MASS FLOW RATE	1.063	194.6	0.080	194.5	180.0	181.5	192.2	21.0	22.0	180.0	2.44	3.87	25.37	25.37	1.29	24.09	1.182	0.11	-
TOTAL (PULP) VOL FLOW RATE	1.150	185.8	0.023	180.1	180.0	180.4	178.0	20.0	20.0	180.0	2.64	3.10	22.74	22.74	0.39	22.35	0.290	0.10	-
PULP DENSITY (%w/w Solids)	0.00%	0.04%	90.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	4.56%	4.56%	90.00%	0.00%	98.00%	0.00%	0.00%
LIQUIDS																			
URANIUM	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
VANADIUM	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
MOLYBDENUM	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
PHOSPHORUS	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
NICKEL	-	-	-	0.164	-	0.164	-	-	0.164	-	-	-	-	-	-	-	-	-	-
ZINC	-	-	-	0.401	-	0.401	-	-	0.401	-	-	-	-	-	-	-	-	-	-
MANGANESE	-	-	-	0.028	-	0.028	-	-	0.028	-	-	-	-	-	-	-	-	-	-
RE (Y & Nd)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
ALL OTHERS	0.64	117.41	0.005	116.8	180.0	181.5	114.8	2.1	2.958	180.0	1.46	2.32	4.01	4.01	0.02	4.0	-	0.02	-
SOLIDS																			
URANIUM	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
VANADIUM	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
MOLYBDENUM	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
PHOSPHORUS	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
NICKEL	-	-	-	-	-	-	-	-	-	-	-	-	0.164	0.164	0.164	-	0.164	-	-
ZINC	-	-	-	-	-	-	-	-	-	-	-	-	0.401	0.401	0.401	-	0.401	-	-
MANGANESE	-	-	-	-	-	-	-	-	-	-	-	-	0.028	0.028	0.028	-	0.028	-	-
RE (Y & Nd)	-	0.047	0.047	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
ALL OTHERS	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
WATER																			
WATER MASS FLOW RATE	0.43	77.11	0.003	77.1	-	-	77.5	18.9	18.5	-	0.98	1.55	20.21	20.21	0.11	20.1	0.024	0.08	-
WATER VOL FLOW RATE	0.43	77.11	0.003	77.1	-	-	77.5	18.9	18.5	-	0.98	1.55	20.21	20.21	0.11	20.1	0.024	0.08	-
DRY SOLIDS S.G.	2.3	4.7	4.7	4.7	4.7	4.7	4.7	4.7	4.7	4.7	4.7	4.7	4.3	4.3	4.3	4.3	4.3	4.3	4.3
WATER S.G.	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0
PULP S.G.	0.92	1.05	3.49	1.08	1.00	1.01	1.08	1.05	1.10	1.00	0.92	1.25	1.12	1.12	3.33	1.08	4.07	1.10	1.00

Berlin Option A – Mass Balance Stream Numbers



U3O8 CORP. PRELIMINARY ECONOMIC ASSESSMENT ON THE BERLIN DEPOSIT, COLOMBIA

Report Prepared for

U3O8 Corp.

APPENDIX H: OPTION A
PROCESSING PLANT MASS BALANCE CALCULATIONS

Report Prepared by

Tenova Mining & Minerals (Australia) Pty Ltd

M6088.A

The Uranium Discovery Company

U3O8 CORPORATION

BERLIN PROJECT - OPTION A

PEA STUDY

PROCESSING PLANT MASS BALANCE CALCULATIONS

M6088.A-P150-002

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0	25/01/2013	ISSUED FOR USE	J SCHLOFFER	P NIEMANN	L DEKLERK	A WALKER
B	6/11/2012	ISSUED FOR ICR	J SCHLOFFER	P NIEMANN	L DEKLERK	
A	1/11/2012	ISSUED FOR IDR	J SCHLOFFER	R RAITER		
REV	**DATE**	**REVISION HISTORY**	**ORIGINATED**	**CHECKED**	**APPROVED**	**TENOVA APPROVAL**

The Uranium Discovery Company

U3O8 CORP

tenova MINING & MINERALS

COMPOUNDS DATA SHEET

FEED COMPOSITION

Ore Mass Flowrate 122.0 t/h

MineralName	Formula	% $^{w}/_{w}$	PPM	Mr	Mass (tph)
Calcite	$CaCO_3$	49.80		100.09	60.756
Fluorapatite	$Ca_5(PO_4)_3F$	17.55		504.31	21.411
Quartz	SiO_2	15.25		60.09	18.600
Muscovite	$KAl_3Si_3O_{11}.H_2O$	8.50		398.33	10.370
Chlorite	$NaClO_2$	3.50		90.44	4.270
Pyrite	FeS_2	2.50		119.99	3.050
Dolomite	$CaMg(CO_3)_2$	1.35		184.41	1.647
Sphalerite	ZnS	0.50		97.45	0.610
Triuranium Octoxide	U_3O_8	0.12	1,179	842.09	0.1438
Vanadium Pentoxide	V_2O_5	0.41	4,077	181.88	0.497
Yttrium Oxide	Y_2O_3	0.05	461	225.82	0.056
Neodymium (III) Oxide	Nd_2O_3	0.01	93	336.48	0.011
Nickel Sulfide	NiS	0.38	3,843	90.77	0.469
Molybdenum Disulfide	MoS_2	0.09	886	160.1	0.108
	TOTAL	100.00			122.00

MineralName	Al	C	Ca	Cl	F	Fe	H	K	Mg	Mn	Mo	N	Na	Nd
Calcite		0.119992007	0.400439604											
Fluorapatite			0.397374631		0.037675239									
Quartz														
Muscovite	0.20319835						0.005071172	0.09815982						
Chlorite				0.391972579									0.254202	
Pyrite						0.465455455								
Dolomite		0.13025324	0.217341793						0.13182582					
Sphalerite														
Triuranium Octoxide														
Vanadium Pentoxide														
Yttrium Oxide														
Neodymium (III) Oxide														0.8573466
Nickel Sulfide														
Molybdenum Disulfide											0.59937539			
	2.11	7.50	33.20	1.67	0.81	1.42	0.05	1.02	0.22	0.00	0.06	0.00	1.09	0.01

MineralName	Ni	O	P	S	Si	U	V	Y	Zn	TOTAL		Mass Flowrate
Calcite		0.479568								1.00		60.76
Fluorapatite		0.380718	0.184232							1.00		21.41
Quartz		0.532535			0.467465					1.00		18.60
Muscovite		0.482012			0.211558					1.00		10.37
Chlorite		0.353826								1.00		4.27
Pyrite				0.534545						1.00		3.05
Dolomite		0.520579								1.00		1.65
Sphalerite				0.329092					0.670908	1.00		0.61
Triuranium Octoxide		0.152003				0.847997				1.00		0.14
Vanadium Pentoxide		0.43985					0.56015			1.00		0.50
Yttrium Oxide		0.212559						0.787441		1.00		0.06
Neodymium (III) Oxide		0.142653								1.00		0.01
Nickel Sulfide	0.646689			0.353311						1.00		0.47
Molybdenum Disulfide				0.400625						1.00		0.11
	0.30	54.81	3.94	2.04	10.89	0.122	0.28	0.04	0.41	SUM	122.00 t/h	122.00

COMPOUNDS DATA

MineralName	Formula	Mr	S.G.		
Pyrite	FeS_2	119.99	5.02		
Oxygen (gas)	O_2	32.00	-		
Magnetite	Fe_3O_4	231.55	5.17		
Sulphur dioxide (gas)	SO_2	64.07	-		
Sulphur trioxide (gas)	SO_3	80.07	-		
Sulphuric Acid	H_2SO_4	98.09	1.84		
Water	H_2O	18.02	1.00		
Pyrolusite	MnO_2	86.94	5.08		
Acetic Acid	CH_3COOH	60.06	1.05		
Ferric Sulphate	$Fe_2(SO_4)_3$	399.91	3.10		
Calcite	$CaCO_3$	100.09	2.71		
Calcium Acetate	$Ca(CH_3COO)_2$	158.18	1.51		
Carbon Dioxide	CO_2	44.01	-		
Gypsum	$CaSO_4.2H_2O$	172.19	2.32		
Nickel Sulphide	NiS	90.77	5.40		
Nickel Acetate	$Ni(CH_3COO)_2$	176.80	1.80		
Hydrogen Sulfide	H_2S	34.09	-		
Triuranium Octoxide	U_3O_8	842.09	8.30		
Uranyl Acetate Dihydride	$UO_2(CH_3COO)_2\ 2H_2O$	424.17	2.89		
Calcium Sulphate	$CaSO_4$	136.15	2.32		
Phosphoric Acid	H_3PO_4	98.00	1.88		
Hydrogen Fluoride	HF	20.01	-		
Magnesium Sulphate	$MgSO_4$	120.38	2.66		
Uranium Dioxide	UO_2	270.03	10.97		
Uranyl Sulphate	$UO_2.SO_4$	366.10	3.28		
Ferrous Sulphate	$FeSO_4$	151.91	2.84		
Manganese Sulphate	$MnSO_4$	151.01	3.25		
Molybdenum Trisulphate	$Mo(SO_4)_3$	384.17	5.00		
Vanadyl Sulphate	$VOSO_4$	163.01	3.00		
Nickel Sulphate	$NiSO_4$	154.77	4.01		
Zinc Sulphate	$ZnSO_4$	161.45	3.54		
Yttrium Sulphate	$Y_2(SO_4)_3$	466.03	2.70		
Neodymium Sulphate	$Nd_2(SO_4)_3$	576.69	2.85		
Ammonium Hydroxide	NH_4OH	35.06	0.88		
Ammonium Sulphate	$(NH_4)_2SO_4$	132.17	1.77		
Ammonium meta-Vanadate	NH_4VO_3	116.99	2.33		
Calcium Hydroxide	$Ca(OH)_2$	74.10	2.21		
Calcium Molybdate	$CaMoO_4$	200.04	4.35		
Calcium Oxide	CaO	56.08	3.30		
Uranium Peroxide Hydrate	$UO_4\ .2H_2O$	338.07	4.67		
Hydrogen Peroxide	H_2O_2	34.02	1.45		
Hydrogen gas	H_2	2.02	-		
Yttrium Hydroxide	$Y(OH)_3$	139.94	4.50		
Neodymium Hydroxide	$Nd(OH)_3$	195.27	7.50		
Ammonium Carbonate	$(NH_4)_2CO_3$	96.11	1.50		
Manganese Carbonate	$MnCO_3$	114.95	3.12		
Nickel Carbonate	$NiCO_3$	118.71	4.39		
Zinc Carbonate	$ZnCO_3$	125.39	4.45		
Hematite	Fe_2O_3	159.70	5.24		
Ammonium Molybdate	$(NH_4)_2MoO_4$	196.06	2.45		
Ammonium Acetate	$NH_4\ CH_3COO$	77.10	1.17		
ShellSol	-	-	0.75		
TBP	-	-	0.98		
D2EHPA	-	-	0.97		

0.1368

ELEMENTAL MOLECULAR WEIGHTS

Element	Mr	Valence Number	Density
Al	26.98		
C	12.01		
Ca	40.08		
Cl	35.45		
F	19.00		
Fe	55.85		
H	1.01		
K	39.10		
Mg	24.31		
Mn	54.94		
Mo	95.96	6	10.28
N	14.01		
Na	22.99		
Nd	144.24		
Ni	58.70		
O	16.00		
P	30.97		
S	32.07		
Si	28.09		
U	238.03	6	19.10
V	50.94	5	6.00
Y	88.91		
Zn	65.38		
V_2O_5	-	2	3.36
H_2SO_4	98.09	2	
NH_4OH	35.06	1	

Compound	Molecular Formula	Mr	Element of Interest	Fraction
Uranyl Acetate Dihydride	$UO_2(CH_3COO)_2\ 2H_2O$	424.17	U	0.561
Nickel Acetate	$Ni(CH_3COO)_2$	176.80	Ni	0.332
Uranyl Sulphate	$UO_2.SO_4$	366.09	U	0.650
Vanadyl Sulphate	$VOSO_4$	163.01	V	0.312
Molybdenum Trisulphate	$Mo(SO_4)_3$	384.17	Mo	0.250
Phosphoric Acid	H_3PO_4	98.00	P	0.316
Nickel Sulphate	$NiSO_4$	154.77	Ni	0.379
Zinc Sulphate	$ZnSO_4$	161.45	Zn	0.405
Manganese Sulphate	$MnSO_4$	151.01	Mn	0.364
Yttrium Sulphate	$Y_2(SO_4)_3$	466.03	Y	0.382
Neodymium Sulphate	$Nd_2(SO_4)_3$	576.69	Nd	0.500
Pyrolusite	MnO_2	86.94	Mn	0.632
Calcium Molybdate	$CaMoO_4$	200.04	Mo	0.480
Ammonium meta-Vanadate	NH_4VO_3	116.99	V	0.435

COMPOUNDS DATA SHEET

SLURRY & REAGENT SG DATA

CALCULATION TABLE

Slurry	Solids Mass (t)	Solids ($\%^w/_w$)	Slurry Mass (t)	Liquids Mass (t)	Solids SG	Liquid SG	Slurry Volume (m^3)	Slurry SG	Compound 1 (solid)	Compound 2 (liquid)	Solids ($\%^v/_v$)
Ferric Sulphate	1.012	60.00%	1.69	0.67	3.1	1.0	1.001	1.685	$Fe_2(SO_4)_3$	H_2O	32.61%
Ferric Sulphate	2.063	60.00%	3.44	1.38	3.1	1.0	2.041	1.685	$Fe_2(SO_4)_4$	H_2O	32.61%
Pyrolusite	0.836	50.00%	1.67	0.836	5.08	1.0	1.001	1.671	MnO_2	H_2O	16.45%
AA Leach Pulp	122.000	12.00%	1016.67	894.667	3.5	1.0	929.524	1.094	Ore	AA and H_2O	3.75%
Sulfuric Acid	0.105	10.00%	1.05	0.943	1.84	1.0	1.000	1.048	H_2SO_4	H_2O	5.69%
Ammonium Hydroxide	0.600	60.00%	1.00	0.400	0.88	1.0	1.082	0.924	NH_4OH	H_2O	63.03%
Ammonium Hydroxide	0.501	60.00%	0.84	0.334	0.88	1.0	0.903	0.924	NH_4OH	H_2O	63.03%
Ammonium Hydroxide	0.716	4.00%	17.90	17.186	0.88	1.0	18.000	0.995	NH_4OH	H_2O	4.52%
Hydrogen Peroxide	0.300	30.00%	1.00	0.700	1.45	1.0	0.907	1.103	H_2O_2	H_2O	22.81%
Ammonium Carbonate	0.600	60.00%	1.00	0.400	1.50	1.0	0.800	1.250	$(NH_4)_2CO_3$	H_2O	50.00%

CALCULATION TABLE

H_3PO_4 SX Organic Basis of calculation: 1.0 m^3

Reagent	($\%^v/_v$)	Volume (m^3)	S.G.	Mass (t)	
ShellSol	25.0%	0.25	0.75	0.188	
TBP	75.0%	0.75	0.98	0.735	
		1.00		0.923	Organic S.G

Mn/Ni/Zn SX Organic Basis of calculation: 1.0 m^3

Reagent	($\%^v/_v$)	Volume (m^3)	S.G.	Mass (t)	
ShellSol	70.0%	0.70	0.75	0.525	
D2EHPA	30.0%	0.30	0.97	0.291	
		1.00		0.816	Organic S.G

INPUTS & CALCULATIONS SHEET

Description	Value	Units	Area	Notes		
OPERATING SCHEDULES						
Days in full year	365	d				
Hours in full day	24	h				
Total hours in a full year	8,760	h				
Anticipated operating hours in a full year	8,200	h				
WATER SPECIFICATIONS						
Water S.G.	1.0					
Water Density	1.0	t/m^3				
ROM FEED						
ROM material flowrate (dry solids)	1,000,400	t/yr				
ROM material flowrate (dry solids)	122.00	t/h				
ROM material moisture content	10.00%	% w/w				
GRIZZLY SCREEN						
Anticipated operating availability	100.00%	%				
Feed Material (dry solids)	122.00	t/h				
PRIMARY CRUSHING						
Anticipated operating availability	85.00%	%				
Jaw Crusher operational feed flowrate (dry solids)	143.53	t/h				
Jaw Crusher average feed flowrate (dry solids)	122.00	t/h				
Jaw Crusher moisture content in feed material	10.00%	% w/w				
STOCKPILE						
Stockpile feed flowrate (dry solids)	122.00	t/h				
Stockpile material moisture content	10.00%	% w/w				
PRIMARY GRINDING						
SAG Mill fresh feed flowrate (dry solids)	122.00	t/h				
SAG Mill feed material moisture content	10.00%	% w/w				
SAG Mill recycle ratio	150.00%	%				
Combined SAG Mill feed flowrate (dry solids) (fresh + recycle)	305.00	t/h				
Combined SAG Mill discharge flowrate (dry solids)	305.00	t/h				
SCREENING						
Screen feed flowrate (dry solids)	305.00	t/h				
Screen feed material moisture content	10.00%	% w/w				
Approximate Mass Split to U/S	0.40					
Screen U/S flowrate (dry solids)	122.00	t/h				
Screen O/S flowrate (dry solids)	183.00	t/h				
Screen U/S flowrate	134.20	t/h		Note: Should be 135.56 t/h.		
HYDROCYCLONES						
Cyclone Feed flowrate (solids)	366.00	t/h		Equivalent to Ball Mill Recycle of 200% + Cyclone O/F		
Cyclone Feed flowrate (Total)	697.14	t/h				
Cyclone Feed solids density	52.50%	% w/w		Calculated		
Cyclone O/F flowrate (solids)	122.00	t/h		From Screen U/S flowrate		
Cyclone O/F flowrate (liquids)	226.57	t/h		Calculated by difference between Total and Solids		
Cyclone O/F flowrate (Total)	348.57	t/h		Calculated		
Cyclone O/F solids density	35.00%	% w/w		Specified		
Cyclone U/F flowrate (solids)	244.00	t/h		Equivalent to Ball Mill Recycle of 200%		
Cyclone U/F flowrate (liquids)	104.57	t/h		Calculated by difference b/t solids and total flowrates		
Cyclone U/F flowrate (Total)	348.57	t/h		Calculated from U/F solids density		
Cyclone U/F solids density	70.00%	% w/w		Specified (ball mill feed density required)		
Fraction of feed material reporting to O/F	33.33%	% w/w		To be back calculated		
SECONDARY GRINDING						
BALL MILL SUMP FEED						
Ball Mill Sump solids density	52.50%	% w/w		Taken from Cyclone Feed solids density		
Solids flowrate	366.00	t/h		Screen U/F + Ball Mill product		
Total Mass flowrate to Ball Mill Sump	697.14	t/h		Calculated from solids flowrate and density		
Total Liquids flowrate	331.14	t/h		Calculated difference b/t total and solids flowrates		
SOLIDS						
Screen U/S	122.00	t/h		Known		
Ball Mill discharge	244.00	t/h		From cyclone U/F		
Total Solids	366.00	t/h				
LIQUIDS						
Screen U/S entrained water	12.20	t/h		Known		
Ball Mill discharge liquid	104.57	t/h		From cyclone U/F		
Fixed Liquids	116.77					
Leach residue wash water	114.37	t/h		Vary to ensure Ball Mill Sump density is correct		
Gypsum belt filter filtrate and wash water	100.00	t/h		Fixed		
Makeup Liquids	214.37	t/h		Total Liquids minus Fixed Liquids		
Total Ball mill sump feed flowrate	697.14	t/h		Sum of solids and liquids	Check Solids %:	52.50%
BALL MILL						
Fraction of solids reporting directly to AA Leach without entering Ball Mill	0.00%	% w/w		Assume zero initially but confirm figure with Bruno		
Ball Mill recycle ratio	200.00%	%		Specified		
Ball Mill solids pulp density	70.00%	% w/w		Specified		
Ball mill sump discharge flowrate	697.14	t/h		From ball mill sump feed		
Ball Mill sump solids pulp density	52.50%	% w/w		Taken from cyclone feed density		
ACETIC ACID LEACH						
Feed						
AA Leach target solids density	12.00%	% w/w				
AA Leach feed solids flowrate	122.00	t/h		Calculated above (Cyclone O/F)		
AA Leach feed liquids flowrate	226.57	t/h		Calculated above (Cyclone O/F)		
AA Leach feed solids density	35.00%	% w/w		Calculated above (Cyclone O/F)		
Glacial acetic acid (98%) flowrate (might need to inc water for mass balance)	1.99	t/h		16kg AA per ton of ore		
Regenerated acetic acid flowrate	72.91	t/h		Stoichiometric requirement to leach calcite (see reactions sheet)		
Total acetic acid flowrate	74.90	t/h				
Solids pulp density (feed + AA acid)	28.81%	% w/w				
Total Liquid required to reach target solids density	894.67	t/h		From slurry calculation table		
Total ADDITIONAL liquid required to reach target solids density	593.19	t/h			666.11	
Confirm AA Leach solids pulp density	12.00%	% w/w		If value = target calculations OK		
Generation and Consumption						
Solids Consumed						
Calcite	60.76	t/h				
Nickel Sulfide	0.047	t/h		From AA Leach reaction calculation	0.47	
Triuranium Octoxide	0.007	t/h		From AA Leach reaction calculation	0.14	

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INPUTS & CALCULATIONS SHEET

Liquids Consumed				
Acetic Acid	72.98	t/h	From AA Leach reaction calculations	
Water	0.0005	t/h	From AA Leach reaction calculation	
Solids Generated				
-	-	t/h	None	
Liquids Generated				
Calcium Acetate	96.02	t/h	From AA Leach reaction calculation	
Water	10.94	t/h	From AA Leach reaction calculation	
Nickel Acetate	0.09	t/h	From AA Leach reaction calculation	
Uranyl Acetate	0.01	t/h	From AA Leach reaction calculation	
Gases Generated				
Carbon Dioxide	26.71	t/h	From AA Leach reaction calculation	
Hydrogen Sulfide	0.02	t/h	From AA Leach reaction calculation	
Discharge				
		t/h		
Solids Discharged				
Leach Residue	61.19	t/h	Calculated (Feed - Consumed)	
Liquids Discharged				
Acetic Acid (unreacted)	1.92	t/h	Calculated (Feed - Consumed)	
Recycled leach residue and wash waters	214.37	t/h	From above, composition TBD so assume inert temporarily	
Calcium Acetate	96.02	t/h	Calculated	
Water	616.33	t/h	Calculated	
Nickel Acetate	0.09	t/h	From AA Leach reaction calculation	
Uranyl Acetate	0.01	t/h	From AA Leach reaction calculation	
Total Liquids Discharged	928.74	t/h		
Gases Discharged				
Carbon Dioxide	26.71	t/h	From Gases Generated	
Hydrogen Sulfide	0.02	t/h	From Gases Generated	
SOLID/LIQUID SEPARATION - THICKENERS				
Feed				
Thickener Feed solids flowrate	61.19	t/h	From leach residue discharge	
Thickener Feed liquid flowrate	928.74	t/h	From leach residue discharge	
Additional Makeup Water to maintain Mass Balance Accuracy (Review this)	0.00	t/h	Preference to equal zero if possible.	
Thickener Feed total flowrate (slurry)	989.93	t/h	Sum	
Thickener Feed slurry pulp density	6.18%	% $^w/_w$	Calculated	
Discharge Streams Calculations				
Thickener U/F solids density TARGET	50.00%	% $^w/_w$	Specified	
Fraction of feed solids reporting to U/F	100.00%	% $^w/_w$		
Thickener U/F solids flowrate	61.19	t/h	All solids from feed report to U/F	
Thickener U/F liquid flowrate	61.19	t/h	Calculated. %$^w/_w$ = (S/(S+L)), rearrange and solve for L.	Check: 50.00%
Thickener O/F liquid flowrate	867.55	t/h	Calculated (O/F - U/F liquids)	
Fraction of feed solids reporting to U/F	100.00	% $^w/_w$	Specified	
Fraction of feed liquids reporting to U/F	6.59%	% $^w/_w$	Calculated	
Discharge Stream - Thickener U/F				
Thickener U/F solids flowrate	61.19	t/h		
Thickener U/F liquid total flowrate	61.19	t/h		867.55
LIQUID COMPOSITION				
Acetic Acid (unreacted)	0.13	t/h		
Recycled leach residue and wash waters	14.12	t/h		
Makeup water*	0.00	t/h		
Calcium Acetate	6.33	t/h		
Water	40.61	t/h		
Nickel Acetate	0.01	t/h		
Uranyl Acetate	0.00	t/h		
Total	61.19		Check corresponds with Thickener U/F liquid flowrate	
Discharge Stream - Thickener O/F				
Thickener O/F flowrate	867.55	t/h		
LIQUID COMPOSITION				
Acetic Acid (unreacted)	1.80	t/h		
Recycled leach residue and wash waters	200.25	t/h		
Makeup water*	0.00	t/h		
Calcium Acetate	89.69	t/h		
Water	575.72	t/h		
Nickel Acetate	0.09	t/h		
Uranyl Acetate	0.01	t/h		
Total	867.55		Check corresponds with Thickener O/F liquid flowrate	
Thickener O/F - Bleed Stream to Molybdenum Precipitation				
Fraction of Thickener O/F Stream reporting to Mo Precipitation	3.00%	% $^w/_w$	Specified by R.Raiter	
Bleed stream flowrate	26.03	t/h		
LIQUID COMPOSITION				
Acetic Acid (unreacted)	0.05	t/h		
Recycled leach residue and wash waters	6.01	t/h		
Makeup water*	0.00	t/h		
Calcium Acetate	2.69	t/h		
Water	17.27	t/h		
Nickel Acetate	0.00	t/h		
Uranyl Acetate	0.00	t/h		
Total	26.03	t/h	Check corresponds with Bleed Stream flowrate	
ACETIC ACID REGENERATION & GYPSUM PRODUCTION				
Feed				
Regeneration feed flowrate	841.53	t/h		
LIQUID COMPOSITION				
Acetic Acid (unreacted)	1.74	t/h		
Recycled leach residue and wash waters	194.24	t/h		
Makeup water*	0.00	t/h		
Calcium Acetate	87.00	t/h		
Water	558.45	t/h		
Nickel Acetate	0.08	t/h		
Uranyl Acetate	0.01	t/h		
Total	841.53	t/h	Check corresponds with Thickener O/F liquid flowrate	
Sulphuric Acid addition	53.95	t/h		
Total Feed	895.48			
Generation & Consumption				
Calcium acetate consumed	87.00	t/h	From AA regen reaction	
Sulphuric acid consumed	53.95	t/h	From AA regen reaction	
Water consumed	19.82	t/h	From AA regen reaction	
Regenerated acetic acid flowrate	66.07	t/h	From AA regen reaction	
Gypsum flowrate	94.71	t/h	From gypsum reaction calculation	
Discharge				
Acetic Acid (unreacted)	1.74	t/h	From above	
Recycled leach residue and wash waters	194.24	t/h	From above	
Makeup water*	0.00	t/h	From above	
Water	538.63	t/h	Calculated (feed - consumed)	
Nickel Acetate	0.08	t/h	From above	
Uranyl Acetate	0.01	t/h	From above	
Acetic Acid (regenerated) flowrate	66.07	t/h	From above	
Gypsum flowrate	94.71	t/h	From above	
Total	895.48	t/h	Check corresponds with feed inputs	
Solid/Liquid Separation - Gypsum/AA Thickener				
Feed				
Thickener Feed total flowrate	895.48	t/h		
Thickener Feed solids pulp density	10.58%	% $^w/_w$		
Thickener Feed solids (gypsum) flowrate	94.71	t/h		
Fraction of solids reporting to U/F	100.00%	%	Specified	
Thickener U/F solids density	50.00%	% $^w/_w$	Specified	

MASS BALANCE CHECK:

	FEED IN		DISCHARGE OUT
Solids	122.0		61.19
Liquids	894.7		928.74
Gas	0.0001		26.73
TOTAL	1016.7	=	1016.7

Difference: 0.000

INPUTS & CALCULATIONS SHEET

Discharge - Thickener U/F					
Thickener U/F solids flowrate	94.71	t/h		Specified	
Thickener U/F liquids flowrate	94.71	t/h		Specified	
Factor for fraction of liquids reporting to U/F	0.12	-		Calculated	
LIQUID COMPOSITION					
Acetic Acid (regenerated + unreacted)	8.02	t/h		Calculated (factor x feed (regen discharge) flowrate)	
Recycled leach residue and wash waters	22.97	t/h		Calculated (factor x feed (regen discharge) flowrate)	
Makeup water*	0.00	t/h		Calculated (factor x feed (regen discharge) flowrate)	
Water	63.70	t/h		Calculated (factor x feed (regen discharge) flowrate)	
Nickel Acetate	0.01	t/h		Calculated (factor x feed (regen discharge) flowrate)	
Uranyl Acetate	0.001	t/h		Calculated (factor x feed (regen discharge) flowrate)	
Liquid Flowrate Total	94.71	t/h		Check corresponds with feed inputs	
Discharge - Thickener O/F					
Thickener O/F liquids flowrate	706.07	t/h		Calculated by difference between feed and U/F	
LIQUID COMPOSITION					
Acetic Acid (regenerated + unreacted)	59.79	t/h		Calculated by difference between feed and U/F	
Recycled leach residue and wash waters	171.27	t/h		Calculated by difference between feed and U/F	
Makeup water*	0.00	t/h		Calculated by difference between feed and U/F	
Water	474.93	t/h		Calculated by difference between feed and U/F	
Nickel Acetate	0.07	t/h		Calculated by difference between feed and U/F	
Uranyl Acetate	0.01	t/h		Calculated by difference between feed and U/F	
Total	706.07			Check corresponds with feed inputs	
Solid/Liquid Separation - Gypsum Belt Filter					
Filter Feed					
Solids flowrate	94.71	t/h			
Liquids flowrate	94.71	t/h			
Total Slurry Flowrate	189.41	t/h			
Filter Cake Product					
Percentage of solids reporting to filter product	100.00%	% $^{w}/_{w}$			
Filter cake moisture content	10.00%	% $^{w}/_{w}$			
Filter cake (gypsum solids only) flowrate	94.71	t/h			
Filter cake total mass flowrate inc 10% moisture	105.23	t/h			
Filter cake liquids only flowrate (assume H_2O only after washing)	10.52	t/h			
Filtrate & Wash Water Calculations					
First Filtrate flowrate	84.18	t/h		Assume filter cake well washed and cake contains only water.	
Second Filtrate flowrate	10.52	t/h			
Wash Water IN	100.00	t/h		Specified in Ball Mill feed slurry dilution calculation above	
Wash Water OUT	89.48	t/h		Calculated	
Wash Water entrained in filter cake	10.52	t/h		Replaces filtrate	
Recycle to Ball Mill Sump					
Total Flowrate	100.00	t/h			
LIQUID COMPOSITION					
First Filtrate	84.18	t/h			
Second Filtrate	10.52	t/h			
Wash Water	5.29	t/h			
Total	100.00	t/h			
SPECIFIC COMPOSITION					
Acetic Acid (regenerated + unreacted)	8.02	t/h			
Recycled leach residue and wash waters	22.97	t/h			
Makeup water*	0.00	t/h			
Water + Wash Water	69.00	t/h			
Nickel Acetate	0.01	t/h			
Uranyl Acetate	0.001	t/h			
Liquid Flowrate Total	100.00	t/h			
Wash water recycle to Ferric Leach Pre-Wash					
Total Flowrate	84.18	t/h		Recycle to minimise water volumes sent to WWT	
SULPHURIC ACID LEACH PRE-WASH					
Mixing Tank Feed					
Acetic Acid Leach Thickener U/F Slurry Flowrate	122.38	t/h			
Slurry solids pulp density	50.00%	% $^{w}/_{w}$			
LIQUID COMPOSITION					
Acetic Acid (unreacted)	0.13	t/h			
Recycled leach residue and wash waters	14.12	t/h			
Makeup water*	0.00	t/h			
Calcium Acetate	6.33	t/h			
Water	40.61	t/h			
Nickel Acetate	0.01	t/h			
Uranyl Acetate	0.001	t/h			
Total	61.19	t/h			
Total Wash Water flowrate, comprising of:	166.82	t/h	52.45	Specified from Ball Mill feed slurry dilution	CHECK and modify if required
Wash water recycled from Gypsum Belt Filter	84.18	t/h		Specified from above	
Fresh wash water flowrate	82.64	t/h		Calculated by difference	
Mixing Tank Discharge (Thickener Feed)					
Solids flowrate	61.19	t/h			
Liquids flowrate	228.01	t/h			
Slurry solids pulp density	21.16%	% $^{w}/_{w}$			
LIQUID COMPOSITION					
Acetic Acid (unreacted)	0.13	t/h			
Recycled leach residue and wash waters	14.12	t/h			
Makeup water*	0.00	t/h			
Calcium Acetate	6.33	t/h		244.76	
Water	207.43	t/h			
Nickel Acetate	0.01	t/h			
Uranyl Acetate	0.001	t/h			
Total	228.01	t/h			
Thickener					
Thickener U/F solids density	35.00%	% $^{w}/_{w}$		Specified (Ferric Leach feed)	
Percentage of solids reporting to U/F	100.00%	% $^{w}/_{w}$		Specified	
Discharge - Thickener U/F					
Thickener U/F slurry flowrate	174.83	t/h		Calculated (solids mass/solids density)	
Solids flowrate	61.19	t/h		All solids report to U/F	
Liquids flowrate	113.64	t/h		Calculated by difference	
Factor for fraction of liquid reporting to U/F	0.50	-			
LIQUID COMPOSITION					
Acetic Acid (unreacted)	0.063	t/h		Calculated	
Recycled leach residue and wash waters	7.039	t/h		Calculated	
Makeup water*	0.000	t/h		Calculated	
Calcium Acetate	3.153	t/h		Calculated	
Water	103.380	t/h		Calculated	
Nickel Acetate	0.003	t/h		Calculated	
Uranyl Acetate	0.000	t/h		Calculated	
Total	113.64	t/h			
Discharge - Thickener O/F					
Thickener O/F flowrate	114.37	t/h		Feed - U/F	
LIQUID COMPOSITION					
Acetic Acid (unreacted)	0.06	t/h		Calculated by balance of Feed - U/F	
Recycled leach residue and wash waters	7.08	t/h		Calculated by balance of Feed - U/F	
Makeup water*	0.00	t/h		Calculated by balance of Feed - U/F	
Calcium Acetate	3.17	t/h		Calculated by balance of Feed - U/F	
Water	104.05	t/h		Calculated by balance of Feed - U/F	
Nickel Acetate	0.003	t/h		Calculated by balance of Feed - U/F	
Uranyl Acetate	0.0004	t/h		Calculated by balance of Feed - U/F	
Total	114.37	t/h			
Thickener O/F Recycle to Ball Mill flowrate - Target	114.37	t/h		*Note: increase wash water flowrate later so Makeup Stream*	
Thickener O/F Recycle to Ball Mill flowrate - Actual	114.37	t/h		*flowrate below equals zero*	
Thickener O/F Recycle to Ball Mill flowrate - Makeup	0.00	t/h			
Thickener O/F Recycle to Ball Mill flowrate (Actual + Makeup)	114.37	t/h			

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INPUTS & CALCULATIONS SHEET

SULPHURIC ACID LEACH				
Feed				
Sulphuric Acid Leach feed slurry flowrate (TOTAL)	174.83	t/h		From Thickener U/F
Sulphuric Acid Leach feed flowrate (liquids)	113.64	t/h		Specified
Sulphuric Acid Leach feed flowrate (solids)	61.19	t/h		Specified
Sulphuric Acid Leach slurry feed solids density	35.00%	% $^{w}/_{w}$		Specified
REAGENTS				
H_2SO_4 flowrate	25.00	t/h		Specified as 25 t/h by Raul Raiter (confirm with Raul)
Pyrolusite - MnO_2 (50%$^{w}/_{w}$ slurry) flowrate	1.67	t/h		Calculated in Compounds Data Sheet (1m^3/hr, R.Raiter)
Ferric Sulphate - $Fe_2(SO_4)_3$ (60%$^{w}/_{w}$ slurry) flowrate	3.44	t/h		Calculated in Reagent Consumption Sheet
Total Solids & Liquids Feed flowrate	204.94	t/h		
Total Gas Feed flowrate	0.00	t/h		Only utilise this field if necessary
FEED SOLIDS (AA LEACH RESIDUE) COMPOSITION				
Fluorapatite - $Ca_5(PO_4)_3F$	21.411			
Quartz - SiO_2	18.600			
Muscovite - $KAl_3Si_3O_{11}.H_2O$	10.370			
Chlorite - $NaClO_2$	4.270			
Pyrite - FeS_2	3.050			
Dolomite - $CaMg(CO_3)_2$	1.647			
Sphalerite - ZnS	0.610			
Triuranium Octoxide - U_3O_8	0.137			
Vanadium Pentoxide - V_2O_5	0.497			
Yttrium Oxide - Y_2O_3	0.056			
Neodymium (III) Oxide - Nd_2O_3	0.011			
Nickel Sulfide - NiS	0.422			
Molybdenum Disulfide - MoS_2	0.108			
Total	61.19			

Generation and Consumption

COMPOUND	INPUT (t/h)	GENERATION (t/h)	CONSUMPTION (t/h)	OUTPUT (t/h) (INPUT + GENERATION - CONSUMPTION)	
SOLIDS					
Fluorapatite - $Ca_5(PO_4)_3F$	21.411	0	21.411	0.000	
Quartz - SiO_2	18.600	0	0	18.600	
Muscovite - $KAl_3Si_3O_{11}.H_2O$	10.370	0	0	10.370	
Chlorite - $NaClO_2$	4.270	0	0	4.270	
Pyrite - FeS_2	3.050	0	0	3.050	
Dolomite - $CaMg(CO_3)_2$	1.647	0	1.647	0.000	
Sphalerite - ZnS	0.610	0	0.598	0.012	
Triuranium Octoxide - U_3O_8	0.137	0	0.134	0.003	
Vanadium Pentoxide - V_2O_5	0.497	0	0.363	0.134	
Yttrium Oxide - Y_2O_3	0.056	0	0.051	0.005	
Neodymium (III) Oxide - Nd_2O_3	0.011	0	0.007	0.004	
Nickel Sulfide - NiS	0.422	0	0.253	0.169	
Molybdenum Disulfide - MoS_2	0.108	0	0.055	0.053	
Subtotal	61.190	0	24.520	36.670	
Ferric Sulphate - $Fe_2(SO_4)_3$	2.063	0.382	2.063	0.382	
Ferrous Sulphate - $FeSO_4$	0	0.856	0.145	0.711	
Pyrolusite - MnO_2	0.836	0	0.041	0.795	
Uranium Dioxide - UO_2	0	0.129	0.129	0.000	
Gypsum - $CaSO_4$	0	30.118	0	30.118	
TOTAL	64.089	31.484	26.898	68.676	
LIQUIDS					
Ferric Leach feed liquids flowrate (H_2O, calcium acetate etc)	113.638	0	0	113.638	
Water within pyrolusite & ferric sulphate reagents	2.211	0	0	2.211	
Sulphuric Acid - H_2SO_4	25.000	0	24.110	0.890	
Water - H_2O	0	0.424	0	0.424	
Phosphoric Acid - H_3PO_4	0	12.482	0	12.482	
Magnesium Sulphate - $MgSO_4$	0	1.075	0	1.075	
Uranyl Sulphate - $UO_2.SO_4$	0	0.175	0	0.175	0.1135239
Manganese Sulphate - $MnSO_4$	0	0.072	0	0.072	
Molybdenum Sulphate - $Mo(SO_4)_2$	0	0.132	0	0.132	
Vanadyl Sulphate - $VOSO_4$	0	0.651	0	0.651	17.14285714
Nickel Sulphate - $NiSO_4$	0	0.432	0	0.432	
Zinc Sulphate - $ZnSO_4$	0	0.990	0	0.990	
Yttrium Sulphate - $Y_2(SO_4)_3$	0	0.106	0	0.106	
Neodymium Sulphate - $Nd_2(SO_4)_3$	0	0.012	0	0.012	
TOTAL	140.850	16.552	24.110	133.292	
GAS					
Hydrogen Fluoride - HF	0	0.850	0	0.850	0.340033113
Carbon Dioxide - CO_2	0	0.786	0	0.786	
Oxygen - O_2	0	0.073	0	0.073	
Hydrogen Sulfide - H_2S	0	0.328	0	0.328	
Sulfur Dioxide - SO_2	0	0	0	0	
Hydrogen - H_2	0	0.001	0	0.001	
TOTAL	0.000	2.036	0.000	2.036	Discrepancy: -0.935 t
TOTAL OF ALL PHASES (SOLID + LIQUID + GAS)	204.939	50.072	51.007	204.003	Discrepancy: -0.46%

Description	Value	Units	Area	Notes
SOLID/LIQUID SEPARATION - CCD CIRCUIT				
CCD Feed				
Solids flowrate to CCD from leach discharge	68.68	t/h		From reactions calculations summary table above
Liquid flowrate to CCD from leach discharge	133.29	t/h		From reactions calculations summary table above
Slurry pulp density to CCD	34.00%	% $^{w}/_{w}$		Calculated
Mo IX Wash Water Recycle (Wash Number 1)	4.50	t/h		Check this - 36m3 per cycle, but per hour rate is less.
U& V IX Wash Water Recycle (Wash Number 1)	13.50	t/h		Check this - 36m3 per cycle, but per hour rate is less.
Subtotal Liquids to CCD	151.29	t/h		Sum
CCD wash water flowrate	95.00	t/h		Specified
Total Liquids to CCD	246.29	t/h		Sum
PLS Volumetric flowrate	180.00	m^3/h		Specified
PLS S.G	1.08			Specified by P.Miller
PLS mass flowrate	194.40	t/h	194.40	Calculated
CCD U/F Solids density	56.96%	% $^{w}/_{w}$		Calculated
Assumed PLS losses to tailings	0.00%	% $^{w}/_{w}$		Specified
Factor to account for PLS losses to U/F (tailings)	1.00	-		
Discharge - CCD O/F				
PLS flowrate	194.40	t/h		Calculated
Fraction of solids to O/F	0.00	% $^{w}/_{w}$		
PLS Composition				
Sulphuric Acid Leach feed liquids flowrate (H_2O, calcium acetate etc)	113.64	t/h		
Water - H_2O	63.74	t/h		Calculated (Balance of PLS - components in PLS)
Sulphuric Acid - H_2SO_4	0.89	t/h		
Phosphoric Acid - H_3PO_4	12.48	t/h		
Magnesium Sulphate - $MgSO_4$	1.08	t/h		
Uranyl Sulphate - $UO_2.SO_4$	0.17	t/h		
Manganese Sulphate - $MnSO_4$	0.07	t/h		
Molybdenum Sulphate - $Mo(SO_4)_2$	0.13	t/h		
Vanadyl Sulphate - $VOSO_4$	0.65	t/h		
Nickel Sulphate - $NiSO_4$	0.43	t/h		
Zinc Sulphate - $ZnSO_4$	0.99	t/h		
Yttrium Sulphate - $Y_2(SO_4)_3$	0.11	t/h		
Neodymium Sulphate - $Nd_2(SO_4)_3$	0.01	t/h		
	194.40	t/h		
Discharge - CCD U/F				
Solids flowrate to U/F	68.68	t/h		100% solids report to U/F
Liquids flowrate to U/F	51.89	t/h		Calculated
Total	120.57	t/h		

INPUTS & CALCULATIONS SHEET

Liquids Composition				
Water - H_2O	51.89	t/h		Calculated by difference b/t liquids to U/F and other cpts
Sulphuric Acid Leach feed liquids flowrate (H_2O, calcium acetate etc)	0.000	t/h		
Sulfuric Acid - H_2SO_4	0.000	t/h		
Phosphoric Acid - H_3PO_4	0.000	t/h		
Magnesium Sulphate - $MgSO_4$	0.000	t/h		
Uranyl Sulphate - $UO_2.SO_4$	0.000	t/h		
Manganese Sulphate - $MnSO_4$	0.000	t/h		
Molybdenum Sulphate - $Mo(SO_4)_2$	0.000	t/h		
Vanadyl Sulphate - $VOSO_4$	0.000	t/h		
Nickel Sulphate - $NiSO_4$	0.000	t/h		
Zinc Sulphate - $ZnSO_4$	0.000	t/h		
Yttrium Sulphate - $Y_2(SO_4)_3$	0.000	t/h		
Neodymium Sulphate - $Nd_2(SO_4)_3$	0.000	t/h		
	51.89	t/h		
ION EXCHANGE				
PLS mass flowrate	194.40	t/h		From above
PLS S.G	1.08			Specified by P.Miller
PLS Volumetric flowrate	180.00	m^3/h		

PLS ASSAY CALCULATED RESULTS:

Element	Mr	Compound	Mr	Element Mass %	Calculated PLS kg/h
Mo	95.96	$Mo(SO_4)_2$	384.17	24.98%	33.04
U	238.03	$UO_2.SO_4$	366.10	65.02%	113.55
V	50.94	$VOSO_4$	163.01	31.25%	203.39
P	30.97	H_3PO_4	98.00	31.60%	3,944.59
Nd	144.24	$Nd_2(SO_4)_3$	576.69	50.02%	6.23
Y	88.91	$Y_2(SO_4)_3$	466.03	38.16%	40.30
Mn	54.94	$MnSO_4$	151.01	36.38%	26.21
Ni	58.70	$NiSO_4$	154.77	37.93%	163.71
Zn	65.38	$ZnSO_4$	161.45	40.50%	401.07

PLS ASSAY LABORATORY RESULTS: Note: proceed with lab PLS assay results from here onwards.

Element	PPM	g/L	kg/h		t/h
Mo	183.0	0.183	32.94		0.033
U	595.0	0.595	107.10		0.107
V	1,391.0	1.391	250.38		0.250
P	27,061.0	27.061	4,870.98		4.871
Nd	44.0	0.044	7.92		0.008
Y	195.0	0.195	35.10		0.035
Mn	145.6	0.146	26.21	Mn value calculated from table above, not defined by P.Miller.	0.026
Ni	857.0	0.857	154.26		0.154
Zn	1,807.0	1.807	325.26		0.325

Description	Value	Units	Area	Notes
ION EXCHANGE - MOLYBDENUM				
Loading				
Resin capacity (C_r)	1.00	eq.Me/L		
Column volume	18.00	m^3		Calculated by solving for bed volume with known flowrate
				Calculation of column size: Flowrate = 10 x Bed Volumes / hour
Mo flowrate contained in PLS	0.183	g/L		Specified
Mo flowrate contained in PLS	32,940.00	g/h		Calculated
1 x Equivalent of Mo Metal	15.99	eq.g		Calculation: 1 x Equivalent gram = (Mr / Valence #)
Equivalent grams of Mo Metal flowrate	2,059.61	eq.g/h		Calculated
Duration required to fully load $18m^3$ column	8.0	h		
Number of IX columns required	3.0	-		
Total Mo loaded to IX columns per 8.0 hours	263,520.0	g		
Elution				
Volumetric Elution Flowrates				
Step 1. Displacement water (Wash Number 1) mass flowrate	36.00	m^3/cycle		2 x Bed Volumes (3 x cycles per day)
Step 2. Alkaline elution (4.0%$^w/_w$ NH_4OH) mass flowrate	18.00	m^3/cycle		1 x Bed Volume (3 x cycles per day)
Step 3. Displacement water (Wash Number 2) mass flowrate	36.00	m^3/cycle		2 x Bed Volumes (3 x cycles per day)
Mass Flowrates				
Step 1. Displacement water (Wash Number 1) mass flowrate	36.00	t/cycle		2 x Bed Volumes - (recycle to CCD Circuit)
Step 2. Alkaline elution (4.0%$^w/_w$ NH_4OH) mass flowrate	17.90	t/cycle		1 x Bed Volume
Step 3. Displacement water (Wash Number 2) mass flowrate	36.00	t/cycle		2 x Bed Volumes
Discharge to Mo Precipitation Tank				
Alkaline elution (4.0%$^w/_w$ NH_4OH) mass flowrate	17.90	t/cycle		Calculated
Displacement water (Wash Number 2) mass flowrate	36.00	t/cycle		
Mo metal	0.26	t/cycle		Hours per cycle x Hourly flowrate of metal
Total	54.17	t/cycle		
Average Hourly Mass Flowrates				
Alkaline elution (4.0%$^w/_w$ NH_4OH) mass flowrate	2.24	t/h		96% water, 4% acid
Displacement water (Wash Number 1) mass flowrate	4.50	t/h		
Displacement water (Wash Number 2) mass flowrate	4.50	t/h		
Mo metal	0.0329	t/h		100% of Mo assumed recovery
Total	11.27	t/h		
MOLYBDENUM PRECIPITATION				
Mo Elution Liquor				
NH_4OH (4.0%$^w/_w$) mass flowrate	0.0895	t/h		
Water mass flowrate	11.15	t/h		
n(NH_4OH) =	2.5531	kg-moles		
Concentration (Molar)	0.23	M		
Number of moles of Molybdenum n(Mo) =	0.3433	kg-moles		Mo is valence 6^+.
Resin capacity (C_r)	1.00	eq.Me/L		
Equivalent grams of NH_4OH/L	35.06	eq.g		Calculation: 1 x Equivalent gram = (Mr / Valence #)
Moles NH_4OH consumed in eluting Mo^{6+}	0.07	kg-moles		Calculated
Remaining NH_4OH (not consumed by elution) to be neutralised	2.48	kg-moles		Input - consumption
Precipitation				
NH_4OH mass flowrate required to bring solution to pH = 7	0.000	t/h		Calculated
NH_4OH (4.0%$^w/_w$ solution) mass flowrate	0.000	t/h		Calculated
** pH is alkaline already, no requirement for raising pH.*				
Composition of Elution Liquor flowing to Precipitation Tank				
Water mass flowrate	11.15	t/h		
NH_4OH mass flowrate	0.0167	t/h		
$(NH_4)_2MoO_4$	0.067	t/h		
Total	11.232			
Composition of Calcium Acetate Bleed Stream for Mo Precipitation				
Bleed stream flowrate (3% of Thickener O/F Stream)	26.03	t/h		Note: Potential to reduce this stream to 1.0 t/h
BLEED STREAM LIQUID COMPOSITION			%$^w/_w$	
Acetic Acid (unreacted)	0.054	t/h	0.21%	
Recycled leach residue and wash waters	6.007	t/h	23.08%	
Makeup water*	0.000	t/h	0.00%	
Calcium Acetate	2.691	t/h	10.34%	
Water	17.272	t/h	66.36%	
Nickel Acetate	0.003	t/h	0.01%	
Uranyl Acetate	0.000	t/h	0.00%	
Total	26.027	t/h	100.00%	
Consumption				
$(NH_4)_2MoO_4$	0.067	t/h		From Reactions & Conversions sheet
$Ca(CH_3COO)_2$	0.054	t/h		From Reactions & Conversions sheet
Generation				
Product: $CaMoO_4$ mass flowrate	0.069	t/h		From Reactions & Conversions sheet
$NH_4.CH_3COO$	0.053	t/h		From Reactions & Conversions sheet

INPUTS & CALCULATIONS SHEET

Discharge to S/L Separation				
Solids				
$CaMoO_4$ mass flowrate	0.069	t/h		From above
Liquids				
Ammonium acetate	0.053	t/h		
Water	28.420	t/h		Sum of water contained in reagents and produced.
Acetic Acid (unreacted)	0.054	t/h		
Recycled leach residue and wash waters	6.007	t/h		
Calcium Acetate	2.636	t/h		
Nickel Acetate	0.003	t/h		
Uranyl Acetate	0.000	t/h		
Total (Solids & Liquids)	37.242	t/h		
Solids density to S/L Separation	0.18%	% $^w/_w$		Calculated
S/L Separation				
Moisture content of $CaMoO_4$ product	2.00%	% $^w/_w$		Specified
$CaMoO_4$ total mass flowrate (inc moisture)	0.070	t/h		Calculated
Moisture flowrate in $CaMoO_4$ product	0.001	t/h		Calculated
Solids content of material reporting to WWT	0.00%	% $^w/_w$		Specified
Mass flowrate of material reporting to WWT	37.172	t/h		Balance by calculation
ION EXCHANGE - URANIUM & VANADIUM				
V Loading				
PLS mass flowrate	194.37	t/h		Mo removed
PLS Volumetric flowrate	180.00	m^3/h		Mo removed
Column volume	18.00	m^3		Calculated by solving for bed volume with known flowrate
				Calculation of column size: Flowrate = 10 x Bed Volumes / hour
V flowrate contained in PLS	1.391	g/L		Specified
V flowrate contained in PLS	250,380	g/h		Calculated
V_2O_5 flowrate contained in PLS	2.483	g/L		Specified
V_2O_5 flowrate contained in PLS	446,988	g/h		Calculated
1 x Equivalent of V Metal (V_2O_5)	90.94	eq.g		Calculation: 1 x Equivalent gram = (Mr / Valence #)
Equivalent grams of V Metal flowrate	4,915.19	eq.g/h		Calculated
Duration required to fully load $18m^3$ column	4.0	h		Specified
Number of IX columns required	6	-		Specified (2 columns per 8 hour duration)
Total V_2O_5 loaded to IX columns per 4.0 hours	1,787,951.1	g		Calculated
U Loading				
PLS mass flowrate	194.12	t/h		Mo & V removed
PLS Volumetric flowrate	179.96	m^3/h		Mo & V removed
Column volume	18.00	m^3		Calculated by solving for bed volume with known flowrate
				Calculation of column size: Flowrate = 10 x Bed Volumes / hour
U flowrate contained in PLS	0.595	g/L		Specified
U flowrate contained in PLS	107,100	g/h		Calculated
1 x Equivalent of U Metal	39.67	eq.g		Calculation: 1 x Equivalent gram = (Mr / Valence #)
Equivalent grams of U Metal flowrate	2,699.66	eq.g/h		Calculated
Duration required to fully load $18m^3$ column	8.0	h		Specified
Number of IX columns required	3	-		Specified
Total U loaded to IX columns per 8.0 hours	856,800.0	g		Calculated
Discharge Liquid to SX				
PLS mass flowrate	193.56	t/h		Mo, U & V removed
PLS Volumetric flowrate	179.78	m^3/h		Mo, U & V removed
V Elution				
Volumetric Elution Flowrates				
Step 1. Displacement water (Wash Number 1) mass flowrate	36.00	m^3/cycle		2 x Bed Volumes (3 x cycles per day with 2 columns)
Step 2. Acid elution (10%$^w/_w$ H_2SO_4) mass flowrate	18.00	m^3/cycle		1 x Bed Volume (3 x cycles per day with 2 columns)
Step 3. Displacement water (Wash Number 2) mass flowrate	36.00	m^3/cycle		2 x Bed Volumes (3 x cycles per day with 2 columns)
Mass Flowrates				
Step 1. Displacement water (Wash Number 1) mass flowrate	36.00	t/cycle		2 x Bed Volumes - (recycle to CCD Circuit)
Step 2. Acid elution (10%$^w/_w$ H_2SO_4) mass flowrate	18.86	t/cycle		1 x Bed Volume
Step 3. Displacement water (Wash Number 2) mass flowrate	36.00	t/cycle		2 x Bed Volumes
Discharge to V Precipitation Tank				
Acid elution (10%$^w/_w$ H_2SO_4) mass flowrate	18.86	t/cycle		Calculated
Displacement water (Wash Number 2) mass flowrate	36.00	t/cycle		
V_2O_5 mass flowrate	1.79	t/cycle		Hours per cycle x Hourly flowrate of metal
Total	56.65	t/cycle		
Average Hourly Mass Flowrates				
Acid elution (10%$^w/_w$ H_2SO_4) mass flowrate	4.72	t/h		Acid flowrate / Cycle duration
Displacement water (Wash Number 1) mass flowrate	9.00	t/h		Water flowrate / Cycle duration
Displacement water (Wash Number 2) mass flowrate	9.00	t/h		Water flowrate / Cycle duration
V_2O_5 mass flowrate	0.45	t/h		100% of V assumed recovery
Total	23.16	t/h		
Discharge to V Precipitation Tank - Hourly				
H_2SO_4 mass flowrate (not including elution consumption)	0.47	t/h		
Displacement water (Wash Number 2) mass flowrate	13.24	t/h		
V_2O_5 mass flowrate	0.45	t/h		
Total	14.16	t/h		
H_2SO_4 molar flowrate	4.81	kg-moles/hr		
Acid consumption during Vanadium elution process				
Resin capacity (C_r)	1.00	eq.Me/L		
Equivalent grams of H_2SO_4/L	49.05	eq.g		Calculation: 1 x Equivalent gram = (Mr / Valence #)
FACTOR FOR ACID CONSUMPTION	0.02	UNITS		
Moles H_2SO_4 consumed in eluting V_2O_5		kg-moles		Calculated
Mass of H_2SO_4 consumed in eluting V_2O_5	0.0096	t/h		Calculated
U Elution				
Volumetric Elution Flowrates				
Step 1. Displacement water (Wash Number 1) mass flowrate	36.00	m^3/cycle		2 x Bed Volumes (3 x cycles per day)
Step 2. Acid elution (10%$^w/_w$ H_2SO_4) mass flowrate	18.00	m^3/cycle		1 x Bed Volume (3 x cycles per day)
Step 3. Displacement water (Wash Number 2) mass flowrate	36.00	m^3/cycle		2 x Bed Volumes (3 x cycles per day)
Mass Flowrates				
Step 1. Displacement water (Wash Number 1) mass flowrate	36.00	t/cycle		2 x Bed Volumes - (recycle to CCD Circuit)
Step 2. Acid elution (10%$^w/_w$ H_2SO_4) mass flowrate	18.86	t/cycle		1 x Bed Volume
Step 3. Displacement water (Wash Number 2) mass flowrate	36.00	t/cycle		2 x Bed Volumes
Discharge to U Precipitation Tank				
Acid elution (10%$^w/_w$ H_2SO_4) mass flowrate	18.86	t/cycle		Calculated
Displacement water (Wash Number 2) mass flowrate	36.00	t/cycle		
U mass flowrate	0.86	t/cycle		Hours per cycle x Hourly flowrate of metal
Total	55.72	t/cycle		
Average Hourly Mass Flowrates				
Acid elution (10%$^w/_w$ H_2SO_4) mass flowrate	2.36	t/h		Acid flowrate / Cycle duration
Displacement water (Wash Number 1) mass flowrate	4.50	t/h		Water flowrate / Cycle duration
Displacement water (Wash Number 2) mass flowrate	4.50	t/h		Water flowrate / Cycle duration
U mass flowrate	0.11	t/h		100% of U assumed recovery
Total	11.46	t/h		
Discharge to U Precipitation Tank - Hourly				
H_2SO_4 mass flowrate (not including elution consumption)	0.24	t/h		
Water mass flowrate	6.62	t/h		
U mass flowrate	0.11	t/h		
Total	6.96	t/h		
H_2SO_4 molar flowrate	2.40	kg-moles/hr		

INPUTS & CALCULATIONS SHEET

Acid consumption during Uranium elution process				
Resin capacity (C_r)	1.00	eq.Me/L		
Equivalent grams of H_2SO_4/L	49.05	eq.g		Calculation: 1 x Equivalent gram = (Mr / Valence #)
FACTOR FOR ACID CONSUMPTION	0.02	UNITS		
Moles H_2SO_4 consumed in eluting U		kg-moles		Calculated
Mass of H_2SO_4 consumed in eluting U	0.0048	t/h		Calculated
URANIUM PRECIPITATION				
Total Elution Liquor (U + V Elution)				
Streams entering U Precipitation Tank				
10%$^w/_w$ H_2SO_4 mass flowrate	7.073	t/h		Sum of elution liquors from U and V IX Columns
Displacement water (Wash Number 1) mass flowrate	13.500	t/h		Sum of elution liquors from U and V IX Columns
U mass flowrate	0.107	t/h		
V_2O_5 mass flowrate	0.447	t/h		
	20.680	t/h		
Actual Composition (taking into account elution acid consumption)				
100%$^w/_w$ H_2SO_4 mass flowrate	0.693	t/h		Calculated
Displacement water (Wash Number 1) mass flowrate	19.866	t/h		Calculated
U mass flowrate	0.107	t/h		
V_2O_5 mass flowrate	0.447	t/h		
	20.666	t/h		Sum
Peroxide (30%$^w/_w$ solution) mass flowrate	0.051	t/h		
Peroxide (100%$^w/_w$) mass flowrate	0.015	t/h		
Water contained in peroxide solution	0.036	t/h		
UO_2 mass flowrate	0.121	t/h		Calculated
Product: $UO_4 . 2H_2O$ mass flowrate	0.152	t/h		From reactions sheet
Water consumption	0.016	t/h		From reactions sheet
Peroxide consumption	0.015	t/h		From reactions sheet
Oxygen generation	0.014	t/h		From reactions sheet
Neutralization Calculation				
$n(H_2SO_4)$ =	7.064	kg-moles		Calculated
Acid concentration (Molar)	0.36	M		Calculated
Therefore, concentration of H^+ = 0.36M = 0.36 mol/L = 0.36 g/L, (as Mr(H) = 1.0)				
$pH = -log_{10}[H^+] = -log_{10}(0.36) = 0.44$	0.44	pH		pH of elution liquor.
pH (elution liquor) = 0.44				
Now calculate $[H^+]$ at pH = 4 where precipitation occurs.				
$pH = -log_{10}[H^+]$				Log Rule: $log_B(B^x) = x.log_B(B) = x$
$pH = 4 = -log_{10}[H^+]$, solving for $[H^+]$: $[H^+] = 10^{-4}$ M				
$[H^+] = 10^{-4}$ M = 10^{-4} mol/L = 0.0001 g/L (as Mr(H) = 1.0)				
100%$^w/_w$ H_2SO_4 concentration at pH = 4	0.0001	g/L		
To raise pH from pH = 0.44 to pH = 4.0, need to neutralize H^+:				
0.36g/L - 0.0001g/L = 0.3599g/L = 0.3599 mol/L	0.3599	mol/L		Calculated
NH_4OH mass flowrate to achieve pH rise	0.495	t/h		From reactions sheet
NH_4OH (60%$^w/_w$) mass flowrate to achieve pH rise	0.835	t/h		Calculated
100%$^w/_w$ H_2SO_4 mass flowrate (0.0001g/L)	0.000002	t/h		(2 grams per hour acid)
Discharge to S/L Separation (prior to V Precipitation Tank)				
100%$^w/_w$ H_2SO_4 mass flowrate	0.000002	t/h		From above
H_2O mass flowrate	20.479	t/h		Calculated (input + generation - consumption = output)
$UO_4 . 2H_2O$ mass flowrate	0.152	t/h		From reactions sheet
V_2O_5 mass flowrate	0.4470	t/h		Unchanged from input
$(NH_4)_2SO_4$ mass flowrate	0.934	t/h		From reactions sheet
Total	22.01	t/h		
Total liquid mass flowrate reporting to S/L Separation	21.86	t/h		
S/L Separation (Thickener & Centrifuge)				
Assume S/L Separation product contains 10%$^w/_w$ liquid	10.00%			Specified
Solids Discharge (Product)				
$UO_4 . 2H_2O$ solids mass flowrate	0.152	t/h		
Total UO_4 product mass flowrate including moisture	0.169	t/h		Assume 10% moisture content
Liquids content contained in solids product	0.0169	t/h		
Fraction of liquid reporting to solids	0.0008	-		
Composition of Liquid contained in solids product				
H_2SO_4 (100%$^w/_w$) mass flowrate	0.000000002	t/h		
H_2O mass flowrate	0.016	t/h		
V_2O_5 mass flowrate	0.00035	t/h		
$(NH_4)_2SO_4$ mass flowrate	0.00072	t/h		
Total	0.0169			
Liquids Discharge (to Vanadium precipitation)				
H_2O mass flowrate	20.464	t/h		
100%$^w/_w$ H_2SO_4 mass flowrate	0.000002	t/h		
V_2O_5 mass flowrate	0.4466	t/h		
$(NH_4)_2SO_4$ mass flowrate	0.933	t/h		
Total	21.84	t/h		
VANADIUM PRECIPITATION				
NH_4OH (pure) mass flowrate to achieve pH rise to pH 7	0.0000014	t/h		From reactions sheet
NH_4OH (pure) mass flowrate to achieve ammonium vanadate precipitation	0.172	t/h		
NH_4OH (60%$^w/_w$) mass flowrate to achieve pH rise	0.287	t/h		Calculated
Product: NH_4VO_3 solids mass flowrate	0.5746	t/h		From reactions sheet
H_2O generated in neutralization and precipitation reactions mass flowrate	0.044	t/h		Calculated
Discharge to S/L Separation				
H_2O mass flowrate	20.623	t/h		
$(NH_4)_2SO_4$ mass flowrate	0.932876	t/h		
NH_4VO_3 mass flowrate	0.575	t/h		
Total	22.13	t/h		Calculated
S/L Separation (Thickener & Centrifuge)				
Assume S/L Separation product contains 10%$^w/_w$ liquid	10.00%			Specified
Solids Discharge (Product)				
Solids (dry) product: NH_4VO_3 mass flowrate	0.575	t/h		Calculated
Solids product: NH_4VO_3 mass flowrate (including moisture)	0.638	t/h		Calculated
Liquid component in product mass flowrate	0.064	t/h		Calculated
Fraction of liquid reporting to solids	0.00296	-		Factor (calculated)
Composition of Liquid contained in solids product				
H_2O mass flowrate	0.061	t/h		
$(NH_4)_2SO_4$ mass flowrate	0.002763	t/h		
NH_4VO_3 mass flowrate	0.002	t/h		
Total	0.07	t/h		Sum
Liquids Discharge (to WWT)				
H_2O mass flowrate	20.562	t/h		
$(NH_4)_2SO_4$ mass flowrate	0.930	t/h		
NH_4VO_3 (solids) mass flowrate	0.000	t/h		
Total	21.49	t/h		Sum
SOLVENT EXTRACTION - PHOSPHORIC ACID				
IX Discharge Composition				
PLS mass flowrate	193.56	t/h		Mo, U & V removed
PLS Volumetric flowrate	179.22	m^3/h		Mo, U & V removed

INPUTS & CALCULATIONS SHEET

PLS Composition prior to IX					
PLS flowrate	194.40	t/h			
Ferric Leach feed liquids flowrate (H_2O, calcium acetate etc)	113.64	t/h			
Water - H_2O	63.74	t/h			
Sulphuric Acid - H_2SO_4	0.89	t/h			
Phosphoric Acid - H_3PO_4	12.48	t/h			
Magnesium Sulphate - $MgSO_4$	1.08	t/h			
Uranyl Sulphate - $UO_2.SO_4$	0.18	t/h			
Manganese Sulphate - $MnSO_4$	0.08	t/h			
Molybdenum Sulphate - $Mo(SO_4)_2$	0.13	t/h			
Vanadyl Sulphate - $VOSO_4$	0.65	t/h			
Nickel Sulphate - $NiSO_4$	0.43	t/h			
Zinc Sulphate - $ZnSO_4$	0.99	t/h			
Yttrium Sulphate - $Y_2(SO_4)_3$	0.11	t/h			
Neodymium Sulphate - $Nd_2(SO_4)_3$	0.01	t/h			
	194.41	t/h			
PLS Composition after IX					
Ferric Leach feed liquids flowrate (H_2O, calcium acetate etc)	113.64	t/h			
Water - H_2O	63.87	t/h		Adjust water by +0.13t/h.	
Sulphuric Acid - H_2SO_4	0.89	t/h			
Phosphoric Acid - H_3PO_4	12.48	t/h			
Magnesium Sulphate - $MgSO_4$	1.08	t/h			
Manganese Sulphate - $MnSO_4$	0.08	t/h			
Nickel Sulphate - $NiSO_4$	0.43	t/h			
Zinc Sulphate - $ZnSO_4$	0.99	t/h			
Yttrium Sulphate - $Y_2(SO_4)_3$	0.11	t/h			
Neodymium Sulphate - $Nd_2(SO_4)_3$	0.01	t/h			
Total	193.57	t/h			
SX Feed Composition					
SX Feed (PLS) mass flowrate IN	193.57	t/h			
SX Feed (PLS) volumetric flowrate IN	179.22	m^3/h		SG = 1.08	
SX Feed (PLS) SG	1.08				
Organic Solvent volumetric flowrate IN	180.00	m^3/h			
Organic Solvent mass flowrate IN	180.00	t/h		To be confirmed by R.Raiter	
Organic Solvent SG	1.00			Assumption.	
Fraction of H_3PO_4 transferred to Organic Phase	100.00%	%			
Stripping Agent - H_2O mass flowrate IN	52.00	t/h			
Volumetric flowrate of H_3PO_4	6.64	m^3/h			
Mass flowrate of Stripping agent transferred to PLS	12.48	t/h			
Volumetric flowrate of Stripping agent transferred to PLS	12.48	m^3/h		Identical to mass of H_3PO_4 (differing volume due to density)	
SX Product Composition					
PLS to RE Precipitation Tank					
SX Product (PLS) mass flowrate OUT (after stripping)	193.57	t/h			
SX Product (PLS) volumetric flowrate OUT (after stripping)	185.07	m^3/h			
Loaded Organic Solvent volumetric flowrate OUT	186.64	m^3/h			
Loaded Organic Solvent mass flowrate OUT	192.48	t/h			
Volumetric flowrate of H_3PO_4	6.64	m^3/h			
Mass flowrate of H_3PO_5	12.48	t/h			
Barren Organic Solvent volumetric flowrate OUT	180.00	m^3/h			
Barren Organic Solvent mass flowrate OUT	180.00	t/h			
Loaded stripping agent to H_3PO_4 Concentration					
Loaded stripping agent - H_2O and H_3PO_4 mass flowrate OUT	52.00	t/h			
Composition					
Water - H_2O	39.52	t/h			
Phosphoric Acid - H_3PO_4	12.48	t/h			
Total	52.00				
PLS Composition after SX					
Ferric Leach feed liquids flowrate (H_2O, calcium acetate etc)	113.64	t/h			
Water - H_2O	76.36	t/h			
Sulphuric Acid - H_2SO_4	0.89	t/h			
Phosphoric Acid - H_3PO_4	0	t/h			
Magnesium Sulphate - $MgSO_4$	1.08	t/h			
Manganese Sulphate - $MnSO_4$	0.08	t/h			
Nickel Sulphate - $NiSO_4$	0.43	t/h			
Zinc Sulphate - $ZnSO_4$	0.99	t/h			
Yttrium Sulphate - $Y_2(SO_4)_3$	0.11	t/h			
Neodymium Sulphate - $Nd_2(SO_4)_3$	0.01	t/h			
Total	193.57	t/h			
PHOSPHORIC ACID CONCENTRATION UPGRADE					
Feed Composition					
Water - H_2O	39.52	t/h			
Phosphoric Acid - H_3PO_4	12.48	t/h			
Total	52.00				
Product Composition					
Desired product concentration	83.70%	$\%^w/_w$			
Water - H_2O	2.43	t/h		Calculated by difference using desired product concentration	
Phosphoric Acid - H_3PO_4	12.48	t/h			
Total	14.91	t/h			
Mass of H_2O removed	37.09	t/h		Calculated	
Energy required to achieve water removal (from 25°C starting point)	26.51	MW		$Q = m \times [(C_p \times \Delta T) + \Delta H_{VAP}]$	
				C_p = 4.18 kJ/kg.K	Volumetric flowrate of diesel fuel required:
Consider Lower Calorific Value of Diesel Burning	43,400	kJ/kg		ΔH_{VAP} = 2260 kJ/kg	0.611 kg/s
RARE EARTH PRECIPITATION					Density of diesel fuel:
					0.832 kg/dm^3
Feed to RE Precipitation Tank					
Composition					
Ferric Leach feed liquids flowrate (H_2O, calcium acetate etc)	113.64	t/h			
Water - H_2O	76.36	t/h			
Sulphuric Acid - H_2SO_4	0.89	t/h			
Phosphoric Acid - H_3PO_4	0	t/h			
Magnesium Sulphate - $MgSO_4$	1.08	t/h			
Manganese Sulphate - $MnSO_4$	0.08	t/h			
Nickel Sulphate - $NiSO_4$	0.43	t/h			
Zinc Sulphate - $ZnSO_4$	0.99	t/h			
Yttrium Sulphate - $Y_2(SO_4)_3$	0.11	t/h			
Neodymium Sulphate - $Nd_2(SO_4)_3$	0.01	t/h			
Total	193.57	t/h			
Ammonium hydroxide consumption					
Ammonium hydroxide (100% NH_4OH) for neutralization mass flowrate	0.633	t/h			
Ammonium hydroxide (60% NH_4OH) for neutralization mass flowrate	1.055	t/h			
Ammonium hydroxide (100% NH_4OH) for precipitation mass flowrate	0.0045	t/h			
Ammonium hydroxide (60% NH_4OH) for precipitation mass flowrate	0.0076	t/h			
Ammonium hydroxide (100% NH_4OH) total mass flowrate	0.638	t/h			
Ammonium hydroxide (60% NH_4OH) total mass flowrate	1.063	t/h			
RE Precipitation Tank discharge					
Water - H_2O generated by neutralization reaction	0.33	t/h			
Ammonium Sulphate - $(NH_4)_2SO_4$ generated by neutralization reaction	1.19	t/h			
Ammonium Sulphate - $(NH_4)_2SO_4$ generated by precipitation reaction	0.01	t/h			

INPUTS & CALCULATIONS SHEET

Composition				
Ferric Leach feed liquids flowrate (H_2O, calcium acetate etc)	113.64	t/h		
Water - H_2O (inc generated amount + reagent dilution amount)	77.11	t/h		
Phosphoric Acid - H_3PO_4	0	t/h		
Magnesium Sulphate - $MgSO_4$	1.08	t/h		
Manganese Sulphate - $MnSO_4$	0.08	t/h		
Nickel Sulphate - $NiSO_4$	0.43	t/h		
Zinc Sulphate - $ZnSO_4$	0.99	t/h		
Yttrium Hydroxide - $Y(OH)_3$	0.063	t/h		Solids
Neodymium Hydroxide - $Nd(OH)_3$	0.008	t/h		Solids
Ammonium Sulphate - $(NH_4)_2SO_4$ (generated)	1.20	t/h		
Total	194.59	t/h		
S/L Separation				
Assume S/L Separation product contains $10\%^w/_w$ liquid	10.00%			Specified
Solids Discharge (Product)				
Solids (dry) product: $Y(OH)_3$ & $Nd(OH)_3$ combined mass flowrate	0.072	t/h		Calculated
Solids product: NH_4VO_3 mass flowrate (including moisture)	0.080	t/h		Calculated
Liquid component in product mass flowrate	0.008	t/h		Calculated
Fraction of liquid reporting to solids	0.00004	-		Factor (calculated)
Composition of Liquid contained in solids product				
Ferric Leach feed liquids flowrate (H_2O, calcium acetate etc) mass flowrate	0.004665	t/h		Calculated
H_2O mass flowrate	0.003165	t/h		Calculated
Magnesium Sulphate - $MgSO_4$	0.000044	t/h		Calculated
Manganese Sulphate - $MnSO_4$	0.000003	t/h		Calculated
Nickel Sulphate - $NiSO_4$	0.000018	t/h		Calculated
Zinc Sulphate - $ZnSO_4$	0.000041	t/h		Calculated
Ammonium Sulphate - $(NH_4)_2SO_4$ (generated)	0.000049	t/h		Calculated
Total	0.008	t/h		Sum
Liquids Discharge (to Mn, Ni & Zn SX)				
Ferric Leach feed liquids flowrate (H_2O, calcium acetate etc) mass flowrate	113.634	t/h		Calculated
H_2O mass flowrate	77.103	t/h		Calculated
Magnesium Sulphate - $MgSO_4$	1.075	t/h		Calculated
Manganese Sulphate - $MnSO_4$	0.076	t/h		Calculated
Nickel Sulphate - $NiSO_4$	0.432	t/h		Calculated
Zinc Sulphate - $ZnSO_4$	0.990	t/h		Calculated
Ammonium Sulphate - $(NH_4)_2SO_4$ (generated)	1.202	t/h		Calculated
Total	194.512	t/h		Sum
SOLVENT EXTRACTION - MANGANESE, NICKEL & ZINC				
SX Feed Composition				
SX Feed (PLS) mass flowrate IN	194.51	t/h		
SX Feed (PLS) volumetric flowrate IN	180.10	m^3/h	1.08	SG = 1.08
Organic Solvent mvolumetric flowrate IN	180.00	m^3/h		
Organic Solvent mass flowrate IN	180.00	t/h		R.Raiter to confirm.
Fraction of Mn, Ni & Zn transferred to Organic Phase	100.00%	%		
Stripping Agent - $10\%^w/_w$ H_2SO_4 volumetric flowrate IN	20.00	m^3/h		R.Raiter specified.
Stripping Agent - $10\%^w/_w$ H_2SO_4 mass flowrate IN	20.96	t/h	1.048	SG = 1.048 (refer compounds data sheet)
Volumetric flowrate of $MnSO_4$	0.02	m^3/h		
Volumetric flowrate of $NiSO_4$	0.11	m^3/h		
Volumetric flowrate of $ZnSO_4$	0.28	m^3/h		
Total	0.41	m^3/h		
Loaded Organic Solvent volumetric flowrate OUT	180.41	m^3/h		
Loaded Organic Solvent mass flowrate OUT	181.50	t/h		
Volumetric flowrate of Stripping agent transferred to PLS	0.41	m^3/h		To facilitate solvent extraction
Mass flowrate of Stripping agent transferred to PLS	0.43	t/h		
SX Product Composition				
PLS to RE Precipitation Tank				
SX Product (PLS) mass flowrate OUT	193.44	t/h		
SX Product (PLS) volumetric flowrate OUT	179.12	m^3/h		
Organic Solvent mass flowrate OUT	180.00	m^3/h		
Loaded stripping agent to Precipitation Tank				
Loaded stripping agent - H_2SO_4 and Mn, Ni & Zn compounds mass flowrate OUT	22.03	t/h		
Composition				
Water - H_2O	18.48	t/h		
Sulphuric Acid - H_2SO_4	2.05	t/h		
$MnSO_4$ mass flowrate	0.08	t/h		
$NiSO_4$ mass flowrate	0.43	t/h		
$ZnSO_4$ mass flowrate	0.99	t/h		
Total	22.03			
PLS Composition after SX (to WWT)				
Ferric Leach feed liquids flowrate (H_2O, calcium acetate etc)	113.63	t/h		
H_2O mass flowrate	77.49	t/h		
Sulphuric Acid - H_2SO_4 mass flowrate	0.04	t/h		
Phosphoric Acid - H_3PO_4 mass flowrate	0	t/h		
Magnesium Sulphate - $MgSO_4$ mass flowrate	1.08	t/h		
Manganese Sulphate - $MnSO_4$ mass flowrate	0	t/h		
Nickel Sulphate - $NiSO_4$ mass flowrate	0	t/h		
Zinc Sulphate - $ZnSO_4$ mass flowrate	0	t/h		
Yttrium Sulphate - $Y_2(SO_4)_3$ mass flowrate	0	t/h		
Neodymium Sulphate - $Nd_2(SO_4)_3$ mass flowrate	0	t/h		
Total	192.24	t/h		
MANGANESE, NICKEL & ZINC PRECIPITATION				
Feed to Mn, Ni & Zn Precipitation Tank				
Composition				
Water - H_2O	18.48	t/h		
Sulphuric Acid - H_2SO_4	2.05	t/h		
$MnSO_4$ mass flowrate	0.08	t/h		
$NiSO_4$ mass flowrate	0.43	t/h		
$ZnSO_4$ mass flowrate	0.99	t/h		
Total	22.03			
Neutralization Calculation				
$n(H_2SO_4)$ =	20.929	kg-moles		Calculated
Acid concentration (Molar) (Molar = moles/L)	1.13	M		Calculated
Therefore, concentration of H^+ = 1.13M = 1.13 mol/L = 1.13 g/L, (as Mr(H) = 1.0)				
pH = $-\log_{10}[H^+]$ = $-\log_{10}(1.13)$ = -0.053	0.00	pH		pH of elution liquor.
pH (elution liquor) = -0.053 (assume pH = 0)				
Now calculate $[H^+]$ at pH = 6 where precipitation occurs.				
pH = $-\log_{10}[H^+]$				Log Rule: $\log_B(B^x) = x.\log_B(B) = x$
pH = 6 = $-\log_{10}[H^+]$, solving for $[H^+]$: $[H^+] = 10^{-6}$ M				
$[H^+] = 10^{-6}$ M = 10^{-6} mol/L = 0.000001 g/L (as Mr(H) = 1.0)				
$100\%^w/_w$ H_2SO_4 concentration at pH = 6	0.000001	g/L		Calculated
To raise pH from pH = 0 to pH = 6.0, need to neutralize H^+:				
1.13g/L - 0.000001g/L =1.129999g/L = 1.129999 mol/L to be neutralized	1.129999	mol/L		Calculated
Moles of H+ to be neutralized (1.129999 x 18.48) =	20.882	kg-moles		Calculated
NH_4OH molar flowrate to achieve pH rise	2.260	mol/L		
NH_4OH molar flowrate to achieve pH rise	41.764	kg-moles		
NH_4OH mass flowrate to achieve pH rise	1.464	t/h		From reactions sheet
NH_4OH ($60\%^w/_w$) mass flowrate to achieve pH rise	2.440	t/h		Calculated
Residual $100\%^w/_w$ H_2SO_4 mass flowrate (0.000001g/L)	0.00464	t/h		Calculated by difference (input - consumption)

INPUTS & CALCULATIONS SHEET

Intermediate Composition (after neutralization but prior to precipitation)				
H_2O mass flowrate	20.205	t/h		Calculated (input + generation + reagents cpt = output)
100%$^w/_w$ H_2SO_4 mass flowrate	0.004640	t/h		Calculated (input - consumption = output)
Ammonium Sulphate - $(NH_4)_2SO_4$ mass flowrate	2.760	t/h		Generated
$MnSO_4$ mass flowrate	0.08	t/h		Unchanged from input
$NiSO_4$ mass flowrate	0.43	t/h		Unchanged from input
$ZnSO_4$ mass flowrate	0.99	t/h		Unchanged from input
Total	24.47	t/h		
RE Precipitation Tank discharge				
Ammonium carbonate consumption				
Ammonium carbonate (100% $(NH_4)_2CO_3$) for precipitation mass flowrate	2.3218	t/h		
Ammonium carbonate (60% $(NH_4)_2CO_3$) for precipitation mass flowrate	3.8697	t/h		
Composition			24.216	<-- Sum of liquids
H_2O mass flowrate	20.205	t/h		From above
100%$^w/_w$ H_2SO_4 mass flowrate	0.005	t/h		Residual
Ammonium Sulphate - $(NH_4)_2SO_4$ mass flowrate	4.006	t/h		Sum (from reactions sheet)
$MnCO_3$ mass flowrate (solids)	0.058	t/h		Solids
$NiCO_3$ mass flowrate (solids)	0.331	t/h		Solids
$ZnCO_3$ mass flowrate (solids)	0.769	t/h		Solids
Total	25.37	t/h		
Solids total mass flowrate	1.16	t/h		
Liquids total mass flowrate	24.22	t/h		
S/L Separation				
Assume S/L Separation product contains 10%$^w/_w$ liquid	10.00%			Specified
Solids Discharge (Product)				
Solids (dry) product: carbonates combined mass flowrate	1.158	t/h		Calculated
Solids product: carbonates mass flowrate (including moisture)	1.287	t/h		Calculated
Liquid component in product mass flowrate	0.129	t/h		Calculated
Fraction of liquid reporting to solids product	0.00531	-		Factor (calculated)
Composition of Liquid contained in solids product				
H_2O mass flowrate	0.107	t/h		
100%$^w/_w$ H_2SO_4 mass flowrate	0.00002	t/h		
Ammonium Sulphate - $(NH_4)_2SO_4$ mass flowrate	0.021	t/h		
Total	0.129	t/h		Sum
Liquids Discharge (to WWT)				
H_2O mass flowrate	20.098	t/h		Balance
100%$^w/_w$ H_2SO_4 mass flowrate	0.005	t/h		Balance
Ammonium Sulphate - $(NH_4)_2SO_4$ mass flowrate	3.984	t/h		Balance
Total	24.087	t/h		Sum
			24.216	<-- Check: Sum of liquids output in S/L Separation
WASTE WATER TREATMENT				
Sum of all WWT Liquids Discharges				
Sum of contaminants in WWT Liquids Discharges				

REACTIONS & CONVERSIONS SHEET

Reaction	Moles	Reactant 1	Moles	Reactant 2	Moles	Reactant 3	Moles	Product 1	Moles	Product 2	Moles	Product 3	Moles	Product 4
ACETIC ACID LEACH														

Calcite Reaction 1: $1\ CaCO_3 + 2\ CH_3COOH \longrightarrow 1\ Ca(CH_3COO)_2 + 1\ CO_2 + 1\ H_2O$
kmoles: 607.01 1,214.03 607.01 607.01 607.01

Reaction 2: $1\ NiS + 2\ CH_3COOH \longrightarrow 1\ Ni(CH_3COO)_2 + 1\ H_2S$
kmoles: 0.52 1.03 0.52 0.52
0.10 (10% conversion)
0.04688 t

Reaction 3: $2\ U_3O_8 + 12\ CH_3COOH + 1\ O_2 + 6\ H_2O \longrightarrow 6\ UO_2(CH_3COO)_2.2H_2O$
kmoles: 0.009 0.051 0.004 0.026 0.026
0.05 (5% conversion)
0.0072 t
7.19 kg

Compound	M_r	Mass (kg)	kmoles	
Reaction 1				
$CaCO_3$	100.09	60,756.0	607.01	
CO_2	44.01	26,714.7	607.01	
H_2O	18.02	10,938.4	607.01	
CH_3COOH	60.06	72,914.5	1,214.03	
$Ca(CH_3COO)_2$	158.18	96,017.4	607.01	
Reaction 2				
NiS	90.77	468.8	5.16	4.65
CH_3COOH	60.06	62.0	1.03	
$Ni(CH_3COO)_2$	176.80	91.3	0.52	
H_2S	34.09	17.6	0.52	
Reaction 3				
U_3O_8	842.09	143.8	0.17	
CH_3COOH	60.06	3.1	0.05	
H_2O	18.02	0.5	0.03	
$UO_2(CH_3COO)_2.2H_2O$	424.17	10.9	0.03	
O_2	32.00	0.137	0.0043	

Reaction	Moles	Reactant 1	Moles	Reactant 2	Moles	Reactant 3	Moles	Product 1	Moles	Product 2	Moles	Product 3	Moles	Product 4
ACETIC ACID REGENERATION & GYPSUM PRODUCTION														

Ideal Case:
Reaction 2: $1\ Ca(CH_3COO)_2 + 1\ H_2SO_4 + 2\ H_2O \longrightarrow 2\ CH_3COOH + 1\ CaSO_4.2H_2O$
kmoles: 607.01 607.01 1,214.03 1,214.03 607.01

Actual Case:
Reaction 1: $1\ Ca(CH_3COO)_2 + 1\ H_2SO_4 + 2\ H_2O \longrightarrow 2\ CH_3COOH + 1\ CaSO_4.2H_2O$
kmoles: 550.01 550.01 1,100.02 1,100.02 550.01

Compound	M_r	Mass (kg)	kmoles
Ideal Case			
$CaSO_4.2H_2O$	172.2	104,521.7	607.01
$Ca(CH_3COO)_2$	158.18	96,017.4	607.01
CH_3COOH	60.06	72,914.5	1,214.03
H_2SO_4	98.09	59,542.0	607.01
H_2O	18.02	21,876.8	1,214.03
Actual Case			
$CaSO_4.2H_2O$	172.19	94,706.3	550.01
$Ca(CH_3COO)_2$	158.18	87,000.6	550.01
CH_3COOH	60.06	66,067.2	1,100.02
H_2SO_4	98.09	53,950.5	550.01
H_2O	18.02	19,822.4	1,100.02

Reaction	Moles	Reactant 1	Moles	Reactant 2	Moles	Reactant 3	Moles	Product 1	Moles	Product 2	Moles	Product 3	Moles	Product 4
SULPHURIC ACID LEACH														

Fluorapatite: Reaction 1: $1\ Ca_5(PO_4)_3F + 5\ H_2SO_4 \longrightarrow 5\ CaSO_4 + 3\ H_3PO_4 + 1\ HF$
kmoles: 42.46 212.28 212.28 127.37 42.46
1 (100% conversion) gypsum

Dolomite:
Reaction 2: $1\ CaMg(CO_3)_2 + 2\ H_2SO_4 \longrightarrow 1\ CaSO_4 + 1\ MgSO_4 + 2\ CO_2 + 2\ H_2O$
kmoles: 8.93 17.86 8.93 8.93 17.86 17.86
1 (100% conversion) gypsum

Uranium 1:
Reaction 3: $1\ U_3O_8 \longrightarrow 3\ UO_2 + 1\ O_2$
kmoles: 0.16 0.48 0.16
0.98 (98.0% conversion) 133.91 kg

Uranium 2:
Reaction 4: $1\ UO_2 + 1\ Fe_2(SO_4)_3 \longrightarrow 1\ UO_2.SO_4 + 2\ FeSO_4$
kmoles: 0.48 0.48 0.48 0.95
1 (100% conversion)

Pyrolusite:
Reaction 5: $1\ MnO_2 + 2\ H_2SO_4 + 2\ FeSO_4 \longrightarrow 1\ MnSO_4 + 2\ Fe_2(SO_4)_3 + 2\ H_2O + 2\ O_2$
kmoles: 0.48 0.95 0.95 0.48 0.95 0.95 0.95
1 (100% conversion)

Molybdenum:
Reaction 6: $1\ MoS_2 + 3\ H_2SO_4 \longrightarrow 1\ Mo(SO_4)_3 + 2\ H_2S + 1\ H_2$
kmoles: 0.34 1.03 0.34 0.69 0.34
0.51 (51.0% conversion)

Vanadium:
Reaction 7: $1\ V_2O_5 + 2\ H_2SO_4 \longrightarrow 2\ VOSO_4 + 2\ H_2O + 0.5\ O_2$
kmoles: 2.00 3.99 3.99 3.99 1.00
0.73 (73.0% conversion)

Phosphorus:
Reaction 8: $2\ P_2O_5 + 6\ H_2SO_4 \longrightarrow 4\ H_3PO_4 + 6\ SO_2 + 3\ O_2$
kmoles:
0 (100% conversion) NOT RELEVANT

Nickel:
Reaction 9: $1\ NiS + 1\ H_2SO_4 \longrightarrow 1\ NiSO_4 + 1\ H_2S$
kmoles: 2.79 2.79 2.79 2.79
0.60 (60.0% conversion)

Zinc:
Reaction 10: $1\ ZnS + 1\ H_2SO_4 \longrightarrow 1\ ZnSO_4 + 1\ H_2S$
kmoles: 6.13 6.13 6.13 6.13
0.98 (98.0% conversion)

Yttrium:
Reaction 11: $1\ Y_2O_3 + 3\ H_2SO_4 \longrightarrow 1\ Y_2(SO_4)_3 + 3\ H_2O$
kmoles: 0.23 0.68 0.23 0.68
0.91 (91.0% conversion)

Neodymium:
Reaction 12: $1\ Nd_2O_3 + 3\ H_2SO_4 \longrightarrow 1\ Nd_2(SO_4)_3 + 3\ H_2O$
kmoles: 0.02 0.06 0.02 0.06
0.64 (64.0% conversion)

Compound	M_r	Mass (kg)	kmoles	
Reaction 1				
$Ca_5(PO_4)_3F$	504.3	21,411.0	42.46	
H_2SO_4	98.09	20,822.6	212.28	
$CaSO_4$	136.15	28,901.9	212.28	
H_3PO_4	98.00	12,482.1	127.37	
HF	20.01	849.5	42.46	
Reaction 2				
$CaMg(CO_3)_2$	184.41	1,647.0	8.93	
H_2SO_4	98.09	1,752.1	17.86	
$CaSO_4$	136.15	1,216.0	8.93	
$MgSO_4$	120.38	1,075.1	8.93	
CO_2	44.01	786.1	17.86	
H_2O	18.02	321.9	17.86	
Reaction 3				
U_3O_8	842.09	136.6	0.16	Mass = Initial mass - Consumed in AA Leach
UO_2	270.03	128.8	0.48	
O_2	32.00	5.1	0.16	
Reaction 4				
$Fe_2(SO_4)_3$	399.91	190.8	0.48	
$UO_2.SO_4$	366.09	174.7	0.48	
$FeSO_4$	151.91	144.9	0.95	
Reaction 5				
MnO_2	86.94	41.5	0.48	
H_2SO_4	98.09	93.6	0.95	
$FeSO_4$	151.91	144.9	0.95	
$Fe_2(SO_4)_3$	399.91	190.8	0.48	
$MnSO_4$	151.01	72.0	0.48	
Reaction 6				
MoS_2 (Feed)	160.10	108.1	0.68	
MoS_2 (Reacted)	160.10	55.1	0.34	
H_2SO_4	98.09	101.3	1.03	
$Mo(SO_4)_3$	384.17	132.3	0.34	
H_2S	34.09	23.5	0.69	
H_2	2.02	0.7	0.34	
Reaction 7				
V_2O_5 (Feed)	181.88	497.4	2.73	
V_2O_5 (Reacted)	181.88	363.1	2.00	
H_2SO_4	98.09	391.6	3.99	
$VOSO_4$	163.01	650.9	3.99	
H_2O	18.02	71.9	3.99	
O_2	32.00	31.9	1.00	
Reaction 8				
P_2O_5 (Feed)				
H_2SO_4	98.09	0	0	
H_3PO_4	98.00	0	0	
SO_2	64.07	0	0	
O_2	32.00	0	0	
Reaction 9				
NiS (Feed)	90.77	421.9	4.65	
NiS (Reacted)	90.77	253.2	2.79	
H_2SO_4	98.09	273.6	2.79	
$NiSO_4$	154.77	431.6	2.79	
H_2S	34.09	95.1	2.79	
Reaction 10				
ZnS (Feed)	97.45	610.0	6.26	
ZnS (Reacted)	97.45	597.8	6.13	
H_2SO_4	98.09	601.7	6.13	
$ZnSO_4$	161.45	990.4	6.13	
H_2S	34.09	209.1	6.13	
Reaction 11				
Y_2O_3 (Feed)	225.82	56.2	0.25	
Y_2O_3 (Reacted)	225.82	51.2	0.23	
H_2SO_4	98.09	66.7	0.68	
$Y_2(SO_4)_3$	466.03	105.6	0.23	
H_2O	18.02	12.3	0.68	
Reaction 12				
Nd_2O_3 (Feed)	336.48	11.3	0.034	
Nd_2O_3 (Reacted)	336.48	7.3	0.022	
H_2SO_4	98.09	6.4	0.06	
$Nd_2(SO_4)_3$	576.69	12.4	0.02	
H_2O	18.02	1.2	0.06	

Reaction	Moles	Reactant 1	Moles	Reactant 2	Moles	Reactant 3	Moles	Product 1	Moles	Product 2	Moles	Product 3	Moles	Product 4
MOLYBDENUM PRECIPITATION														

Neutralization: Reaction 1: $1\ H_2SO_4 + 2\ NH_4OH \longrightarrow 1\ (NH_4)_2SO_4 + 2\ H_2O$
kmoles: 0.00 0.00 0.00 0.00
1 (100.0% conversion)
REACTION NOT RELEVANT WHEN ELUTION OCCURS WITH NH_4OH (AS NO RESIDUAL H_2SO_4 TO NEUTRALIZE)

Precipitation: Reaction 2: $1\ (NH_4)_2MoO_4 + 1\ Ca(CH_3COO)_2 \longrightarrow 1\ CaMoO_4 + 2\ NH_4.CH_3COO$
kmoles: 0.34 0.34 0.34 0.69
1 (100.0% conversion)

Option A: use calcium acetate from recycle bleed stream

Precipitation: Reaction 3: $1\ Mo^{6+} + 1\ Ca(OH)_2 \longrightarrow 1\ CaMoO_4 + 2\ OH^-$
kmoles: 1.00 1.00 1.00 2.00
1 (100.0% conversion)

Option B: use calcium hydroxide (lime)

Compound	M_r	Mass (kg)	kmoles
Reaction 1			
H_2SO_4	98.09	0.0	0.00
NH_4OH	35.06	0.0	0.00
$(NH_4)_2SO_4$	132.17	0.0	0.00
H_2O	18.02	0.0	0.00
Reaction 2			
$(NH_4)_2MoO_4$	196.06	67.3	0.34
$Ca(CH_3COO)_2$	158.18	54.3	0.34
$CaMoO_4$	200.04	68.7	0.34
$NH_4.CH_3COO$	77.10	52.9	0.69
Reaction 3			
Mo^{6+}	32.07	0.0	0.00
$Ca(OH)_2$	195.27	195.3	1.00
$CaMoO_4$	96.11	96.1	1.00

REACTIONS & CONVERSIONS SHEET

URANIUM PRECIPITATION

	Reaction	Moles	Reactant 1	Moles	Reactant 2	Moles	Reactant 3	Moles	Product 1	Moles	Product 2	Moles	Product 3	Moles	Product 4
Raise to pH=4.	Reaction 1:			1	H_2SO_4 +	2	NH_4OH ⟶	1	$(NH_4)_2SO_4$ +	2	H_2O				
	kmoles:			7.06		14.13		7.06		14.13					
1	(100.0% conversion)														
Precipitation:	Reaction 2:	1	UO_2^{2+} +	1	H_2O_2 +	2	H_2O ⟶	1	$UO_4 . 2H_2O$ +	2	H^+ +	1	O_2		
	kmoles:	0.45		0.45		0.90		0.45		0.90		0.45			
1	(100.0% conversion)														

Compound	M_r	Mass (kg)	kmoles
Reaction 1			
H_2SO_4	98.09	692.9	7.06
NH_4OH	35.06	495.3	14.13
$(NH_4)_2SO_4$	132.17	933.6	7.06
H_2O	18.02	254.6	14.13
Reaction 2			
UO_2^{2+}	270.03	121.5	0.45
H_2O_2	34.02	15.3	0.45
H_2O	18.02	16.2	0.90
$UO_4 .2H_2O$	338.07	152.1	0.45
H_2O	18.02	16.2	0.90
O_2	32.00	14.4	0.45

VANADIUM PRECIPITATION

	Reaction	Moles	Reactant 1	Moles	Reactant 2	Moles	Reactant 3	Moles	Product 1	Moles	Product 2	Moles	Product 3	Moles	Product 4
Raise to pH=7.	Reaction 1:			1	H_2SO_4 +	2	NH_4OH ⟶	1	$(NH_4)_2SO_4$ +	2	H_2O				
(from pH=4)	kmoles:			0.00002		0.00004		0.00002		0.00004					
1	(100.0% conversion)														
Precipitation:	Reaction 2:			1	V_2O_5 +	2	NH_4OH ⟶	2	NH_4VO_3 +	1	H_2O				
	kmoles:			2.46		4.91		4.91		2.46					
1	(100.0% conversion)														

Compound	M_r	Mass (kg)	kmoles
Reaction 1			
H_2SO_4	98.09	0.0020	0.00002
NH_4OH	35.06	0.0014	0.00004
$(NH_4)_2SO_4$	132.17	0.0027	0.00002
H_2O	18.02	0.0007	0.00004
Reaction 2			
V_2O_5	181.88	446.6	2.46
NH_4OH	35.06	172.2	4.91
NH_4VO_3	116.99	574.6	4.91
H_2O	18.02	44.3	2.46

RARE EARTH PRECIPITATION

	Reaction	Moles	Reactant 1	Moles	Reactant 2	Moles	Reactant 3	Moles	Product 1	Moles	Product 2	Moles	Product 3	Moles	Product 4
Neutralization	Reaction 1:			1	H_2SO_4 +	2	NH_4OH ⟶	1	$(NH_4)_2SO_4$ +	2	H_2O				
	kmoles:			9.03		18.06		9.03		18.06					
	(100.0% conversion)														
Precipitation:	Reaction 2:			1	$Y_2(SO_4)_3$ +	6	NH_4OH ⟶	2	$Y(OH)_3$ +	3	$(NH_4)_2SO_4$				
	kmoles:			0.23		1.36		0.45		0.68					
1	(100.0% conversion)														
Precipitation:	Reaction 3:			1	$Nd_2(SO_4)_3$ +	6	NH_4OH ⟶	2	$Nd(OH)_3$ +	3	$(NH_4)_2SO_4$				
	kmoles:			0.022		0.129		0.043		0.065					
1	(100.0% conversion)														

Compound	M_r	Mass (kg)	kmoles
Reaction 1			
H_2SO_4	98.09	885.51	9.03
NH_4OH	35.06	633.01	18.06
$(NH_4)_2SO_4$	132.17	1,193.17	9.03
H_2O	18.02	325.35	18.06
Reaction 2			
$Y_2(SO_4)_3$	466.03	105.62	0.23
NH_4OH	35.06	47.7	1.36
$Y(OH)_3$	139.94	63.4	0.45
$(NH_4)_2SO_4$	132.17	89.9	0.68
Reaction 3			
$Nd_2(SO_4)_3$	576.69	12.4	0.022
NH_4OH	35.06	4.5	0.13
$Nd(OH)_3$	195.27	8.4	0.04
$(NH_4)_2SO_4$	132.17	8.6	0.06

MANGANESE, NICKEL & ZINC PRECIPITATION

	Reaction	Moles	Reactant 1	Moles	Reactant 2	Moles	Reactant 3	Moles	Product 1	Moles	Product 2	Moles	Product 3	Moles	Product 4
Neutralization:	Reaction 1:			1	H_2SO_4 +	2	NH_4OH ⟶	1	$(NH_4)_2SO_4$ +	2	H_2O				
	kmoles:			20.88		41.76		20.88		41.76					
	(100.0% conversion)														
Precipitation:	Reaction 2:			1	$MnSO_4$ +	1	$(NH_4)_2CO_3$ ⟶	1	$MnCO_3$ +	1	$(NH_4)_2SO_4$				
	kmoles:			0.50		0.50		0.50		0.50					
	(100.0% conversion)														
Precipitation:	Reaction 3:			1	$NiSO_4$ +	1	$(NH_4)_2CO_3$ ⟶	1	$NiCO_3$ +	1	$(NH_4)_2SO_4$				
	kmoles:			2.79		2.79		2.79		2.79					
	(100.0% conversion)														
Precipitation:	Reaction 4:			1	$ZnSO_4$ +	1	$(NH_4)_2CO_3$ ⟶	1	$ZnCO_3$ +	1	$(NH_4)_2SO_4$				
	kmoles:			6.13		6.13		6.13		6.13					
	(100.0% conversion)														

Compound	M_r	Mass (kg)	kmoles
Reaction 1			
H_2SO_4	98.09	2,048.32	20.882
NH_4OH	35.06	1,464.25	41.76
$(NH_4)_2SO_4$	132.17	2,759.97	20.88
H_2O	18.02	752.59	41.76
Reaction 2			
$MnSO_4$	151.01	75.83	0.50
$(NH_4)_2CO_3$	96.11	48.26	0.50
$MnCO_3$	114.95	57.72	0.50
$(NH_4)_2SO_4$	132.17	66.37	0.50
Reaction 3			
$NiSO_4$	154.77	431.63	2.79
$(NH_4)_2CO_3$	96.11	268.0	2.79
$NiCO_3$	118.71	331.1	2.79
$(NH_4)_2SO_4$	132.17	368.6	2.79
Reaction 4			
$ZnSO_4$	161.45	990.4	6.13
$(NH_4)_2CO_3$	96.11	589.6	6.13
$ZnCO_3$	125.39	769.2	6.13
$(NH_4)_2SO_4$	132.17	810.8	6.13

WASTE WATER TREATMENT

Reaction	Moles	Reactant 1	Moles	Reactant 2	Moles	Reactant 3	Moles	Product 1	Moles	Product 2	Moles	Product 3	Moles	Product 4

Neutralization of excess acid

The Uranium Discovery Company

U3O8 CORP

tenova MINING & MINERALS

REAGENT CONSUMPTION SHEET

Reaction	Moles	Reactant 1	Moles	Reactant 2	Moles	Reactant 3	Moles	Product 1	Moles	Product 2	Moles	Product 3	Moles	Product 4

ACETIC ACID & GLACIAL ACETIC ACID MAKEUP REQUIREMENT

Calcite Consumption

Calcite Reaction 1: 1 $CaCO_3$ + 2 CH_3COOH ⟶ 1 $Ca(CH_3COO)_2$ + 1 CO_2 + 1 H_2O

kmoles: 607.01 | 1,214.03 | 607.01 | 607.01 | 607.01

Acetic Acid Regeneration

Actual Case:

Reaction 1: 1 $Ca(CH_3COO)_2$ + 1 H_2SO_4 + 2 H_2O ⟶ 2 CH_3COOH + 1 $CaSO_4.2H_2O$

kmoles: 550.01 | 550.01 | 1,100.02 | 1,100.02 | 550.01

Compound	M_r	Mass (kg)	kmoles
Reaction 1			
$CaCO_3$	100.09	60,756.0	607.01
CO_2	44.01	26,714.7	607.01
H_2O	18.02	10,938.4	607.01
CH_3COOH	60.06	72,914.5	1,214.03
$Ca(CH_3COO)_2$	0.00	0.0	607.01
Actual Case			
$CaSO_4.2H_2O$	172.19	94,706.3	550.01
$Ca(CH_3COO)_2$	158.18	87,000.6	550.01
CH_3COOH	60.06	66,067.2	1,100.02
H_2SO_4	98.09	53,950.5	550.01
H_2O	18.02	19,822.4	1,100.02
TOTAL		6,847.25	kg/h

Consumption - Generation = Makeup Amount : 6,847.25 kg/h

Reaction	Moles	Reactant 1	Moles	Reactant 2	Moles	Reactant 3	Moles	Product 1	Moles	Product 2	Moles	Product 3	Moles	Product 4

SULPHURIC ACID CONSUMPTION

Application	Consumption (t/h)
AA regeneration	53.95
Fluorapatite leach	20.82
Dolomite leach	1.75
Pyrolusite reaction	0.09
Molybdenum reaction	0.10
Vanadium reaction	0.39
Nickel reaction	0.27
Zinc reaction	0.60
Yttrium reaction	0.07
Neodymium reaction	0.01
U IX elution	0.24
V IX elution	0.47
Mn, Ni, Zn SX strip	2.10
TOTAL	**80.86** t/h

** This table summarizes the reactions whereby sulfuric acid is consumed throughout various applications in the processing plant.*

Reaction	Moles	Reactant 1	Moles	Reactant 2	Moles	Reactant 3	Moles	Product 1	Moles	Product 2	Moles	Product 3	Moles	Product 4

PYROLUSITE CONSUMPTION

Ferric Leach - Conversion of Ferrous to Ferric Iron

Reaction 5: 1 MnO_2 + 2 H_2SO_4 + 2 $FeSO_4$ ⟶ 1 $MnSO_4$ + 2 $Fe_2(SO_4)_3$ + 2 H_2O + 2 O_2

kmoles: 2.82 | 5.64 | 5.64 | 2.82 | 5.64 | 5.64 | 5.64

1 (100% conversion)

Sum of ferrous iron produced by ferric reactions below.

Compound	M_r	Mass (kg)	kmoles
Reaction 5			
MnO_2	86.94	245.0	2.82
H_2SO_4	98.09	552.8	5.64
$FeSO_4$	151.91	856.1	5.64
$Fe_2(SO_4)_3$	399.91	2,253.7	5.64
$MnSO_4$	151.01	425.5	2.82
TOTAL		244.97	kg/h

Reaction	Moles	Reactant 1	Moles	Reactant 2	Moles	Reactant 3	Moles	Product 1	Moles	Product 2	Moles	Product 3	Moles	Product 4

AMMONIA CONSUMPTION

Molybdenum Precipitation

Neutralization: Reaction 1: 1 H_2SO_4 + 2 NH_4OH ⟶ 1 $(NH_4)_2SO_4$ + 2 H_2O

kmoles: 0.00 | 0.00 | 0.00 | 0.00

1 (100.0% conversion)

Uranium Precipitation

Raise to pH=4 Reaction 1: 1 H_2SO_4 + 2 NH_4OH ⟶ 1 $(NH_4)_2SO_4$ + 2 H_2O

kmoles: 7.06 | 14.13 | 7.06 | 14.13

1 (100.0% conversion)

Vanadium Precipitation

Raise to pH=7 Reaction 1: 1 H_2SO_4 + 2 NH_4OH ⟶ 1 $(NH_4)_2SO_4$ + 2 H_2O

(from pH=4) kmoles: 0.00002 | 0.00004 | 0.00002 | 0.00004

1 (100.0% conversion)

Rare Earth Precipitation

Neutralization Reaction 1: 1 H_2SO_4 + 2 NH_4OH ⟶ 1 $(NH_4)_2SO_4$ + 2 H_2O

kmoles: 9.03 | 18.06 | 9.03 | 18.06

(100.0% conversion)

Precipitation: Reaction 2: 1 $Y_2(SO_4)_3$ + 6 NH_4OH ⟶ 2 $Y(OH)_3$ + 3 $(NH_4)_2SO_4$

kmoles: 0.23 | 1.36 | 0.45 | 0.68

1 (100.0% conversion)

Precipitation: Reaction 3: 1 $Nd_2(SO_4)_3$ + 6 NH_4OH ⟶ 2 $Nd(OH)_3$ + 3 $(NH_4)_2SO_4$

kmoles: 0.022 | 0.129 | 0.043 | 0.065

1 (100.0% conversion)

Manganese, Nickel & Zinc Precipitation

Neutralization: Reaction 1: 1 H_2SO_4 + 2 NH_4OH ⟶ 1 $(NH_4)_2SO_4$ + 2 H_2O

kmoles: 20.88 | 41.76 | 20.88 | 41.76

(100.0% conversion)

Molybdenum Elution

4.00% NH_4OH solution, delivered at : 2.24 t/h

Mass of NH_4OH in 4.00%$^w/_w$ solution = 0.090 t/h

Compound	M_r	Mass (kg)	kmoles
Reaction 1			
H_2SO_4	98.09	0.0	0.00
NH_4OH	35.06	0.0	0.00
$(NH_4)_2SO_4$	132.17	0.0	0.00
H_2O	18.02	0.0	0.00
Reaction 1			
H_2SO_4	98.09	692.9	7.06
NH_4OH	35.06	495.3	14.13
$(NH_4)_2SO_4$	132.17	933.6	7.06
H_2O	18.02	254.6	14.13
Reaction 1			
H_2SO_4	98.09	0.0020	0.00002
NH_4OH	35.06	0.0014	0.00004
$(NH_4)_2SO_4$	132.17	0.0027	0.00002
H_2O	18.02	0.0007	0.00004
Reaction 1			
H_2SO_4	98.09	885.51	9.03
NH_4OH	35.06	633.01	18.06
$(NH_4)_2SO_4$	132.17	1,193.17	9.03
H_2O	18.02	325.35	18.06
Reaction 2			
$Y_2(SO_4)_3$	466.03	105.6	0.23
NH_4OH	35.06	47.7	1.36
$Y(OH)_3$	139.94	63.4	0.45
$(NH_4)_2SO_4$	132.17	89.9	0.68
Reaction 3			
$Nd_2(SO_4)_3$	576.69	12.4	0.022
NH_4OH	35.06	4.5	0.129
$Nd(OH)_3$	195.27	8.4	0.043
$(NH_4)_2SO_4$	132.17	8.6	0.065
Reaction 1			
H_2SO_4	98.09	2,048.32	20.882
NH_4OH	35.06	1,464.25	41.76
$(NH_4)_2SO_4$	132.17	2,759.97	20.88
H_2O	18.02	752.59	41.76
NH_4OH	35.06	89.51	2.55
TOTAL		2,734.28	kg/h

Mass of NH_4OH in 60.00%$^w/_w$ solution = 4,557.14 kg/h

Reaction	Moles	Reactant 1	Moles	Reactant 2	Moles	Reactant 3	Moles	Product 1	Moles	Product 2	Moles	Product 3	Moles	Product 4

HYDROGEN PEROXIDE CONSUMPTION

Uranium Precipitation

Precipitation: Reaction 2: 1 UO_2^{2+} + 1 H_2O_2 + 2 H_2O ⟶ 1 $UO_4 . 2H_2O$ + 2 H^+ + 1 O_2

kmoles: 0.45 | 0.45 | 0.90 | 0.45 | 0.90 | 0.45

1 (100.0% conversion)

Compound	M_r	Mass (kg)	kmoles
Reaction 2			
UO_2^{2+}	270.03	121.5	0.45
H_2O_2	34.02	15.3	0.45
H_2O	18.02	16.2	0.90
$UO_4 .2H_2O$	338.07	152.1	0.45
H_2O	18.02	16.2	0.90
O_2	32.00	14.4	0.45
TOTAL		15.31	kg/h

Reaction	Moles	Reactant 1	Moles	Reactant 2	Moles	Reactant 3	Moles	Product 1	Moles	Product 2	Moles	Product 3	Moles	Product 4

AMMONIUM CARBONATE CONSUMPTION

Manganese, Nickel & Zinc Precipitation

Precipitation: Reaction 2: 1 $MnSO_4$ + 1 $(NH_4)_2CO_3$ ⟶ 1 $MnCO_3$ + 1 $(NH_4)_2SO_4$

kmoles: 0.50 | 0.50 | 0.50 | 0.50

(100.0% conversion)

Precipitation: Reaction 3: 1 $NiSO_4$ + 1 $(NH_4)_2CO_3$ ⟶ 1 $NiCO_3$ + 1 $(NH_4)_2SO_4$

kmoles: 2.79 | 2.79 | 2.79 | 2.79

(100.0% conversion)

Precipitation: Reaction 4: 1 $ZnSO_4$ + 1 $(NH_4)_2CO_3$ ⟶ 1 $ZnCO_3$ + 1 $(NH_4)_2SO_4$

kmoles: 6.13 | 6.13 | 6.13 | 6.13

(100.0% conversion)

Compound	M_r	Mass (kg)	kmoles
Reaction 2			
$MnSO_4$	151.01	75.83	0.50
$(NH_4)_2CO_3$	96.11	48.26	0.50
$MnCO_3$	114.95	57.72	0.50
$(NH_4)_2SO_4$	132.17	66.37	0.50
Reaction 3			
$NiSO_4$	154.77	431.63	2.79
$(NH_4)_2CO_3$	96.11	268.0	2.79
$NiCO_3$	118.71	331.1	2.79
$(NH_4)_2SO_4$	132.17	368.6	2.79
Reaction 4			
$ZnSO_4$	161.45	990.4	6.13
$(NH_4)_2CO_3$	96.11	589.6	6.13
$ZnCO_3$	125.39	769.2	6.13
$(NH_4)_2SO_4$	132.17	810.8	6.13
TOTAL		905.85	kg/h

REAGENT CONSUMPTION SHEET

Reaction	Moles	Reactant 1	Moles	Reactant 2	Moles	Reactant 3	Moles	Product 1	Moles	Product 2	Moles	Product 3	Moles	Product 4

FERRIC SULPHATE CONSUMPTION

Uranium 2:

Reaction 4: 1 UO_2 + 1 $Fe_2(SO_4)_3$ ⟶ 1 $UO_2.SO_4$ + 2 $FeSO_4$

kmoles: 0.48 | 0.48 | 0.48 | 0.95

1 (100% conversion)

Compound	M_r	Mass (kg)	kmoles
Reaction 4			
UO_2	270.03	128.8	0.48
$Fe_2(SO_4)_3$	399.91	190.8	0.48
$UO_2.SO_4$	366.09	174.7	0.48
$FeSO_4$	151.91	144.9	0.95

Vanadium: Reaction 1: 1 V^{3+} + 2 Fe^{3+} ⟶ 2 Fe^{2+} + 1 V^{5+}

kmoles: 2.00 | 3.99 | 3.99 | 2.00

(100% conversion)

Compound	M_r	Mass (kg)	kmoles
Reaction 1			
V^{3+}	181.88	363.1	2.00
$Fe_2(SO_4)_3$	399.91	1,596.7	3.99
V^{5+}	163.01	325.4	2.00
$FeSO_4$	151.91	606.5	3.99

Molybdenum: Reaction 1: 1 Mo^{4+} + 2 Fe^{3+} ⟶ 2 Fe^{2+} + 1 Mo^{6+}

kmoles: 0.34 | 0.69 | 0.69 | 0.34

(100% conversion)

Compound	M_r	Mass (kg)	kmoles
Reaction 1			
Mo^{4+}	160.10	55.1	0.34
$Fe_2(SO_4)_3$	399.91	275.4	0.69
Mo^{6+}	384.17	132.3	0.34
$FeSO_4$	151.91	104.6	0.69

At 60%$^w/_w$ solution $Fe_2(SO_4)_3$, consumption of reagent is: 3,438.14 kg/h

TOTAL 2,062.88 kg/h

Total $FeSO_4$ produced 856.08 kg/h

Reaction	Moles	Reactant 1	Moles	Reactant 2	Moles	Reactant 3	Moles	Product 1	Moles	Product 2	Moles	Product 3	Moles	Product 4

SCRAP IRON CONSUMPTION

Reaction 1: 1 H^+ + 1 Fe^0 + 1 Fe^{3+} ⟶ 2 Fe^{2+} + 1 H^+

kmoles: 16.11 | 16.11 | 32.23

(100% conversion)

Compound	M_r	Mass (kg)	kmoles
Reaction 1			
Fe^0	55.85	900.0	16.11
Fe^{3+}	55.85	900.0	16.11
Fe^{2+}	55.85	1,800.0	32.23

Rate of Fe consumption: 5.0g Fe per dm^3 of PLS (180m^3/hr) As specified by R.Raiter.

TOTAL 900.00 kg/h

Reaction	Moles	Reactant 1	Moles	Reactant 2	Moles	Reactant 3	Moles	Product 1	Moles	Product 2	Moles	Product 3	Moles	Product 4

CALCIUM HYDROXIDE PRODUCTION

Reaction 1: 1 CaO + 1 H_2O ⟶ 1 $Ca(OH)_2$

kmoles: 5.00 | 5.00 | 5.00

1.0 (100.0% conversion)

Compound	M_r	Mass (kg)	kmoles
Reaction 1			
CaO	56.08		
H_2O	18.02		
$Ca(OH)_2$	74.10		

Reaction	Moles	Reactant 1	Moles	Reactant 2	Moles	Reactant 3	Moles	Product 1	Moles	Product 2	Moles	Product 3	Moles	Product 4

FLOCCULANT CONSUMPTION

Assumed flocculant consumption rate: 30.00 g/t

Processing Area	Solids (t/h) Flowrate	Number of Thickeners	Floc to TK1 (g/t)	Floc to TK2 (g/t)	Total Floc per t solids	Total Floc Required (g/h)
AA Leach	61.19	2	15.0	15.0	30.0	1,835.7
AA Regeneration	94.71	1	30.0	0	30.0	2,841.2
Ferric Leach Wash	61.19	1	30.0	0	30.0	1,835.7
Ferric Leach CCD	68.68	2	30.0	15.0	45.0	3,090.4
Mo Precipitation	0.07	1	30.0	0	30.0	2.1
Uranium Precipitation	0.15	1	30.0	0	30.0	4.6
Vanadium Precipitation	0.57	1	30.0	0	30.0	17.2
					TOTAL	9,626.8 g/h

Reaction	Moles	Reactant 1	Moles	Reactant 2	Moles	Reactant 3	Moles	Product 1	Moles	Product 2	Moles	Product 3	Moles	Product 4

SULPHUR CONSUMPTION (SULPHURIC ACID PRODUCTION)

Reaction 1: 1 S + 1 O_2 ⟶ 1 SO_2

kmoles: 937.91 | 937.91 | 937.91

Reaction 2: 1 SO_2 + 0.5 O_2 ⟶ 1 SO_3

kmoles: 937.91 | 468.9571 | 937.91

Reaction 3: 1 SO_3 + 1 H_2O ⟶ 1 H_2SO_4

kmoles: 937.91 | 937.91 | 937.91

Note:

Compound	M_r	Mass (kg)	kmoles
Reaction 3			
H_2SO_4	98.09	92,000	937.91
H_2O	18.02	16,901	937.91
SO_3	80.07	75,099	937.91
Reaction 2			
SO_3	80.07	75,099	937.91
O_2	32.00	15,007	468.96
SO_2	64.07	60,092	937.91
Reaction 1			
SO_2	64.07	60,092	937.91
O_2	32.00	30,013	937.91
S	32.07	30,079	937.91

Reaction	Moles	Reactant 1	Moles	Reactant 2	Moles	Reactant 3	Moles	Product 1	Moles	Product 2	Moles	Product 3	Moles	Product 4

PYRITE CONSUMPTION (FERRIC PRODUCTION)

Reaction 1: 4 FeS_2 + 11 O_2 ⟶ 2 Fe_2O_3 + 8 SO_2

kmoles: 20.00 | 55.01 | 10.00 | 40.01

Reaction 2: 1 Fe_2O_3 + 3 H_2SO_4 ⟶ 1 $Fe_2(SO_4)_3$ + 3 H_2O

kmoles: 10.00 | 30.01 | 10.00 | 30.01

SULPHURIC ACID PRODUCTION FROM SO_2 OFFGAS OF PYRITE COMBUSTION

Reaction 3: 1 SO_2 + 0.5 O_2 ⟶ 1 SO_3

kmoles: 40.01 | 20.00 | 40.01

Reaction 4: 1 SO_3 + 1 H_2O ⟶ 1 H_2SO_4

kmoles: 40.01 | 40.01 | 40.01

Compound	M_r	Mass (kg)	kmoles
Reaction 2			
$Fe_2(SO_4)_3$	399.91	4,000	10.00
H_2O	18.02	541	30.01
H_2SO_4	98.09	2,943	30.01
Fe_2O_3	159.70	1,597	10.00
Reaction 1			
Fe_2O_3	159.70	1,597	10.00
SO_2	64.07	2,563	40.01
O_2	32.00	1,760	55.01
FeS_2	119.99	2,400	20.00
Reaction 3			
SO_2	64.07	2,563	40.01
O_2	32.00	640	20.00
SO_3	80.07	3,204	40.01
Reaction 4			
SO_3	80.07	3,204	40.01
H_2O	18.02	721	40.01
H_2SO_4	98.09	3,924	40.01

Reaction	Moles	Reactant 1	Moles	Reactant 2	Moles	Reactant 3	Moles	Product 1	Moles	Product 2	Moles	Product 3	Moles	Product 4

PYRITE CONSUMPTION (POTENTIALLY USE AS SOURCE OF SULPHURIC ACID)

Reaction 1: 3 FeS_2 + 8 O_2 ⟶ 1 Fe_3O_4 + 6 SO_2

kmoles: 468.96 | 937.91

Reaction 2: 2 SO_2 + 1 O_2 ⟶ 2 SO_3

kmoles: 937.91 | 937.91

Reaction 3: 1 SO_3 + 1 H_2O ⟶ 1 H_2SO_4

kmoles: 937.91

Compound	M_r	Mass (kg)	kmoles
Reaction 1			
H_2SO_4	98.09	92,000	937.91
FeS_2	119.99	56,270	468.96

92 tph sulphuric acid required

92 tph Sulphuric = 937.91 kmoles/hr

468.96 kmoles/hr Pyrite = 56,270.16 kg/hr

U3O8 CORP. PRELIMINARY ECONOMIC ASSESSMENT ON THE BERLIN DEPOSIT, COLOMBIA

Report Prepared for

U3O8 Corp.

APPENDIX I: OPTION B PROCESSING PLANT MASS BALANCE

Report Prepared by

Tenova Mining & Minerals (Australia) Pty Ltd

M6088.A

This document is not controlled when printed.

The Uranium Discovery Company

U3O8 CORPORATION

BERLIN PROJECT - OPTION B

PEA STUDY

PROCESSING PLANT MASS BALANCE

M6088.A-P150-003

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0	25/01/2013	ISSUED FOR USE	J.SCHLOFFER	P.NIEMANN	L.DE KLERK	A.WALKER
B	11/01/2013	ISSUED FOR ICR	J.SCHLOFFER	P.NIEMANN	L.DE KLERK	
A	19/12/2012	ISSUED FOR IDR	J.SCHLOFFER	P.NIEMANN		
REV	**DATE**	**REVISION HISTORY**	**ORIGINATED**	**CHECKED**	**APPROVED**	**TENOVA APPROVAL**

COMPOUNDS DATA SHEET

FEED COMPOSITION

Ore Mass Flowrate 122.0 t/h

MineralName	Formula	% w/w	PPM	Mr	Mass (tph)	Al	C	Ca	Cl	F	Fe	H	K
Calcite	$CaCO_3$	49.80		100.09	60.756		0.119992007	0.400439604					
Fluorapatite	$Ca_5(PO_4)_3F$	17.55		504.31	21.411			0.397374631		0.037675239			
Quartz	SiO_2	15.25		60.09	18.600								
Muscovite	$KAl_3Si_3O_{11}.H_2O$	8.50		398.33	10.370	0.20319835						0.005071172	0.09815982
Chlorite	$NaClO_2$	3.50		90.44	4.270				0.391972579				
Pyrite	FeS_2	2.50		119.99	3.050						0.465455455		
Dolomite	$CaMg(CO_3)_2$	1.35		184.41	1.647		0.13025324	0.217341793					
Sphalerite	ZnS	0.50		97.45	0.610								
Triuranium Octoxide	U_3O_8	0.12	1,179	842.09	0.1438								
Vanadium Pentoxide	V_2O_5	0.41	4,077	181.88	0.497								
Yttrium Oxide	Y_2O_3	0.05	461	225.82	0.056								
Neodymium (III) Oxide	Nd_2O_3	0.01	93	336.48	0.011								
Nickel Sulfide	NiS	0.38	3,843	90.77	0.469								
Molybdenum Disulfide	MoS_2	0.09	886	160.1	0.108								
	TOTAL	100.00			122.00	2.11	7.50	33.20	1.67	0.81	1.42	0.05	1.02

MineralName	Mg	Mn	Mo	N	Na	Nd	Ni	O	P	S	Si	U	V	Y	Zn	TOTAL		Mass Flowrate
Calcite								0.479568								1.00		60.76
Fluorapatite								0.380718	0.184232							1.00		21.41
Quartz								0.532535			0.467465					1.00		18.60
Muscovite								0.482012			0.211558					1.00		10.37
Chlorite					0.254202			0.353826								1.00		4.27
Pyrite										0.534545						1.00		3.05
Dolomite	0.13182582							0.520579								1.00		1.65
Sphalerite										0.329092					0.670908	1.00		0.61
Triuranium Octoxide								0.152003				0.847997				1.00		0.14
Vanadium Pentoxide								0.43985					0.56015			1.00		0.50
Yttrium Oxide								0.212559						0.787441		1.00		0.06
Neodymium (III) Oxide						0.8573466		0.142653								1.00		0.01
Nickel Sulfide							0.646689			0.353311						1.00		0.47
Molybdenum Disulfide			0.59937539							0.400625						1.00		0.11
TOTAL	0.22	0.00	0.06	0.00	1.09	0.01	0.30	54.81	3.94	2.04	10.89	0.122	0.28	0.04	0.41	SUM	122.00 t/h	122.00

COMPOUNDS DATA

MineralName	Formula	Mr	S.G.		
Pyrite	FeS_2	119.99	5.02		
Oxygen (gas)	O_2	32.00	-		
Magnetite	Fe_3O_4	231.55	5.17		
Sulphur dioxide (gas)	SO_2	64.07	-		
Sulphur trioxide (gas)	SO_3	80.07	-		
Sulphuric Acid	H_2SO_4	98.09	1.84		
Water	H_2O	18.02	1.00		
Pyrolusite	MnO_2	86.94	5.08		
Acetic Acid	CH_3COOH	60.06	1.05		
Ferric Sulphate	$Fe_2(SO_4)_3$	399.91	3.10		
Calcite	$CaCO_3$	100.09	2.71		
Calcium Acetate	$Ca(CH_3COO)_2$	158.18	1.51		
Carbon Dioxide	CO_2	44.01	-		
Gypsum	$CaSO_4.2H_2O$	172.19	2.32		
Nickel Sulphide	NiS	90.77	5.40		
Nickel Acetate	$Ni(CH_3COO)_2$	176.80	1.80		
Hydrogen Sulfide	H_2S	34.09	-		
Triuranium Octoxide	U_3O_8	842.09	8.30		
Uranyl Acetate Dihydride	$UO_2(CH_3COO)_2\ 2H_2O$	424.17	2.89		
Calcium Sulphate	$CaSO_4$	136.15	2.32		
Phosphoric Acid	H_3PO_4	98.00	1.88		
Hydrogen Fluoride	HF	20.01	-		
Magnesium Sulphate	$MgSO_4$	120.38	2.66		
Uranium Dioxide	UO_2	270.03	10.97		
Uranyl Sulphate	$UO_2.SO_4$	366.10	3.28		
Ferrous Sulphate	$FeSO_4$	151.91	2.84		
Manganese Sulphate	$MnSO_4$	151.01	3.25		
Molybdenum Trisulphate	$Mo(SO_4)_3$	384.17	5.00		
Vanadyl Sulphate	$VOSO_4$	163.01	3.00		
Nickel Sulphate	$NiSO_4$	154.77	4.01		
Zinc Sulphate	$ZnSO_4$	161.45	3.54		
Yttrium Sulphate	$Y_2(SO_4)_3$	466.03	2.70		
Neodymium Sulphate	$Nd_2(SO_4)_3$	576.69	2.85		
Ammonium Hydroxide	NH_4OH	35.06	0.88		
Ammonium Sulphate	$(NH_4)_2SO_4$	132.17	1.77		
Ammonium meta-Vanadate	NH_4VO_3	116.99	2.33		
Calcium Hydroxide	$Ca(OH)_2$	74.10	2.21		
Calcium Molybdate	$CaMoO_4$	200.04	4.35		
Calcium Oxide	CaO	56.08	3.30		
Uranium Peroxide Hydrate	$UO_4\ .2H_2O$	338.07	4.67		
Hydrogen Peroxide	H_2O_2	34.02	1.45		
Hydrogen gas	H_2	2.02	-		
Yttrium Hydroxide	$Y(OH)_3$	139.94	4.50		
Neodymium Hydroxide	$Nd(OH)_3$	195.27	7.50		
Ammonium Carbonate	$(NH_4)_2CO_3$	96.11	1.50		
Manganese Carbonate	$MnCO_3$	114.95	3.12		
Nickel Carbonate	$NiCO_3$	118.71	4.39		
Zinc Carbonate	$ZnCO_3$	125.39	4.45		
Hematite	Fe_2O_3	159.70	5.24		
Ammonium Molybdate	$(NH_4)_2MoO_4$	196.06	2.45		
Ammonium Chloride	NH_4Cl	53.50	1.53		
Calcium Chloride	$CaCl_2$	110.98	2.15		
Ammonium Acetate	$NH_4.CH_3COO$	77.10	1.17		
ShellSol	-	-	0.75		
TBP	-	-	0.98		
D2EHPA	-	-	0.97		

0.1368

ELEMENTAL MOLECULAR WEIGHTS

Element	Mr	Valence Number	Density
Al	26.98		
C	12.01		
Ca	40.08		
Cl	35.45		
F	19.00		
Fe	55.85		
H	1.01		
K	39.10		
Mg	24.31		
Mn	54.94		
Mo	95.96	6	10.28
N	14.01		
Na	22.99		
Nd	144.24		
Ni	58.70		
O	16.00		
P	30.97		
S	32.07		
Si	28.09		
U	238.03	6	19.10
V	50.94	5	6.00
Y	88.91		
Zn	65.38		
V_2O_5	-	2	3.36
H_2SO_4	98.09	2	
NH_4OH	35.06	1	

Compound	Molecular Formula	Mr	Element of Interest	Fraction
Uranyl Acetate Dihydride	$UO_2(CH_3COO)_2\ 2H_2O$	424.17	U	0.561
Nickel Acetate	$Ni(CH_3COO)_2$	176.80	Ni	0.332
Uranyl Sulphate	$UO_2.SO_4$	366.09	U	0.650
Vanadyl Sulphate	$VOSO_4$	163.01	V	0.312
Molybdenum Trisulphate	$Mo(SO_4)_3$	384.17	Mo	0.250
Phosphoric Acid	H_3PO_4	98.00	P	0.316
Nickel Sulphate	$NiSO_4$	154.77	Ni	0.379
Zinc Sulphate	$ZnSO_4$	161.45	Zn	0.405
Manganese Sulphate	$MnSO_4$	151.01	Mn	0.364
Yttrium Sulphate	$Y_2(SO_4)_3$	466.03	Y	0.382
Neodymium Sulphate	$Nd_2(SO_4)_3$	576.69	Nd	0.500
Pyrolusite	MnO_2	86.94	Mn	0.632
Calcium Molybdate	$CaMoO_4$	200.04	Mo	0.480
Ammonium meta-Vanadate	NH_4VO_3	116.99	V	0.435

COMPOUNDS DATA SHEET

SLURRY & REAGENT SG DATA

CALCULATION TABLE

Slurry	Solids Mass (t)	Solids (%w/w)	Slurry Mass (t)	Liquids Mass (t)	Solids SG	Liquid SG	Slurry Volume (m^3)	Slurry SG	Compound 1 (solid)	Compound 2 (liquid)	Solids (%v/v)
Ferric Sulphate	1.012	60.00%	1.69	0.67	3.1	1.0	1.001	1.685	$Fe_2(SO_4)_3$	H_2O	32.61%
Ferric Sulphate	2.073	60.00%	3.45	1.38	3.1	1.0	2.051	1.685	$Fe_2(SO_4)_4$	H_2O	32.61%
Pyrolusite	0.836	50.00%	1.67	0.836	5.08	1.0	1.001	1.671	MnO_2	H_2O	16.45%
AA Leach Pulp	122.000	12.00%	1016.67	894.667	3.5	1.0	929.524	1.094	Ore	AA and H_2O	3.75%
Sulfuric Acid	0.105	10.00%	1.05	0.943	1.84	1.0	1.000	1.048	H_2SO_4	H_2O	5.69%
Ammonium Hydroxide	0.600	60.00%	1.00	0.400	0.88	1.0	1.082	0.924	NH_4OH	H_2O	63.03%
Ammonium Hydroxide	0.501	60.00%	0.84	0.334	0.88	1.0	0.903	0.924	NH_4OH	H_2O	63.03%
Ammonium Hydroxide	0.716	4.00%	17.90	17.186	0.88	1.0	18.000	0.995	NH_4OH	H_2O	4.52%
Hydrogen Peroxide	0.300	30.00%	1.00	0.700	1.45	1.0	0.907	1.103	H_2O_2	H_2O	22.81%
Ammonium Carbonate	0.600	60.00%	1.00	0.400	1.50	1.0	0.800	1.250	$(NH_4)_2CO_3$	H_2O	50.00%

CALCULATION TABLE

H_3PO_4 SX Organic Basis of calculation: 1.0 m^3

Reagent	(%v/v)	Volume (m^3)	S.G.	Mass (t)	
ShellSol	25.0%	0.25	0.75	0.188	
TBP	75.0%	0.75	0.98	0.735	
		1.00		0.923	Organic S.G

Mn/Ni/Zn SX Organic Basis of calculation: 1.0 m^3

Reagent	(%v/v)	Volume (m^3)	S.G.	Mass (t)	
ShellSol	70.0%	0.70	0.75	0.525	
D2EHPA	30.0%	0.30	0.97	0.291	
		1.00		0.816	Organic S.G

INPUTS & CALCULATIONS SHEET

Description	Value	Units	Area	Notes
OPERATING SCHEDULES				
Days in full year	365	d		
Hours in full day	24	h		
Total hours in a full year	8,760	h		
Anticipated operating hours in a full year	8,200	h		
WATER SPECIFICATIONS				
Water S.G.	1.0			
Water Density	1.0	t/m^3		
ROM FEED				
ROM material flowrate (dry solids)	1,000,400	t/yr		
ROM material flowrate (dry solids)	122.00	t/h		
ROM material moisture content	10.00%	% w/w		
GRIZZLY SCREEN				
Anticipated operating availability	100.00%	%		
Feed Material (dry solids)	122.00	t/h		
PRIMARY CRUSHING				
Anticipated operating availability	85.00%	%		
Jaw Crusher operational feed flowrate (dry solids)	143.53	t/h		
Jaw Crusher average feed flowrate (dry solids)	122.00	t/h		
Jaw Crusher moisture content in feed material	10.00%	% w/w		
STOCKPILE				
Stockpile feed flowrate (dry solids)	122.00	t/h		
Stockpile material moisture content	10.00%	% w/w		
PRIMARY GRINDING				
SAG Mill fresh feed flowrate (dry solids)	122.00	t/h		
SAG Mill feed material moisture content	10.00%	% w/w		
SAG Mill recycle ratio	150.00%	%		
Combined SAG Mill feed flowrate (dry solids) (fresh + recycle)	305.00	t/h		
Combined SAG Mill discharge flowrate (dry solids)	305.00	t/h		
SCREENING				
Screen feed flowrate (dry solids)	305.00	t/h		
Screen feed material moisture content	10.00%	% w/w		
Approximate Mass Split to U/S	0.40			
Screen U/S flowrate (dry solids)	122.00	t/h		Calculated
Screen O/S flowrate (dry solids)	183.00	t/h		Calculated
Screen U/S flowrate (total)	135.56	t/h		Calculated
Screen U/S flowrate (liquid)	13.56	t/h		Calculated
HYDROCYCLONES				
Cyclone Feed flowrate (solids)	366.00	t/h		Equivalent to Ball Mill Recycle of 200% + Cyclone O/F
Cyclone Feed flowrate (Total)	697.14	t/h		
Cyclone Feed solids density	52.50%	% w/w		Calculated
Cyclone O/F flowrate (solids)	122.00	t/h		From Screen U/S flowrate
Cyclone O/F flowrate (liquids)	226.57	t/h		Calculated by difference between Total and Solids
Cyclone O/F flowrate (Total)	348.57	t/h		Calculated
Cyclone O/F solids density	35.00%	% w/w		Specified
Cyclone U/F flowrate (solids)	244.00	t/h		Equivalent to Ball Mill Recycle of 200%
Cyclone U/F flowrate (liquids)	104.57	t/h		Calculated by difference b/t solids and total flowrates
Cyclone U/F flowrate (Total)	348.57	t/h		Calculated from U/F solids density
Cyclone U/F solids density	70.00%	% w/w		Specified (ball mill feed density required)
Fraction of feed material reporting to O/F	33.33%	% w/w		To be back calculated
SECONDARY GRINDING				
BALL MILL SUMP FEED				
Ball Mill Sump solids density	52.50%	% w/w		Taken from Cyclone Feed solids density
Solids flowrate	366.00	t/h		Screen U/F + Ball Mill product
Total Mass flowrate to Ball Mill Sump	697.14	t/h		Calculated from solids flowrate and density
Total Liquids flowrate	331.14	t/h		Calculated difference b/t total and solids flowrates
Ball Mill Sump Dilution Process Water (H_2O) including Screen Spray Bar Water	213.02	t/h		
SOLIDS				
Screen U/S solids	122.00	t/h		Known
Ball Mill solids discharge	244.00	t/h		From cyclone U/F
Total Solids	366.00	t/h		
LIQUIDS				
Screen U/S entrained water	13.56	t/h		Known
Ball Mill discharge liquid	104.57	t/h		From cyclone U/F
Fixed Liquids	118.13			
Total Ball mill sump feed flowrate	484.13	t/h		Sum of solids and liquids
BALL MILL				
Fraction of solids reporting directly to Ferric Leach without entering Ball Mill	0.00%	% w/w		Assume zero initially but confirm figure with Bruno
Ball Mill recycle ratio	200.00%	%		Specified
Ball Mill solids pulp density	70.00%	% w/w		Specified
Ball mill sump discharge flowrate	484.13	t/h		From ball mill sump feed
Ball Mill sump solids pulp density	52.50%	% w/w		Taken from cyclone feed density
SULPHURIC ACID LEACH				

Check Solids %: 75.60%

INPUTS & CALCULATIONS SHEET

Description	Value	Units	Area	Notes
Feed				
Sulphuric Acid Leach feed slurry flowrate (TOTAL)	348.57	t/h		From Cyclone O/F
Sulphuric Acid Leach feed flowrate (liquids)	226.57	t/h		From Cyclone O/F
Sulphuric Acid Leach feed flowrate (solids)	122.00	t/h		From Cyclone O/F
Sulphuric Acid Leach slurry feed solids calculated density	35.00%	% $^w/_w$		Calculated
Sulphuric Acid Leach slurry feed solids target density	35.00%	% $^w/_w$		Specified
REAGENTS				
H_2SO_4 flowrate	87.49	t/h		Stoichiometric consumption + 1 ton additional excess
Pyrolusite - MnO_2 (50%$^w/_w$ slurry) flowrate	1.67	t/h		Calculated in Compounds Data Sheet (1m^3/hr, R.Raiter)
Ferric Sulphate - $Fe_2(SO_4)_3$ (60%$^w/_w$ slurry) flowrate	3.45	t/h		Calculated in Reagent Consumption Sheet
Total Solids & Liquids Feed flowrate	441.19	t/h		
FEED SOLIDS COMPOSITION				
Calcite - $CaCO_3$	60.756			
Fluorapatite - $Ca_5(PO_4)_3F$	21.411			
Quartz - SiO_2	18.600			
Muscovite - $KAl_3Si_3O_{11}.H_2O$	10.370			
Chlorite - $NaClO_2$	4.270			
Pyrite - FeS_2	3.050			
Dolomite - $CaMg(CO_3)_2$	1.647			
Sphalerite - ZnS	0.610			
Triuranium Octoxide - U_3O_8	0.144			
Vanadium Pentoxide - V_2O_5	0.497			
Yttrium Oxide - Y_2O_3	0.056			
Neodymium (III) Oxide - Nd_2O_3	0.011			
Nickel Sulfide - NiS	0.469			
Molybdenum Disulfide - MoS_2	0.108			
Total	122.00			
Generation and Consumption				

COMPOUND	INPUT (t/h)	GENERATION (t/h)	CONSUMPTION (t/h)	OUTPUT (t/h) (INPUT + GENERATION - CONSUMPTION)			
SOLIDS							
Calcite - $CaCO_3$	60.756	0	60.756	0.000			
Fluorapatite - $Ca_5(PO_4)_3F$	21.411	0	21.411	0.000			
Quartz - SiO_2	18.600	0	0	18.600			
Muscovite - $KAl_3Si_3O_{11}.H_2O$	10.370	0	0	10.370			
Chlorite - $NaClO_2$	4.270	0	0	4.270			
Pyrite - FeS_2	3.050	0	0	3.050			
Dolomite - $CaMg(CO_3)_2$	1.647	0	1.647	0.000			
Sphalerite - ZnS	0.610	0	0.598	0.012			
Triuranium Octoxide - U_3O_8	0.144	0	0.141	0.003			
Vanadium Pentoxide - V_2O_5	0.497	0	0.363	0.134			
Yttrium Oxide - Y_2O_3	0.056	0	0.051	0.005			
Neodymium (III) Oxide - Nd_2O_3	0.011	0	0.007	0.004			
Nickel Sulfide - NiS	0.469	0	0.281	0.188			
Molybdenum Disulfide - MoS_2	0.108	0	0.055	0.053			
Subtotal	122.000	0	85.311	36.689	36.689		
Ferric Sulphate - $Fe_2(SO_4)_3$	2.073	0.402	2.073	0.402			
Ferrous Sulphate - $FeSO_4$	0	0.864	0.153	0.711			
Pyrolusite - MnO_2	0.836	0	0.044	0.792			
Uranium Dioxide - UO_2	0	0.136	0.136	0.000			
Gypsum - $CaSO_4$	0	112.763	0	112.763			
TOTAL	124.909	114.164	87.715	151.357	151.357		
LIQUIDS	**Input**	**Generation**	**Consumption**	**Output**			
Ferric Leach feed liquids flowrate (H_2O)	226.571	0	0	226.571			
Water within pyrolusite & ferric sulphate reagents	2.218	0	0	2.218			
Sulphuric Acid - H_2SO_4	87.490	0	86.490	1.000			
Water - H_2O	0	11.364	0	11.364			
Phosphoric Acid - H_3PO_4	0	12.482	0	12.482			
Magnesium Sulphate - $MgSO_4$	0	1.075	0	1.075			
Uranyl Sulphate - $UO_2.SO_4$	0	0.184	0	0.184			
Manganese Sulphate - $MnSO_4$	0	0.076	0	0.076			
Molybdenum Sulphate - $Mo(SO_4)_2$	0	0.132	0	0.132			
Vanadyl Sulphate - $VOSO_4$	0	0.651	0	0.651			
Nickel Sulphate - $NiSO_4$	0	0.480	0	0.480			
Zinc Sulphate - $ZnSO_4$	0	0.990	0	0.990			
Yttrium Sulphate - $Y_2(SO_4)_3$	0	0.106	0	0.106			
Neodymium Sulphate - $Nd_2(SO_4)_3$	0	0.012	0	0.012			
TOTAL	316.280	27.552	86.490	257.341	257.341		
GAS	**Input**	**Generation**	**Consumption**	**Output**			
Hydrogen Fluoride - HF	0	0.850	0	0.850			
Carbon Dioxide - CO_2	0	27.501	0	27.501			
Oxygen - O_2	0	0.069	0	0.069			
Hydrogen Sulfide - H_2S	0	0.338	0	0.338			
Hydrogen - H_2	0	0.001	0	0.001			
TOTAL	0.000	28.759	0.000	28.759	28.759		
						Discrepancy:	-3.731
TOTAL OF ALL PHASES (SOLID + LIQUID + GAS)	441.188	170.474	174.206	437.457		Discrepancy:	-0.85%
			-3.731				
Total INPUT to Leaching Tanks (Solids + Liquids)	441.19						

Description	Value	Units	Area	Notes
SOLID/LIQUID SEPARATION - CCD CIRCUIT				
CCD Feed				
Solids flowrate to CCD from leach discharge	151.36	t/h		From reactions calculations summary table above
Liquid flowrate to CCD from leach discharge	257.34	t/h		From reactions calculations summary table above
Total Slurry to CCD from leach discharge	408.70	t/h		Calculated
Slurry pulp density to CCD	37.03%	% $^w/_w$		Calculated
Mo IX Wash Water Recycle (Wash Number 1)	4.50	t/h		Check this - 36m3 per cycle, but per hour rate is less.
U& V IX Wash Water Recycle (Wash Number 1)	13.50	t/h		Check this - 36m3 per cycle, but per hour rate is less.
Subtotal Liquids to CCD	275.34	t/h		Sum
CCD wash water flowrate	150.00	t/h		Specified
Total Liquids to CCD	425.34	t/h		Sum

INPUTS & CALCULATIONS SHEET

Description	Value	Units	Area	Notes
PLS Volumetric flowrate	245.09	m^3/h		
PLS S.G	1.05			Specified by P.Miller
PLS mass flowrate	257.34	t/h		Calculated
CCD U/F Solids density	47.39%	% $^w/_w$		Calculated
Assumed PLS losses to tailings	0.00%	% $^w/_w$		Specified
Factor to account for PLS losses to U/F (tailings)	1.00	-		
Discharge - CCD O/F				
PLS flowrate	257.34	t/h		Calculated
Fraction of solids to O/F	0.00	% $^w/_w$		
PLS Composition				
Sulphuric Acid Leach feed liquids flowrate (H_2O)	226.571	t/h		
Water - H_2O	13.582	t/h		Calculated (Balance of PLS - components in PLS)
Sulphuric Acid - H_2SO_4	1.000	t/h		
Phosphoric Acid - H_3PO_4	12.482	t/h		
Magnesium Sulphate - $MgSO_4$	1.075	t/h		
Uranyl Sulphate - $UO_2.SO_4$	0.184	t/h		
Manganese Sulphate - $MnSO_4$	0.076	t/h		
Molybdenum Sulphate - $Mo(SO_4)_2$	0.132	t/h		
Vanadyl Sulphate - $VOSO_4$	0.651	t/h		
Nickel Sulphate - $NiSO_4$	0.480	t/h		
Zinc Sulphate - $ZnSO_4$	0.990	t/h		
Yttrium Sulphate - $Y_2(SO_4)_3$	0.106	t/h		
Neodymium Sulphate - $Nd_2(SO_4)_3$	0.012	t/h		
	257.341	t/h		
Discharge - CCD U/F				
Solids flowrate to U/F	151.36	t/h		100% solids report to U/F
Liquids flowrate to U/F	168.00	t/h		Calculated
Total	319.36	t/h		
Liquids Composition				
Water - H_2O	168.00	t/h		Calculated by difference b/t liquids to U/F and other cpts
Sulphuric Acid Leach feed liquids flowrate (H_2O, calcium acetate etc)	0.000	t/h		
Sulphuric Acid - H_2SO_4	0.000	t/h		
Phosphoric Acid - H_3PO_4	0.000	t/h		
Magnesium Sulphate - $MgSO_4$	0.000	t/h		
Uranyl Sulphate - $UO_2.SO_4$	0.000	t/h		
Manganese Sulphate - $MnSO_4$	0.000	t/h		
Molybdenum Sulphate - $Mo(SO_4)_2$	0.000	t/h		
Vanadyl Sulphate - $VOSO_4$	0.000	t/h		
Nickel Sulphate - $NiSO_4$	0.000	t/h		
Zinc Sulphate - $ZnSO_4$	0.000	t/h		
Yttrium Sulphate - $Y_2(SO_4)_3$	0.000	t/h		
Neodymium Sulphate - $Nd_2(SO_4)_3$	0.000	t/h		
	168.00	t/h		
ION EXCHANGE				
PLS mass flowrate	257.34	t/h		From above
PLS S.G	1.05			Specified by P.Miller
PLS Volumetric flowrate	245.09	m^3/h		
PLS ASSAY CALCULATED RESULTS:				

Element	Element Mr	Compound	Compound Mr	Element Mass %	Calculated PLS kg/h	Calculated PLS g/L	Calculated PLS PPM
Mo	95.96	$Mo(SO_4)_2$	384.17	24.98%	33.04	0.135	134.8
U	238.03	$UO_2.SO_4$	366.10	65.02%	119.53	0.488	487.7
V	50.94	$VOSO_4$	163.01	31.25%	203.39	0.830	829.9
P	30.97	H_3PO_4	98.00	31.60%	3,944.59	16.095	16,094.7
Nd	144.24	$Nd_2(SO_4)_3$	576.69	50.02%	6.23	0.025	25.4
Y	88.91	$Y_2(SO_4)_3$	466.03	38.16%	40.30	0.164	164.4
Mn	54.94	$MnSO_4$	151.01	36.38%	27.59	0.113	112.6
Ni	58.70	$NiSO_4$	154.77	37.93%	181.90	0.742	742.2
Zn	65.38	$ZnSO_4$	161.45	40.50%	401.07	1.636	1,636.4

Description	Value	Units	Area	Notes
ION EXCHANGE - MOLYBDENUM				
Loading				
Resin capacity (C_r)	1.00	eq.Me/L		
Column volume	18.00	m^3		Specified
				Column size calculation: Flowrate = 10 x Bed Volumes / hour
Mo flowrate contained in PLS	0.135	g/L		Specified
Mo flowrate contained in PLS	33,037.99	g/h		Calculated
1 x Equivalent of Mo Metal	15.99	eq.g		Calculation: 1 x Equivalent gram = (Mr / Valence #)
Equivalent grams of Mo Metal flowrate	2,065.74	eq.g/h		Calculated
Duration required to fully load $18m^3$ column	8.0	h		
Number of IX columns required	3.0	-		
Total Mo loaded to IX columns per 8.0 hours	264,303.9	g		
Elution				
Volumetric Elution Flowrates				
Step 1. Displacement water (Wash Number 1) mass flowrate	36.00	m^3/cycle		2 x Bed Volumes (3 x cycles per day)
Step 2. Alkaline elution (4.0%$^w/_w$ NH_4OH) mass flowrate	18.00	m^3/cycle		1 x Bed Volume (3 x cycles per day)
Step 3. Displacement water (Wash Number 2) mass flowrate	36.00	m^3/cycle		2 x Bed Volumes (3 x cycles per day)
Mass Flowrates				
Step 1. Displacement water (Wash Number 1) mass flowrate	36.00	t/cycle		2 x Bed Volumes - (recycle to CCD Circuit)
Step 2. Alkaline elution (4.0%$^w/_w$ NH_4OH) mass flowrate	17.90	t/cycle		1 x Bed Volume
Step 3. Displacement water (Wash Number 2) mass flowrate	36.00	t/cycle		2 x Bed Volumes
Discharge to Mo Precipitation Tank				
Alkaline elution (4.0%$^w/_w$ NH_4OH) mass flowrate	17.90	t/cycle		Calculated
Displacement water (Wash Number 2) mass flowrate	36.00	t/cycle		
Mo metal	0.26	t/cycle		Hours per cycle x Hourly flowrate of metal
Total	54.17	t/cycle		
Average Hourly Mass Flowrates				
Alkaline elution (4.0%$^w/_w$ NH_4OH) mass flowrate	2.24	t/h		96% water, 4% acid

INPUTS & CALCULATIONS SHEET

Description	Value	Units	Area	Notes
Displacement water (Wash Number 1) mass flowrate	4.50	t/h		
Displacement water (Wash Number 2) mass flowrate	4.50	t/h		
Mo metal	0.0330	t/h		100% of Mo assumed recovery
Total	11.27	t/h		
MOLYBDENUM PRECIPITATION				
Mo Elution Liquor				
NH_4OH (4.0% $^w/_w$) mass flowrate	0.0895	t/h		
Water mass flowrate	11.15	t/h		
n(NH_4OH) =	2.5531	kg-moles		
Concentration (Molar)	0.23	M		
Number of moles of Molybdenum n(Mo) =	0.3443	kg-moles		Mo is valence 6^+.
Resin capacity (C_r)	1.00	eq.Me/L		
Equivalent grams of NH_4OH/L	35.06	eq.g		Calculation: 1 x Equivalent gram = (Mr / Valence #)
Moles NH_4OH consumed in eluting Mo^{6+}	0.07	kg-moles		Calculated
Remaining NH_4OH (not consumed by elution) to be neutralised	2.48	kg-moles		Input - consumption
Precipitation				
NH_4OH mass flowrate required to bring solution to pH = 7	0.000	t/h		Calculated
NH_4OH (4.0% $^w/_w$ solution) mass flowrate	0.000	t/h		Calculated
** pH is alkaline already, no requirement for raising pH.*				
Slurry solids density	20.00%			
Lime consumed in Mo precipitation $Ca(OH)_2$	0.026	t/h		
Water contained in 20% lime slurry	0.102	t/h		
Total	0.128			
Solids Precipitated				
$CaMoO_4$ mass flowrate	0.069	t/h		From Reactions & Conversions precipitation calculation
Composition of Elution Liquor flowing to Precipitation Tank				
Water mass flowrate	11.15	t/h		
NH_4OH mass flowrate	0.0167	t/h		
$(NH_4)_2MoO_4$	0.068	t/h		
$Ca(OH)_2$	0.026	t/h		
Water contained within Ca(OH)2 reagent slurry (20% $^w/_w$ solids)	0.102	t/h		
Total	11.360	t/h		
Discharge to S/L Separation				
Solids				
$CaMoO_4$ mass flowrate	0.069	t/h		From above
Liquids				
Water mass flowrate (inc that contained within lime slurry)	11.250	t/h		
NH_4OH mass flowrate	0.041	t/h		
Total (Solids & Liquids)	11.360	t/h		
Solids density to S/L Separation	0.61%	% $^w/_w$		Calculated
S/L Separation				
Moisture content of $CaMoO_4$ product	2.00%	% $^w/_w$		Specified
$CaMoO_4$ total mass flowrate (inc moisture)	0.070	t/h		Calculated
Moisture flowrate in $CaMoO_4$ product	0.001	t/h		Calculated
Solids content of material reporting to WWT	0.00%	% $^w/_w$		Specified
Mass flowrate of material reporting to WWT	11.290	t/h		Balance by calculation
ION EXCHANGE - URANIUM & VANADIUM				
V Loading				
PLS mass flowrate	257.31	t/h		Mo removed
PLS Volumetric flowrate	245.08	m^3/h		Mo removed
Column volume	18.00	m^3		Calculated by solving for bed volume with known flowrate Calculation of column size: Flowrate = 10 x Bed Volumes / hour
V flowrate contained in PLS	0.830	g/L		Specified
V flowrate contained in PLS	203,389	g/h		Calculated
V_2O_5 flowrate contained in PLS	1.482	g/L		Specified
V_2O_5 flowrate contained in PLS	363,098	g/h		Calculated
1 x Equivalent of V Metal (V_2O_5)	90.94	eq.g		Calculation: 1 x Equivalent gram = (Mr / Valence #)
Equivalent grams of V Metal flowrate	3,992.72	eq.g/h		Calculated
Duration required to fully load 18m^3 column	4.0	h		Specified
Number of IX columns required	6	-		Specified (2 columns per 8 hour duration)
Total V_2O_5 loaded to IX columns per 4.0 hours	1,452,390.5	g		Calculated
U Loading				
PLS mass flowrate	257.10	t/h		Mo & V removed
PLS Volumetric flowrate	245.05	m^3/h		Mo & V removed
Column volume	18.00	m^3		Calculated by solving for bed volume with known flowrate Calculation of column size: Flowrate = 10 x Bed Volumes / hour
U flowrate contained in PLS	0.488	g/L		Specified
U flowrate contained in PLS	119,531	g/h		Calculated
1 x Equivalent of U Metal	39.67	eq.g		Calculation: 1 x Equivalent gram = (Mr / Valence #)
Equivalent grams of U Metal flowrate	3,013.02	eq.g/h		Calculated
Duration required to fully load 18m^3 column	8.0	h		Specified
Number of IX columns required	3	-		Specified
Total U loaded to IX columns per 8.0 hours	956,251.8	g		Calculated
Discharge Liquid to SX				
PLS mass flowrate	256.62	t/h		Mo, U & V removed
PLS Volumetric flowrate	244.89	m^3/h		Mo, U & V removed
V Elution				
Volumetric Elution Flowrates				
Step 1. Displacement water (Wash Number 1) mass flowrate	36.00	m^3/cycle		2 x Bed Volumes (3 x cycles per day with 2 columns)
Step 2. Acid elution (10% $^w/_w$ H_2SO_4) mass flowrate	18.00	m^3/cycle		1 x Bed Volume (3 x cycles per day with 2 columns)
Step 3. Displacement water (Wash Number 2) mass flowrate	36.00	m^3/cycle		2 x Bed Volumes (3 x cycles per day with 2 columns)

INPUTS & CALCULATIONS SHEET

Description	Value	Units	Area	Notes
Mass Flowrates				
Step 1. Displacement water (Wash Number 1) mass flowrate	36.00	t/cycle		2 x Bed Volumes - (recycle to CCD Circuit)
Step 2. Acid elution (10%$^w/_w$ H_2SO_4) mass flowrate	18.86	t/cycle		1 x Bed Volume
Step 3. Displacement water (Wash Number 2) mass flowrate	36.00	t/cycle		2 x Bed Volumes
Discharge to V Precipitation Tank				
Acid elution (10%$^w/_w$ H_2SO_4) mass flowrate	18.86	t/cycle		Calculated
Displacement water (Wash Number 2) mass flowrate	36.00	t/cycle		
V_2O_5 mass flowrate	1.45	t/cycle		Hours per cycle x Hourly flowrate of metal
Total	56.31	t/cycle		
Average Hourly Mass Flowrates				
Acid elution (10%$^w/_w$ H_2SO_4) mass flowrate	4.72	t/h		Acid flowrate / Cycle duration
Displacement water (Wash Number 1) mass flowrate	9.00	t/h		Water flowrate / Cycle duration
Displacement water (Wash Number 2) mass flowrate	9.00	t/h		Water flowrate / Cycle duration
V_2O_5 mass flowrate	0.36	t/h		100% of V assumed recovery
Total	23.08	t/h		
Discharge to V Precipitation Tank - Hourly				
H_2SO_4 mass flowrate (not including elution consumption)	0.47	t/h		
Displacement water (Wash Number 2) mass flowrate	13.24	t/h		
V_2O_5 mass flowrate	0.36	t/h		
Total	14.08	t/h		
H_2SO_4 molar flowrate	4.81	kg-moles/hr		
Acid consumption during Vanadium elution process				
Resin capacity (C_r)	1.00	eq.Me/L		
Equivalent grams of H_2SO_4/L	49.05	eq.g		Calculation: 1 x Equivalent gram = (Mr / Valence #)
FACTOR FOR ACID CONSUMPTION	0.02	UNITS		
Mass of H_2SO_4 consumed in eluting V_2O_5	0.0096	t/h		Calculated
U Elution				
Volumetric Elution Flowrates				
Step 1. Displacement water (Wash Number 1) mass flowrate	36.00	m^3/cycle		2 x Bed Volumes (3 x cycles per day)
Step 2. Acid elution (10%$^w/_w$ H_2SO_4) mass flowrate	18.00	m^3/cycle		1 x Bed Volume (3 x cycles per day)
Step 3. Displacement water (Wash Number 2) mass flowrate	36.00	m^3/cycle		2 x Bed Volumes (3 x cycles per day)
Mass Flowrates				
Step 1. Displacement water (Wash Number 1) mass flowrate	36.00	t/cycle		2 x Bed Volumes - (recycle to CCD Circuit)
Step 2. Acid elution (10%$^w/_w$ H_2SO_4) mass flowrate	18.86	t/cycle		1 x Bed Volume
Step 3. Displacement water (Wash Number 2) mass flowrate	36.00	t/cycle		2 x Bed Volumes
Discharge to U Precipitation Tank				
Acid elution (10%$^w/_w$ H_2SO_4) mass flowrate	18.86	t/cycle		Calculated
Displacement water (Wash Number 2) mass flowrate	36.00	t/cycle		
U mass flowrate	0.96	t/cycle		Hours per cycle x Hourly flowrate of metal
Total	55.82	t/cycle		
Average Hourly Mass Flowrates				
Acid elution (10%$^w/_w$ H_2SO_4) mass flowrate	2.36	t/h		Acid flowrate / Cycle duration
Displacement water (Wash Number 1) mass flowrate	4.50	t/h		Water flowrate / Cycle duration
Displacement water (Wash Number 2) mass flowrate	4.50	t/h		Water flowrate / Cycle duration
U mass flowrate	0.12	t/h		100% of U assumed recovery
Total	11.48	t/h		
Discharge to U Precipitation Tank - Hourly				
H_2SO_4 mass flowrate (not including elution consumption)	0.24	t/h		
Water mass flowrate	6.62	t/h		
U mass flowrate	0.12	t/h		
Total	6.98	t/h		
H_2SO_4 molar flowrate	2.40	kg-moles/hr		
Acid consumption during Uranium elution process				
Resin capacity (C_r)	1.00	eq.Me/L		
Equivalent grams of H_2SO_4/L	49.05	eq.g		Calculation: 1 x Equivalent gram = (Mr / Valence #)
FACTOR FOR ACID CONSUMPTION	0.02	UNITS		
Mass of H_2SO_4 consumed in eluting U	0.0048	t/h		Calculated
URANIUM PRECIPITATION				
Total Elution Liquor (U + V Elution)				
Streams entering U Precipitation Tank				
10%$^w/_w$ H_2SO_4 mass flowrate	7.073	t/h		Sum of elution liquors from U and V IX Columns
Displacement water (Wash Number 1) mass flowrate	13.500	t/h		Sum of elution liquors from U and V IX Columns
U mass flowrate	0.120	t/h		
V_2O_5 mass flowrate	0.363	t/h		
	20.692	t/h		
Actual Composition (taking into account elution acid consumption)				
100%$^w/_w$ H_2SO_4 mass flowrate	0.693	t/h		Calculated
Displacement water (Wash Number 1) mass flowrate	19.866	t/h		Calculated
U mass flowrate	0.120	t/h		
V_2O_5 mass flowrate	0.363	t/h		
	20.678	t/h		Sum
Peroxide (30%$^w/_w$ solution) mass flowrate	0.057	t/h		
Peroxide (100%$^w/_w$) mass flowrate	0.017	t/h		
Water contained in peroxide solution	0.040	t/h		
UO_2 mass flowrate	0.136	t/h		Calculated
Product: $UO_4 . 2H_2O$ mass flowrate	0.170	t/h		From reactions sheet
Water consumption	0.018	t/h		From reactions sheet
Peroxide consumption	0.017	t/h		From reactions sheet
Oxygen generation	0.016	t/h		From reactions sheet
Neutralization Calculation				
$n(H_2SO_4)$ =	7.064	kg-moles		Calculated
Acid concentration (Molar)	0.36	M		Calculated
Therefore, concentration of H^+ = 0.36M = 0.36 mol/L = 0.36 g/L, (as Mr(H) = 1.0)				
$pH = -log_{10}[H^+] = -log_{10}(0.36) = 0.44$	0.44	pH		pH of elution liquor.
pH (elution liquor) = 0.44				
Now calculate $[H^+]$ at pH = 4 where precipitation occurs.				
$pH = -log_{10}[H^+]$				Log Rule: $log_B(B^x) = x.log_B(B) = x$
$pH = 4 = -log_{10}[H^+]$, solving for $[H^+]$: $[H^+] = 10^{-4}$ M				
$[H^+] = 10^{-4}$ M = 10^{-4} mol/L = 0.0001 g/L (as Mr(H) = 1.0)				
100%$^w/_w$ H_2SO_4 concentration at pH = 4	0.0001	g/L		

INPUTS & CALCULATIONS SHEET

Description	Value	Units	Area	Notes
To raise pH from pH = 0.44 to pH = 4.0, need to neutralize H^+:				
0.36g/L - 0.0001g/L = 0.3599g/L = 0.3599 mol/L	0.3599	mol/L		Calculated
NH_4OH mass flowrate to achieve pH rise	0.495	t/h		From reactions sheet
NH_4OH (60%$^w/_w$) mass flowrate to achieve pH rise	0.835	t/h		Calculated
100%$^w/_w$ H_2SO_4 mass flowrate (0.0001g/L)	0.000002	t/h		(2 grams per hour acid)
Discharge to S/L Separation (prior to V Precipitation Tank)				
100%$^w/_w$ H_2SO_4 mass flowrate	0.000002	t/h		From above
H_2O mass flowrate	20.482	t/h		Calculated (input + generation - consumption = output)
UO_4 . $2H_2O$ mass flowrate	0.170	t/h		From reactions sheet
V_2O_5 mass flowrate	0.3631	t/h		Unchanged from input
$(NH_4)_2SO_4$ mass flowrate	0.934	t/h		From reactions sheet
Total	21.95	t/h		
Total liquid mass flowrate reporting to S/L Separation	21.78	t/h		
S/L Separation (Thickener & Centrifuge)				
Assume S/L Separation product contains 10%$^w/_w$ liquid	10.00%			Specified
Solids Discharge (Product)				
UO_4 . $2H_2O$ solids mass flowrate	0.170	t/h		
Total UO_4 product mass flowrate including moisture	0.189	t/h		Assume 10% moisture content
Liquids content contained in solids product	0.0189	t/h		
Fraction of liquid reporting to solids	0.0009	-		
Composition of Liquid contained in solids product				
H_2SO_4 (100%$^w/_w$) mass flowrate	0.000000002	t/h		
H_2O mass flowrate	0.018	t/h		
V_2O_5 mass flowrate	0.00031	t/h		
$(NH_4)_2SO_4$ mass flowrate	0.00081	t/h		
Total	0.0189			
Liquids Discharge (to Vanadium precipitation)				
H_2O mass flowrate	20.464	t/h		
100%$^w/_w$ H_2SO_4 mass flowrate	0.000002	t/h		
V_2O_5 mass flowrate	0.3628	t/h		
$(NH_4)_2SO_4$ mass flowrate	0.933	t/h		
Total	21.76	t/h		
VANADIUM PRECIPITATION				
NH_4OH (pure) mass flowrate to achieve pH rise to pH 7	0.0000014	t/h		From reactions sheet
NH_4OH (pure) mass flowrate to achieve ammonium vanadate precipitation	0.140	t/h		
NH_4OH (60%$^w/_w$) mass flowrate to achieve pH rise	0.233	t/h		Calculated
Product: NH_4VO_3 solids mass flowrate	0.4667	t/h		From reactions sheet
H_2O generated in neutralization and precipitation reactions mass flowrate	0.036	t/h		Calculated
Discharge to S/L Separation				
H_2O mass flowrate	20.593	t/h		
$(NH_4)_2SO_4$ mass flowrate	0.932789	t/h		
NH_4VO_3 mass flowrate	0.467	t/h		
Total	21.99	t/h		Calculated
S/L Separation (Thickener & Centrifuge)				
Assume S/L Separation product contains 10%$^w/_w$ liquid	10.00%			Specified
Solids Discharge (Product)				
Solids (dry) product: NH_4VO_3 mass flowrate	0.467	t/h		Calculated
Solids product: NH_4VO_3 mass flowrate (including moisture)	0.519	t/h		Calculated
Liquid component in product mass flowrate	0.052	t/h		Calculated
Fraction of liquid reporting to solids	0.00241	-		Factor (calculated)
Composition of Liquid contained in solids product				
H_2O mass flowrate	0.050	t/h		
$(NH_4)_2SO_4$ mass flowrate	0.002247	t/h		
NH_4VO_3 mass flowrate	0.001	t/h		
Total	0.05	t/h		Sum
Liquids Discharge (to WWT)				
H_2O mass flowrate	20.543	t/h		
$(NH_4)_2SO_4$ mass flowrate	0.931	t/h		
NH_4VO_3 (solids) mass flowrate	0.000	t/h		
Total	21.47	t/h		Sum
SOLVENT EXTRACTION - PHOSPHORIC ACID				
IX Discharge Composition				
PLS mass flowrate	256.62	t/h		Mo, U & V removed
PLS Volumetric flowrate	244.40	m^3/h		Mo, U & V removed
PLS Composition prior to IX				
PLS flowrate	257.34	t/h		
Ferric Leach feed liquids flowrate (H_2O)	226.57	t/h		
Water - H_2O	13.58	t/h		
Sulphuric Acid - H_2SO_4	1.00	t/h		
Phosphoric Acid - H_3PO_4	12.48	t/h		
Magnesium Sulphate - $MgSO_4$	1.08	t/h		
Uranyl Sulphate - $UO_2.SO_4$	0.18	t/h		
Manganese Sulphate - $MnSO_4$	0.08	t/h		
Molybdenum Sulphate - $Mo(SO_4)_2$	0.13	t/h		
Vanadyl Sulphate - $VOSO_4$	0.65	t/h		
Nickel Sulphate - $NiSO_4$	0.48	t/h		
Zinc Sulphate - $ZnSO_4$	0.99	t/h		
Yttrium Sulphate - $Y_2(SO_4)_3$	0.11	t/h		
Neodymium Sulphate - $Nd_2(SO_4)_3$	0.01	t/h		
	257.34	t/h		
PLS Composition after IX				
Ferric Leach feed liquids flowrate (H_2O)	226.57	t/h		
Water - H_2O	13.83	t/h		Adjust water by +0.25t/h.
Sulphuric Acid - H_2SO_4	1.00	t/h		
Phosphoric Acid - H_3PO_4	12.48	t/h		
Magnesium Sulphate - $MgSO_4$	1.08	t/h		
Manganese Sulphate - $MnSO_4$	0.08	t/h		

INPUTS & CALCULATIONS SHEET

Description	Value	Units	Area	Notes	
Nickel Sulphate - $NiSO_4$	0.48	t/h			
Zinc Sulphate - $ZnSO_4$	0.99	t/h			
Yttrium Sulphate - $Y_2(SO_4)_3$	0.11	t/h			
Neodymium Sulphate - $Nd_2(SO_4)_3$	0.01	t/h			
Total	256.62	t/h			
SX Feed Composition					
SX Feed (PLS) mass flowrate IN	256.62	t/h			
SX Feed (PLS) volumetric flowrate IN	244.40	m^3/h		SG = 1.05	
SX Feed (PLS) SG	1.05				
Organic Solvent volumetric flowrate IN	240.00	m^3/h			
Organic Solvent mass flowrate IN	240.00	t/h		To be confirmed by R.Raiter	
Organic Solvent SG	1.00				
Fraction of H_3PO_4 transferred to Organic Phase	100.00%	%			
Stripping Agent - H_2O mass flowrate IN	52.00	t/h		Specified	
Volumetric flowrate of H_3PO_4	6.64	m^3/h			
Mass flowrate of Stripping agent transferred to PLS	12.48	t/h			
Volumetric flowrate of Stripping agent transferred to PLS	12.48	m^3/h		Identical to mass of H_3PO_4 (differing volume due to density)	
SX Product Composition					
PLS to RE Precipitation Tank					
SX Product (PLS) mass flowrate OUT (after stripping)	256.62	t/h			
SX Product (PLS) volumetric flowrate OUT (after stripping)	250.24	m^3/h			
Loaded Organic Solvent volumetric flowrate OUT	186.64	m^3/h			
Loaded Organic Solvent mass flowrate OUT	192.48	t/h			
Volumetric flowrate of H_3PO_4	6.64	m^3/h			
Mass flowrate of H_3PO_5	12.48	t/h			
Barren Organic Solvent volumetric flowrate OUT	180.00	m^3/h			
Barren Organic Solvent mass flowrate OUT	180.00	t/h			
Loaded stripping agent to H_3PO_4 Concentration					
Loaded stripping agent - H_2O and H_3PO_4 mass flowrate OUT	52.00	t/h		Calculated	
Composition					
Water - H_2O	39.52	t/h			
Phosphoric Acid - H_3PO_4	12.48	t/h			
Total	52.00			Calculated	
Phosphoric Acid - H_3PO_4 Concentration	31.59%	$\%^w/_w$		Calculated	
PLS Composition after SX					
Ferric Leach feed liquids flowrate (H_2O, calcium acetate etc)	226.57	t/h			
Water - H_2O	26.31	t/h			
Sulphuric Acid - H_2SO_4	1.00	t/h			
Phosphoric Acid - H_3PO_4	0	t/h			
Magnesium Sulphate - $MgSO_4$	1.08	t/h			
Manganese Sulphate - $MnSO_4$	0.08	t/h			
Nickel Sulphate - $NiSO_4$	0.48	t/h			
Zinc Sulphate - $ZnSO_4$	0.99	t/h			
Yttrium Sulphate - $Y_2(SO_4)_3$	0.11	t/h			
Neodymium Sulphate - $Nd_2(SO_4)_3$	0.01	t/h			
Total	256.62	t/h			
PHOSPHORIC ACID CONCENTRATION UPGRADE					
Feed Composition					
Water - H_2O	39.52	t/h			
Phosphoric Acid - H_3PO_4	12.48	t/h			
Total	52.00				
Product Composition					
Desired product concentration	83.70%	$\%^w/_w$			
Water - H_2O	2.43	t/h		Calculated by difference using desired product concentration	
Phosphoric Acid - H_3PO_4	12.48	t/h			
Total	14.91	t/h			
Mass of H_2O removed	37.09	t/h		Calculated	Density of diesel fuel: 0.832 kg/dm^3
Energy required to achieve water removal (from 25^{0}C starting point)	26.51	MW		$Q = m \times [(C_p \times \Delta T) + \Delta H_{VAP}]$ C_p = 4.18 kJ/kg.K	Volumetric flowrate of diesel fuel required: 0.611 kg/s
Consider Lower Calorific Value of Diesel Burning	43,400	kJ/kg		ΔH_{VAP} = 2260 kJ/kg	
RARE EARTH PRECIPITATION					
Feed to RE Precipitation Tank					
Composition					
Ferric Leach feed liquids flowrate (H_2O)	226.57	t/h			
Water - H_2O	26.31	t/h			
Sulphuric Acid - H_2SO_4	1.00	t/h			
Phosphoric Acid - H_3PO_4	0	t/h			
Magnesium Sulphate - $MgSO_4$	1.08	t/h			
Manganese Sulphate - $MnSO_4$	0.08	t/h			
Nickel Sulphate - $NiSO_4$	0.48	t/h			
Zinc Sulphate - $ZnSO_4$	0.99	t/h			
Yttrium Sulphate - $Y_2(SO_4)_3$	0.11	t/h			
Neodymium Sulphate - $Nd_2(SO_4)_3$	0.01	t/h			
Total	256.62	t/h			
Ammonium hydroxide consumption					
Ammonium hydroxide (100% NH_4OH) for neutralization mass flowrate	0.715	t/h			
Ammonium hydroxide (60% NH_4OH) for neutralization mass flowrate	1.191	t/h			
Ammonium hydroxide (100% NH_4OH) for precipitation mass flowrate	0.0045	t/h			
Ammonium hydroxide (60% NH_4OH) for precipitation mass flowrate	0.0076	t/h			
Ammonium hydroxide (100% NH_4OH) total mass flowrate	0.719	t/h			
Ammonium hydroxide (60% NH_4OH) total mass flowrate	1.199	t/h			
RE Precipitation Tank discharge					

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INPUTS & CALCULATIONS SHEET

Description	Value	Units	Area	Notes
Water - H_2O generated by neutralization reaction	0.37	t/h		
Ammonium Sulphate - $(NH_4)_2SO_4$ generated by neutralization reaction	1.35	t/h		
Ammonium Sulphate - $(NH_4)_2SO_4$ generated by precipitation reaction	0.01	t/h		
Composition				
Sulphuric Acid Leach feed liquids flowrate (H_2O)	226.57	t/h		
Water - H_2O (inc generated amount + reagent dilution amount)	27.16	t/h		
Phosphoric Acid - H_3PO_4	0	t/h		
Magnesium Sulphate - $MgSO_4$	1.08	t/h		
Manganese Sulphate - $MnSO_4$	0.08	t/h		
Nickel Sulphate - $NiSO_4$	0.48	t/h		
Zinc Sulphate - $ZnSO_4$	0.99	t/h		
Yttrium Hydroxide - $Y(OH)_3$	0.063	t/h		Solids
Neodymium Hydroxide - $Nd(OH)_3$	0.008	t/h		Solids
Ammonium Sulphate - $(NH_4)_2SO_4$ (generated)	1.36	t/h		
Total	257.78	t/h		
S/L Separation				
Assume S/L Separation product contains 10%$^w/_w$ liquid	10.00%			Specified
Solids Discharge (Product)				
Solids (dry) product: $Y(OH)_3$ & $Nd(OH)_3$ combined mass flowrate	0.072	t/h		Calculated
Solids product: NH_4VO_3 mass flowrate (including moisture)	0.080	t/h		Calculated
Liquid component in product mass flowrate	0.008	t/h		Calculated
Fraction of liquid reporting to solids	0.00003	-		Factor (calculated)
Composition of Liquid contained in solids product				
Sulphuric Acid Leach feed liquids flowrate (H_2O) mass flowrate	0.007020	t/h		Calculated
H_2O mass flowrate	0.000842	t/h		Calculated
Magnesium Sulphate - $MgSO_4$	0.000033	t/h		Calculated
Manganese Sulphate - $MnSO_4$	0.000002	t/h		Calculated
Nickel Sulphate - $NiSO_4$	0.000015	t/h		Calculated
Zinc Sulphate - $ZnSO_4$	0.000031	t/h		Calculated
Ammonium Sulphate - $(NH_4)_2SO_4$ (generated)	0.000042	t/h		Calculated
Total	0.008	t/h		Sum
Liquids Discharge (to Mn, Ni & Zn SX)				
Sulphuric Acid Leach feed liquids flowrate (H_2O) mass flowrate	226.564	t/h		Calculated
H_2O mass flowrate	27.160	t/h		Calculated
Magnesium Sulphate - $MgSO_4$	1.075	t/h		Calculated
Manganese Sulphate - $MnSO_4$	0.076	t/h		Calculated
Nickel Sulphate - $NiSO_4$	0.480	t/h		Calculated
Zinc Sulphate - $ZnSO_4$	0.990	t/h		Calculated
Ammonium Sulphate - $(NH_4)_2SO_4$ (generated)	1.356	t/h		Calculated
Total	257.701	t/h		Sum
SOLVENT EXTRACTION - MANGANESE, NICKEL & ZINC				
SX Feed Composition				
SX Feed (PLS) mass flowrate IN	257.70	t/h		
SX Feed (PLS) volumetric flowrate IN	245.43	m^3/h	1.05	SG = 1.05
Organic Solvent volumetric flowrate IN	240.00	m^3/h		
Organic Solvent mass flowrate IN	240.00	t/h		R.Raiter to confirm.
Fraction of Mn, Ni & Zn transferred to Organic Phase	100.00%	%		
Stripping Agent - 10%$^w/_w$ H_2SO_4 volumetric flowrate IN	20.00	m^3/h		R.Raiter specified.
Stripping Agent - 10%$^w/_w$ H_2SO_4 mass flowrate IN	20.96	t/h	1.048	SG = 1.048 (refer to Compounds Data Sheet)
Volumetric flowrate of $MnSO_4$	0.02	m^3/h		
Volumetric flowrate of $NiSO_4$	0.12	m^3/h		
Volumetric flowrate of $ZnSO_4$	0.28	m^3/h		
Total	0.42	m^3/h		
Loaded Organic Solvent volumetric flowrate OUT	240.42	m^3/h		
Loaded Organic Solvent mass flowrate OUT	241.55	t/h		
Volumetric flowrate of Stripping agent transferred to PLS	0.42	m^3/h		To facilitate solvent extraction
Mass flowrate of Stripping agent transferred to PLS	0.44	t/h		
SX Product Composition				
PLS to RE Precipitation Tank				
SX Product (PLS) mass flowrate OUT	256.60	t/h		
SX Product (PLS) volumetric flowrate OUT	244.38	m^3/h		
Organic Solvent mass flowrate OUT	240.00	m^3/h		
Loaded stripping agent to Precipitation Tank				
Loaded stripping agent - H_2SO_4 and Mn, Ni & Zn compounds mass flowrate OUT	22.06	t/h		
Composition				
Water - H_2O	18.47	t/h		
Sulphuric Acid - H_2SO_4	2.05	t/h		
$MnSO_4$ mass flowrate	0.08	t/h		
$NiSO_4$ mass flowrate	0.48	t/h		
$ZnSO_4$ mass flowrate	0.99	t/h		
Total	22.06			
PLS Composition after SX (to WWT)				
Sulphuric Acid Leach feed liquids flowrate (H_2O)	226.56	t/h		
H_2O mass flowrate	27.56	t/h		
Sulphuric Acid - H_2SO_4 mass flowrate	0.04	t/h		
Phosphoric Acid - H_3PO_4 mass flowrate	0	t/h		
Magnesium Sulphate - $MgSO_4$ mass flowrate	1.08	t/h		
Manganese Sulphate - $MnSO_4$ mass flowrate	0	t/h		
Nickel Sulphate - $NiSO_4$ mass flowrate	0	t/h		
Zinc Sulphate - $ZnSO_4$ mass flowrate	0	t/h		
Yttrium Sulphate - $Y_2(SO_4)_3$ mass flowrate	0	t/h		
Neodymium Sulphate - $Nd_2(SO_4)_3$ mass flowrate	0	t/h		
Total	255.24	t/h		
MANGANESE, NICKEL & ZINC PRECIPITATION				
Feed to Mn, Ni & Zn Precipitation Tank				
Composition				
Water - H_2O	18.47	t/h		

INPUTS & CALCULATIONS SHEET

Description	Value	Units	Area	Notes
Sulphuric Acid - H_2SO_4	2.05	t/h		
$MnSO_4$ mass flowrate	0.08	t/h		
$NiSO_4$ mass flowrate	0.48	t/h		
$ZnSO_4$ mass flowrate	0.99	t/h		
Total	22.06			
Neutralization Calculation				
$n(H_2SO_4)$ =	20.917	kg-moles		Calculated
Acid concentration (Molar) (Molar = moles/L)	1.13	M		Calculated
Therefore, concentration of H^+ = 1.13M = 1.13 mol/L = 1.13 g/L, (as Mr(H) = 1.0)				
pH = $-\log_{10}[H^+]$ = $-\log_{10}(1.13)$ = -0.053	0.00	pH		pH of elution liquor.
pH (elution liquor) = -0.053 (assume pH = 0)				
Now calculate $[H^+]$ at pH = 6 where precipitation occurs.				
pH = $-\log_{10}[H^+]$				Log Rule: $\log_B(B^x) = x.\log_B(B) = x$
pH = 6 = $-\log_{10}[H^+]$, solving for $[H^+]$: $[H^+] = 10^{-6}$ M				
$[H^+] = 10^{-6}$ M = 10^{-6} mol/L = 0.000001 g/L (as Mr(H) = 1.0)				
100%$^w/_w$ H_2SO_4 concentration at pH = 6	0.000001	g/L		Calculated
To raise pH from pH = 0 to pH = 6.0, need to neutralize H^+:				
1.13g/L - 0.000001g/L =1.129999g/L = 1.129999 mol/L to be neutralized	1.129999	mol/L		Calculated
Moles of H+ to be neutralized (1.129999 x 18.48) =	20.882	kg-moles		Calculated
NH_4OH molar flowrate to achieve pH rise	2.260	mol/L		
NH_4OH molar flowrate to achieve pH rise	41.764	kg-moles		
NH_4OH mass flowrate to achieve pH rise	1.464	t/h		From reactions sheet
NH_4OH (60%$^w/_w$) mass flowrate to achieve pH rise	2.440	t/h		Calculated
Residual 100%$^w/_w$ H_2SO_4 mass flowrate (0.000001g/L)	0.00339	t/h		Calculated by difference (input - consumption)
Intermediate Composition (after neutralization but prior to precipitation)				
H_2O mass flowrate	20.194	t/h		Calculated (input + generation + reagents cpt = output)
100%$^w/_w$ H_2SO_4 mass flowrate	0.003386	t/h		Calculated (input - consumption = output)
Ammonium Sulphate - $(NH_4)_2SO_4$ mass flowrate	2.760	t/h		Generated
$MnSO_4$ mass flowrate	0.08	t/h		Unchanged from input
$NiSO_4$ mass flowrate	0.48	t/h		Unchanged from input
$ZnSO_4$ mass flowrate	0.99	t/h		Unchanged from input
Total	24.50	t/h		
RE Precipitation Tank discharge				
Ammonium carbonate consumption				
Ammonium carbonate (100% $(NH_4)_2CO_3$) for precipitation mass flowrate	2.3516	t/h		
Ammonium carbonate (60% $(NH_4)_2CO_3$) for precipitation mass flowrate	3.9194	t/h		
Composition			24.244	Sum of liquids
H_2O mass flowrate	20.194	t/h		From above
100%$^w/_w$ H_2SO_4 mass flowrate	0.003	t/h		Residual
Ammonium Sulphate - $(NH_4)_2SO_4$ mass flowrate	4.047	t/h		Sum (from reactions sheet)
$MnCO_3$ mass flowrate (solids)	0.058	t/h		Solids
$NiCO_3$ mass flowrate (solids)	0.368	t/h		Solids
$ZnCO_3$ mass flowrate (solids)	0.769	t/h		Solids
Total	25.44	t/h		
Solids total mass flowrate	1.19	t/h		
Liquids total mass flowrate	24.24	t/h		
S/L Separation				
Assume S/L Separation product contains 10%$^w/_w$ liquid	10.00%			Specified
Solids Discharge (Product)				
Solids (dry) product: carbonates combined mass flowrate	1.195	t/h		Calculated
Solids product: carbonates mass flowrate (including moisture)	1.327	t/h		Calculated
Liquid component in product mass flowrate	0.133	t/h		Calculated
Fraction of liquid reporting to solids product	0.00548	-		Factor (calculated)
Composition of Liquid contained in solids product				
H_2O mass flowrate	0.111	t/h		
100%$^w/_w$ H_2SO_4 mass flowrate	0.00002	t/h		
Ammonium Sulphate - $(NH_4)_2SO_4$ mass flowrate	0.022	t/h		
Total	0.133	t/h		Sum
Liquids Discharge (to WWT)				
H_2O mass flowrate	20.083	t/h		Balance
100%$^w/_w$ H_2SO_4 mass flowrate	0.003	t/h		Balance
Ammonium Sulphate - $(NH_4)_2SO_4$ mass flowrate	4.025	t/h		Balance
Total	24.111	t/h		Sum
			24.244	Check: Sum of liquids output in S/L Separation
WASTE WATER TREATMENT				
Sum of all WWT Liquids Discharges				
Sum of contaminants in WWT Liquids Discharges				

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REACTIONS & CONVERSIONS SHEET

	Reaction	Reactant 1	Reactant 2	Reactant 3		Product 1	Product 2	Product 3	Product 4
	SULPHURIC ACID LEACH								
Calcite:									
	Reaction 1:		1 $CaCO_3$	+ 1 H_2SO_4	⟶	1 $CaSO_4$	+ 1 H_2O	+ 1 CO_2	
	kmoles:		607.01	607.01		607.01	607.01	607.01	
1.00	(100% conversion)					gypsum			
Fluorapatite:									
	Reaction 2:		1 $Ca_5(PO_4)_3F$	+ 5 H_2SO_4	⟶	5 $CaSO_4$	+ 3 H_3PO_4	+ 1 HF	
	kmoles:		42.46	212.28		212.28	127.37	42.46	
1.00	(100% conversion)					gypsum			
Dolomite:									
	Reaction 3:		1 $CaMg(CO_3)_2$	+ 2 H_2SO_4	⟶	1 $CaSO_4$	+ 1 $MgSO_4$	+ 2 CO_2	+ 2 H_2O
	kmoles:		8.93	17.86		8.93	8.93	17.86	17.86
1.00	(100% conversion)					gypsum			
Uranium 1:									
	Reaction 4:			1 U_3O_8	⟶	3 UO_2	+ 1 O_2		
	kmoles:			0.17		0.50	0.17		
0.98	(98.0% conversion)			140.96 kg					
Uranium 2:									
	Reaction 5:		1 UO_2	+ 1 $Fe_2(SO_4)_3$	⟶	1 $UO_2.SO_4$	+ 2 $FeSO_4$		
	kmoles:		0.50	0.50		0.50	1.00		
1.00	(100% conversion)								
Pyrolusite:									
	Reaction 6:	1 MnO_2	+ 2 H_2SO_4	+ 2 $FeSO_4$	⟶	1 $MnSO_4$	+ 2 $Fe_2(SO_4)_3$	+ 2 H_2O	+ 2 O_2
	kmoles:	0.50	1.00	1.00		0.50	1.00	1.00	1.00
1.00	(100% conversion)								
Molybdenum:									
	Reaction 7:		1 MoS_2	+ 3 H_2SO_4	⟶	1 $Mo(SO_4)_3$	+ 2 H_2S	+ 1 H_2	
	kmoles:		0.34	1.03		0.34	0.69	0.34	
0.51	(51.0% conversion)								
Vanadium:									
	Reaction 8:		1 V_2O_5	+ 2 H_2SO_4	⟶	2 $VOSO_4$	+ 2 H_2O	+ 0.5 O_2	
	kmoles:		2.00	3.99		3.99	3.99	1.00	
0.73	(73.0% conversion)								
Nickel:									
	Reaction 10:		1 NiS	+ 1 H_2SO_4	⟶	1 $NiSO_4$	+ 1 H_2S		
	kmoles:		3.10	3.10		3.10	3.10		
0.60	(60.0% conversion)								
Zinc:									
	Reaction 11:		1 ZnS	+ 1 H_2SO_4	⟶	1 $ZnSO_4$	+ 1 H_2S		

Compound	M_r	Mass (kg)	kmoles
Reaction 1			
$CaCO_3$	100.09	60,756.0	607.01
H_2SO_4	98.09	59,542.0	607.01
$CaSO_4$	136.15	82,644.9	607.01
H_2O	18.02	10,938.4	607.01
CO_2	44.01	26,714.7	607.01
Reaction 2			
$Ca_5(PO_4)_3F$	504.3	21,411.0	42.46
H_2SO_4	98.09	20,822.6	212.28
$CaSO_4$	136.15	28,901.9	212.28
H_3PO_4	98.00	12,482.1	127.37
HF	20.01	849.5	42.46
Reaction 3			
$CaMg(CO_3)_2$	184.41	1,647.0	8.93
H_2SO_4	98.09	1,752.1	17.86
$CaSO_4$	136.15	1,216.0	8.93
$MgSO_4$	120.38	1,075.1	8.93
CO_2	44.01	786.1	17.86
H_2O	18.02	321.9	17.86
Reaction 4			
U_3O_8 (Feed)	842.09	143.8	0.17
U_3O_8 (Reacted)	842.09	141.0	0.17
UO_2	270.03	135.6	0.50
O_2	32.00	5.4	0.17
Reaction 5			
UO_2	270.03	135.6	0.50
$Fe_2(SO_4)_3$	399.91	200.8	0.50
$UO_2.SO_4$	366.09	183.8	0.50
$FeSO_4$	151.91	152.6	1.00
Reaction 6			
MnO_2	86.94	43.7	0.50
H_2SO_4	98.09	98.5	1.00
$FeSO_4$	151.91	152.6	1.00
$Fe_2(SO_4)_3$	399.91	401.7	1.00
$MnSO_4$	151.01	75.8	0.50
H_2O	18.02	18.1	1.00
O_2	32.00	32.1	1.00
Reaction 7			
MoS_2 (Feed)	160.10	108.1	0.68
MoS_2 (Reacted)	160.10	55.1	0.34
H_2SO_4	98.09	101.3	1.03
$Mo(SO_4)_3$	384.17	132.3	0.34
H_2S	34.09	23.5	0.69
H_2	2.02	0.7	0.34
Reaction 8			
V_2O_5 (Feed)	181.88	497.4	2.73
V_2O_5 (Reacted)	181.88	363.1	2.00
H_2SO_4	98.09	391.6	3.99
$VOSO_4$	163.01	650.9	3.99
H_2O	18.02	71.9	3.99
O_2	32.00	31.9	1.00
Reaction 10			
NiS (Feed)	90.77	468.80	5.16
NiS (Reacted)	90.77	281.3	3.10
H_2SO_4	98.09	304.0	3.10
$NiSO_4$	154.77	479.6	3.10
H_2S	34.09	105.6	3.10
Reaction 11			
ZnS (Feed)	97.45	610.0	6.26

Total Gypsum Production

Mass	kmoles
112,762.8	828.23

Total Water Production

Mass	kmoles
11,363.7	630.62

Total Oxygen Production

Mass	kmoles
69.4	2.17

Total H_2S Production

Mass	kmoles
338.2	9.92

Total CO_2 Production

Mass	kmoles
27,500.8	624.88

REACTIONS & CONVERSIONS SHEET

	kmoles:	6.13		6.13		6.13		6.13	
0.98	(98.0% conversion)								
Yttrium:									
	Reaction 12:	1 Y_2O_3	+	3 H_2SO_4	⟶	1 $Y_2(SO_4)_3$	+	3 H_2O	
	kmoles:	0.23		0.68		0.23		0.68	
0.91	(91.0% conversion)								
Neodymium:									
	Reaction 13:	1 Nd_2O_3	+	3 H_2SO_4	⟶	1 $Nd_2(SO_4)_3$	+	3 H_2O	
	kmoles:	0.02		0.06		0.02		0.06	
0.64	(64.0% conversion)								

Compound	M_r	Mass (kg)	kmoles
ZnS (Reacted)	97.45	597.8	6.13
H_2SO_4	98.09	601.7	6.13
$ZnSO_4$	161.45	990.4	6.13
H_2S	34.09	209.1	6.13
Reaction 12			
Y_2O_3 (Feed)	225.82	56.2	0.25
Y_2O_3 (Reacted)	225.82	51.2	0.23
H_2SO_4	98.09	66.7	0.68
$Y_2(SO_4)_3$	466.03	105.6	0.23
H_2O	18.02	12.3	0.68
Reaction 13			
Nd_2O_3 (Feed)	336.48	11.3	0.034
Nd_2O_3 (Reacted)	336.48	7.3	0.022
H_2SO_4	98.09	6.4	0.06
$Nd_2(SO_4)_3$	576.69	12.4	0.02
H_2O	18.02	1.2	0.06

MOLYBDENUM PRECIPITATION

	Reaction	Moles	Reactant 1	Moles	Reactant 2	Moles	Reactant 3	Moles	Product 1	Moles	Product 2	Moles	Product 3	Moles	Product 4
Neutralization:	Reaction 1:			1	H_2SO_4 +	2	NH_4OH ⟶	1	$(NH_4)_2SO_4$ +	2	H_2O				
	kmoles:			0.00		0.00		0.00		0.00					
1	(100.0% conversion)														
Precipitation:	Reaction 2:			1	$(NH_4)_2MoO_4$ +	1	$Ca(OH)_2$ ⟶	1	$CaMoO_4$ +	2	NH_4OH				
	kmoles:			0.34		0.34		0.34		0.69					
1	(100.0% conversion)														

REACTION NOT RELEVANT WHEN ELUTION OCCURS WITH NH_4OH (AS NO RESIDUAL H_2SO_4 TO NEUTRALIZE)

Compound	M_r	Mass (kg)	kmoles
Reaction 1			
H_2SO_4	98.09	0.0	0.00
NH_4OH	35.06	0.0	0.00
$(NH_4)_2SO_4$	132.17	0.0	0.00
H_2O	18.02	0.0	0.00
Reaction 2			
$(NH_4)_2MoO_4$	196.06	67.5	0.34
$Ca(OH)_2$	74.10	25.5	0.34
$CaMoO_4$	200.04	68.9	0.34
NH_4OH	35.06	24.1	0.69

URANIUM PRECIPITATION

	Reaction	Moles	Reactant 1	Moles	Reactant 2	Moles	Reactant 3	Moles	Product 1	Moles	Product 2	Moles	Product 3	Moles	Product 4
Raise to pH=4.	Reaction 1:			1	H_2SO_4 +	2	NH_4OH ⟶	1	$(NH_4)_2SO_4$ +	2	H_2O				
	kmoles:			7.06		14.13		7.06		14.13					
1	(100.0% conversion)														
Precipitation:	Reaction 2:	1	UO_2^{2+} +	1	H_2O_2 +	2	H_2O ⟶	1	$UO_4 . 2H_2O$ +	2	H^+ +	1	O_2		
	kmoles:	0.50		0.50		1.00		0.50		1.00		0.50			
1	(100.0% conversion)														

Compound	M_r	Mass (kg)	kmoles
Reaction 1			
H_2SO_4	98.09	692.9	7.06
NH_4OH	35.06	495.3	14.13
$(NH_4)_2SO_4$	132.17	933.6	7.06
H_2O	18.02	254.6	14.13
Reaction 2			
UO_2^{2+}	270.03	135.6	0.50
H_2O_2	34.02	17.1	0.50
H_2O	18.02	18.1	1.00
$UO_4 .2H_2O$	338.07	169.8	0.50
H_2O	18.02	18.1	1.00
O_2	32.00	16.1	0.50

VANADIUM PRECIPITATION

	Reaction	Moles	Reactant 1	Moles	Reactant 2	Moles	Reactant 3	Moles	Product 1	Moles	Product 2	Moles	Product 3	Moles	Product 4
Raise to pH=7.	Reaction 1:			1	H_2SO_4 +	2	NH_4OH ⟶	1	$(NH_4)_2SO_4$ +	2	H_2O				
(from pH=4)	kmoles:			0.00002		0.00004		0.00002		0.00004					
1	(100.0% conversion)														
Precipitation:	Reaction 2:			1	V_2O_5 +	2	NH_4OH ⟶	2	NH_4VO_3 +	1	H_2O				
	kmoles:			1.99		3.99		3.99		1.99					
1	(100.0% conversion)														

Compound	M_r	Mass (kg)	kmoles
Reaction 1			
H_2SO_4	98.09	0.0020	0.00002
NH_4OH	35.06	0.0014	0.00004
$(NH_4)_2SO_4$	132.17	0.0027	0.00002
H_2O	18.02	0.0007	0.00004
Reaction 2			
V_2O_5	181.88	362.8	1.99
NH_4OH	35.06	139.9	3.99
NH_4VO_3	116.99	466.7	3.99
H_2O	18.02	35.9	1.99

REACTIONS & CONVERSIONS SHEET

Reaction	Moles	Reactant 1	Moles	Reactant 2	Moles	Reactant 3	Moles	Product 1	Moles	Product 2	Moles	Product 3	Moles	Product 4
RARE EARTH PRECIPITATION														

		Reactant		Reactant		Product		Product
Neutralization	Reaction 1:	1 H_2SO_4	+	2 NH_4OH	⟶	1 $(NH_4)_2SO_4$	+	2 H_2O
	kmoles:	10.19		20.39		10.19		20.39
	(100.0% conversion)							
Precipitation:	Reaction 2:	1 $Y_2(SO_4)_3$	+	6 NH_4OH	⟶	2 $Y(OH)_3$	+	3 $(NH_4)_2SO_4$
	kmoles:	0.23		1.36		0.45		0.68
1	(100.0% conversion)							
Precipitation:	Reaction 3:	1 $Nd_2(SO_4)_3$	+	6 NH_4OH	⟶	2 $Nd(OH)_3$	+	3 $(NH_4)_2SO_4$
	kmoles:	0.022		0.129		0.043		0.065
1	(100.0% conversion)							

Compound	M_r	Mass (kg)	kmoles
Reaction 1			
H_2SO_4	98.09	1,000.00	10.19
NH_4OH	35.06	714.85	20.39
$(NH_4)_2SO_4$	132.17	1,347.44	10.19
H_2O	18.02	367.42	20.39
Reaction 2			
$Y_2(SO_4)_3$	466.03	105.62	0.23
NH_4OH	35.06	47.7	1.36
$Y(OH)_3$	139.94	63.4	0.45
$(NH_4)_2SO_4$	132.17	89.9	0.68
Reaction 3			
$Nd_2(SO_4)_3$	576.69	12.4	0.022
NH_4OH	35.06	4.5	0.13
$Nd(OH)_3$	195.27	8.4	0.04
$(NH_4)_2SO_4$	132.17	8.6	0.06

Reaction	Moles	Reactant 1	Moles	Reactant 2	Moles	Reactant 3	Moles	Product 1	Moles	Product 2	Moles	Product 3	Moles	Product 4
MANGANESE, NICKEL & ZINC PRECIPITATION														

		Reactant		Reactant		Product		Product
Neutralization:	Reaction 1:	1 H_2SO_4	+	2 NH_4OH	⟶	1 $(NH_4)_2SO_4$	+	2 H_2O
	kmoles:	20.88		41.76		20.88		41.76
	(100.0% conversion)							
Precipitation:	Reaction 2:	1 $MnSO_4$	+	1 $(NH_4)_2CO_3$	⟶	1 $MnCO_3$	+	1 $(NH_4)_2SO_4$
	kmoles:	0.50		0.50		0.50		0.50
	(100.0% conversion)							
Precipitation:	Reaction 3:	1 $NiSO_4$	+	1 $(NH_4)_2CO_3$	⟶	1 $NiCO_3$	+	1 $(NH_4)_2SO_4$
	kmoles:	3.10		3.10		3.10		3.10
	(100.0% conversion)							
Precipitation:	Reaction 4:	1 $ZnSO_4$	+	1 $(NH_4)_2CO_3$	⟶	1 $ZnCO_3$	+	1 $(NH_4)_2SO_4$
	kmoles:	6.13		6.13		6.13		6.13
	(100.0% conversion)							

Compound	M_r	Mass (kg)	kmoles
Reaction 1			
H_2SO_4	98.09	2,048.32	20.882
NH_4OH	35.06	1,464.25	41.76
$(NH_4)_2SO_4$	132.17	2,759.97	20.88
H_2O	18.02	752.59	41.76
Reaction 2			
$MnSO_4$	151.01	75.83	0.50
$(NH_4)_2CO_3$	96.11	48.26	0.50
$MnCO_3$	114.95	57.72	0.50
$(NH_4)_2SO_4$	132.17	66.37	0.50
Reaction 3			
$NiSO_4$	154.77	479.59	3.10
$(NH_4)_2CO_3$	96.11	297.8	3.10
$NiCO_3$	118.71	367.9	3.10
$(NH_4)_2SO_4$	132.17	409.6	3.10
Reaction 4			
$ZnSO_4$	161.45	990.4	6.13
$(NH_4)_2CO_3$	96.11	589.6	6.13
$ZnCO_3$	125.39	769.2	6.13
$(NH_4)_2SO_4$	132.17	810.8	6.13

REAGENT CONSUMPTION SHEET

Reaction	Moles	Reactant 1	Moles	Reactant 2	Moles	Reactant 3	Moles	Product 1	Moles	Product 2	Moles	Product 3	Moles	Product 4
SULPHURIC ACID CONSUMPTION														

Application	Consumption (t/h)
Calcite leach	59.54
Fluorapatite leach	20.82
Dolomite leach	1.75
Pyrolusite reaction	0.10
Molybdenum reaction	0.10
Vanadium reaction	0.39
Nickel reaction	0.30
Zinc reaction	0.60
Yttrium reaction	0.07
Neodymium reaction	0.01
U IX elution	0.24
V IX elution	0.47
Mn, Ni, Zn SX strip	2.10
TOTAL	**86.49** t/h

**** This table summarizes the reactions whereby sulphuric acid is consumed throughout various applications in the processing plant.***

Reaction	Moles	Reactant 1	Moles	Reactant 2	Moles	Reactant 3	Moles	Product 1	Moles	Product 2	Moles	Product 3	Moles	Product 4
PYROLUSITE CONSUMPTION														

Sulphuric Acid Leach - Conversion of Ferrous to Ferric Iron

	Reactant 1		Reactant 2		Reactant 3		Product 1		Product 2		Product 3		Product 4
Reaction 1:	1 MnO_2	+	2 H_2SO_4	+	2 $FeSO_4$	⟶	1 $MnSO_4$	+	2 $Fe_2(SO_4)_3$	+	2 H_2O	+	2 O_2
kmoles:	2.84		5.69		5.69		2.84		5.69		5.69		5.69
1 (100% conversion)													

Sum: of ferrous iron produced by ferric reactions below.

Compound	M_r	Mass (kg)	kmoles
Reaction 1			
MnO_2	86.94	247.2	2.84
H_2SO_4	98.09	557.7	5.69
$FeSO_4$	151.91	863.7	5.69
$Fe_2(SO_4)_3$	399.91	2,273.8	5.69
$MnSO_4$	151.01	429.3	2.84
TOTAL		247.16	kg/h

Reaction	Moles	Reactant 1	Moles	Reactant 2	Moles	Reactant 3	Moles	Product 1	Moles	Product 2	Moles	Product 3	Moles	Product 4
AMMONIA CONSUMPTION														

Molybdenum Precipitation

		Reactant 1		Reactant 2		Product 1		Product 2
Neutralization:	Reaction 1:	1 H_2SO_4	+	2 NH_4OH	⟶	1 $(NH_4)_2SO_4$	+	2 H_2O
	kmoles:	0.00		0.00		0.00		0.00
	1 (100.0% conversion)							

Uranium Precipitation

		Reactant 1		Reactant 2		Product 1		Product 2
Raise to pH=4	Reaction 2:	1 H_2SO_4	+	2 NH_4OH	⟶	1 $(NH_4)_2SO_4$	+	2 H_2O
	kmoles:	7.06		14.13		7.06		14.13
	1 (100.0% conversion)							

Vanadium Precipitation

		Reactant 1		Reactant 2		Product 1		Product 2
Raise to pH=7	Reaction 3:	1 H_2SO_4	+	2 NH_4OH	⟶	1 $(NH_4)_2SO_4$	+	2 H_2O
(from pH=4)	kmoles:	0.00002		0.00004		0.00002		0.00004
	1 (100.0% conversion)							

Rare Earth Precipitation

		Reactant 1		Reactant 2		Product 1		Product 2
Neutralization	Reaction 4:	1 H_2SO_4	+	2 NH_4OH	⟶	1 $(NH_4)_2SO_4$	+	2 H_2O
	kmoles:	10.19		20.39		10.19		20.39

Compound	M_r	Mass (kg)	kmoles
Reaction 1			
H_2SO_4	98.09	0.0	0.00
NH_4OH	35.06	0.0	0.00
$(NH_4)_2SO_4$	132.17	0.0	0.00
H_2O	18.02	0.0	0.00
Reaction 2			
H_2SO_4	98.09	692.9	7.06
NH_4OH	35.06	495.3	14.13
$(NH_4)_2SO_4$	132.17	933.6	7.06
H_2O	18.02	254.6	14.13
Reaction 3			
H_2SO_4	98.09	0.0020	0.00002
NH_4OH	35.06	0.0014	0.00004
$(NH_4)_2SO_4$	132.17	0.0027	0.00002
H_2O	18.02	0.0007	0.00004
Reaction 4			
H_2SO_4	98.09	1,000.00	10.19
NH_4OH	35.06	714.85	20.39
$(NH_4)_2SO_4$	132.17	1,347.44	10.19

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REAGENT CONSUMPTION SHEET

(100.0% conversion)

Precipitation: Reaction 5: 1 $Y_2(SO_4)_3$ + 6 NH_4OH ⟶ 2 $Y(OH)_3$ + 3 $(NH_4)_2SO_4$

kmoles: 0.23 | 1.36 | 0.45 | 0.68

1 (100.0% conversion)

Precipitation: Reaction 6: 1 $Nd_2(SO_4)_3$ + 6 NH_4OH ⟶ 2 $Nd(OH)_3$ + 3 $(NH_4)_2SO_4$

kmoles: 0.022 | 0.129 | 0.043 | 0.065

1 (100.0% conversion)

Manganese, Nickel & Zinc Precipitation

Neutralization: Reaction 7: 1 H_2SO_4 + 2 NH_4OH ⟶ 1 $(NH_4)_2SO_4$ + 2 H_2O

kmoles: 20.88 | 41.76 | 20.88 | 41.76

(100.0% conversion)

Molybdenum Elution

4.00% NH_4OH solution, delivered at : 2.24 t/h

Mass of NH_4OH in 4.00%$^w/_w$ solution = 0.090 t/h

Compound	M_r	Mass (kg)	kmoles
H_2O	18.02	367.42	20.39
Reaction 5			
$Y_2(SO_4)_3$	466.03	105.6	0.23
NH_4OH	35.06	47.7	1.36
$Y(OH)_3$	139.94	63.4	0.45
$(NH_4)_2SO_4$	132.17	89.9	0.68
Reaction 6			
$Nd_2(SO_4)_3$	576.69	12.4	0.022
NH_4OH	35.06	4.5	0.129
$Nd(OH)_3$	195.27	8.4	0.043
$(NH_4)_2SO_4$	132.17	8.6	0.065
Reaction 7			
H_2SO_4	98.09	2,048.32	20.882
NH_4OH	35.06	1,464.25	41.76
$(NH_4)_2SO_4$	132.17	2,759.97	20.88
H_2O	18.02	752.59	41.76
NH_4OH	35.06	89.51	2.55
TOTAL		2,816.13	kg/h

Mass of NH_4OH in 60.00%$^w/_w$ solution = 4,693.55 kg/h

Reaction	Moles	Reactant 1	Moles	Reactant 2	Moles	Reactant 3	Moles	Product 1	Moles	Product 2	Moles	Product 3	Moles	Product 4
CALCIUM HYDROXIDE CONSUMPTION														

Molybdenum Precipitation

Precipitation: Reaction 1: 1 $(NH_4)_2Mo_4$ + 1 $Ca(OH)_2$ ⟶ 1 $CaMoO_4$ + 2 NH_4Cl

kmoles: 0.34 | 0.34 | 0.34 | 0.69

1 (100.0% conversion)

Compound	M_r	Mass (kg)	kmoles
Reaction 1			
$(NH_4)_2Mo_4$	196.06	67.5	0.34
$Ca(OH)_2$	74.10	25.5	0.34
$CaMoO_4$	200.04	68.9	0.34
NH_4Cl	53.50	18.4	0.34
TOTAL		25.51	kg/h

Mass of $Ca(OH)_2$ in 20.00%$^w/_w$ solution = 127.56 kg/h

Reaction	Moles	Reactant 1	Moles	Reactant 2	Moles	Reactant 3	Moles	Product 1	Moles	Product 2	Moles	Product 3	Moles	Product 4
HYDROGEN PEROXIDE CONSUMPTION														

Uranium Precipitation

Precipitation: Reaction 1: 1 UO_2^{2+} + 1 H_2O_2 + 2 H_2O ⟶ 1 $UO_4 \cdot 2H_2O$ + 2 H^+ + 1 O_2

kmoles: 0.50 | 0.50 | 1.00 | 0.50 | 1.00 | 0.50

1 (100.0% conversion)

Compound	M_r	Mass (kg)	kmoles
Reaction 1			
UO_2^{2+}	270.03	135.6	0.50
H_2O_2	34.02	17.1	0.50
H_2O	18.02	18.1	1.00
$UO_4 \cdot 2H_2O$	338.07	169.8	0.50
H_2O	18.02	18.1	1.00
O_2	32.00	16.1	0.50
TOTAL		17.08	kg/h

Reaction	Moles	Reactant 1	Moles	Reactant 2	Moles	Reactant 3	Moles	Product 1	Moles	Product 2	Moles	Product 3	Moles	Product 4
AMMONIUM CARBONATE CONSUMPTION														

Manganese, Nickel & Zinc Precipitation

Compound	M_r	Mass (kg)	kmoles
Reaction 2			

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Precipitation: Reaction 2: 1 $MnSO_4$ + 1 $(NH_4)_2CO_3$ ⟶ 1 $MnCO_3$ + 1 $(NH_4)_2SO_4$
kmoles: 0.50 0.50 0.50 0.50
(100.0% conversion)

Precipitation: Reaction 3: 1 $NiSO_4$ + 1 $(NH_4)_2CO_3$ ⟶ 1 $NiCO_3$ + 1 $(NH_4)_2SO_4$
kmoles: 3.10 3.10 3.10 3.10
(100.0% conversion)

Precipitation: Reaction 4: 1 $ZnSO_4$ + 1 $(NH_4)_2CO_3$ ⟶ 1 $ZnCO_3$ + 1 $(NH_4)_2SO_4$
kmoles: 6.13 6.13 6.13 6.13
(100.0% conversion)

$MnSO_4$	151.01	75.83	0.50
$(NH_4)_2CO_3$	96.11	48.26	0.50
$MnCO_3$	114.95	57.72	0.50
$(NH_4)_2SO_4$	132.17	66.37	0.50
Reaction 3			
$NiSO_4$	154.77	479.59	3.10
$(NH_4)_2CO_3$	96.11	297.8	3.10
$NiCO_3$	118.71	367.9	3.10
$(NH_4)_2SO_4$	132.17	409.6	3.10
Reaction 4			
$ZnSO_4$	161.45	990.4	6.13
$(NH_4)_2CO_3$	96.11	589.6	6.13
$ZnCO_3$	125.39	769.2	6.13
$(NH_4)_2SO_4$	132.17	810.8	6.13
TOTAL		935.65	kg/h

Reaction	Moles	Reactant 1	Moles	Reactant 2	Moles	Reactant 3	Moles	Product 1	Moles	Product 2	Moles	Product 3	Moles	Product 4
FERRIC SULPHATE CONSUMPTION														

Uranium 2:
Reaction 1: 1 UO_2 + 1 $Fe_2(SO_4)_3$ ⟶ 1 $UO_2.SO_4$ + 2 $FeSO_4$
kmoles: 0.50 0.50 0.50 1.00
1 (100% conversion)

Vanadium: Reaction 2: 1 V^{3+} + 2 $Fe^{3}+$ ⟶ 2 $Fe^{2}+$ + 1 V^{5+}
kmoles: 2.00 3.99 3.99 2.00
(100% conversion)

Molybdenum: Reaction 3: 1 Mo^{4+} + 2 $Fe^{3}+$ ⟶ 2 $Fe^{2}+$ + 1 Mo^{6+}
kmoles: 0.34 0.69 0.69 0.34
(100% conversion)

At 60%$^{w}/_{w}$ solution $Fe_2(SO_4)_3$, consumption of reagent is: 3,454.87 kg/h

Compound	M_r	Mass (kg)	kmoles
Reaction 1			
UO_2	270.03	135.6	0.50
$Fe_2(SO_4)_3$	399.91	200.8	0.50
$UO_2.SO_4$	366.09	183.8	0.50
$FeSO_4$	151.91	152.6	1.00
Reaction 2			
V^{3+}	181.88	363.1	2.00
$Fe_2(SO_4)_3$	399.91	1,596.7	3.99
V^{5+}	163.01	325.4	2.00
$FeSO_4$	151.91	606.5	3.99
Reaction 3			
Mo^{4+}	160.10	55.1	0.34
$Fe_2(SO_4)_3$	399.91	275.4	0.69
Mo^{6+}	384.17	132.3	0.34
$FeSO_4$	151.91	104.6	0.69
TOTAL		2,072.92	kg/h
Total $FeSO_4$ produced		863.71	kg/h

Reaction	Moles	Reactant 1	Moles	Reactant 2	Moles	Reactant 3	Moles	Product 1	Moles	Product 2	Moles	Product 3	Moles	Product 4
SCRAP IRON CONSUMPTION														

Reaction 1: 1 H^{+} + 1 Fe^{0} + 1 $Fe^{3}+$ ⟶ 2 $Fe^{2}+$ + 1 H^{+}
kmoles: 16.11 16.11 32.23
(100% conversion)

Compound	M_r	Mass (kg)	kmoles
Reaction 1			
Fe^{0}	55.85	900.0	16.11
$Fe^{3}+$	55.85	900.0	16.11
$Fe^{2}+$	55.85	1,800.0	32.23

REAGENT CONSUMPTION SHEET

Rate of Fe consumption: 5.0g Fe per dm^3 of PLS (180m^3/hr) As specified by R.Raiter.

TOTAL 900.00 kg/h

Reaction	Moles	Reactant 1	Moles	Reactant 2	Moles	Reactant 3	Moles	Product 1	Moles	Product 2	Moles	Product 3	Moles	Product 4

CALCIUM HYDROXIDE PRODUCTION

Reaction 1:	1	CaO	+	1	H_2O	⟶	1 $Ca(OH)_2$
kmoles:	5.00			5.00			5.00

1.0 (100.0% conversion)

Compound	M_r	Mass (kg)	kmoles
Reaction 1			
CaO	56.08		
H_2O	18.02		
$Ca(OH)_2$	74.10		

Reaction	Moles	Reactant 1	Moles	Reactant 2	Moles	Reactant 3	Moles	Product 1	Moles	Product 2	Moles	Product 3	Moles	Product 4

FLOCCULANT CONSUMPTION

Assumed flocculant consumption rate: 30.00 g/t

Processing Area	Solids (t/h) Flowrate	Number of Thickeners	Floc to TK1 (g/t)	Floc to TK2 (g/t)	Total Floc per t solids	Total Floc Required (g/h)
Ferric Leach CCD	151.36	2	30.0	15.0	45.0	6,811.1
Mo Precipitation	#REF!	1	30.0	0	30.0	#REF!
Uranium Precipitation	0.17	1	30.0	0	30.0	5.1
Vanadium Precipitation	0.47	1	30.0	0	30.0	14.0
					TOTAL	#REF! g/h

Reaction	Moles	Reactant 1	Moles	Reactant 2	Moles	Reactant 3	Moles	Product 1	Moles	Product 2	Moles	Product 3	Moles	Product 4

SULPHUR CONSUMPTION (SULPHURIC ACID PRODUCTION)

Reaction 1:	1	S	+	1	O_2	⟶	1 SO_2
kmoles:	891.94			891.94			891.94
Reaction 2:	1	SO_2	+	0.5	O_2	⟶	1 SO_3
kmoles:	891.94			445.9688			891.94
Reaction 3:	1	SO_3	+	1	H_2O	⟶	1 H_2SO_4
kmoles:	891.94			891.94			891.94

Note:

Compound	M_r	Mass (kg)	kmoles
Reaction 3			
H_2SO_4	98.09	87,490	891.94
H_2O	18.02	16,073	891.94
SO_3	80.07	71,417	891.94
Reaction 2			
SO_3	80.07	71,417	891.94
O_2	32.00	14,271	445.97
SO_2	64.07	57,146	891.94
Reaction 1			
SO_2	64.07	57,146	891.94
O_2	32.00	28,542	891.94
S	32.07	28,604	891.94

Reaction	Moles	Reactant 1	Moles	Reactant 2	Moles	Reactant 3	Moles	Product 1	Moles	Product 2	Moles	Product 3	Moles	Product 4

PYRITE CONSUMPTION (FERRIC PRODUCTION)

Reaction 1:	4	FeS_2	+	11	O_2	⟶	2 Fe_2O_3	+	8	SO_2
kmoles:	20.00			55.01			10.00		40.01	
Reaction 2:	1	Fe_2O_3	+	3	H_2SO_4	⟶	1 $Fe_2(SO_4)_3$	+	3	H_2O
kmoles:	10.00			30.01			10.00		30.01	

Compound	M_r	Mass (kg)	kmoles
Reaction 2			
$Fe_2(SO_4)_3$	399.91	4,000	10.00
H_2O	18.02	541	30.01
H_2SO_4	98.09	2,943	30.01
Fe_2O_3	159.70	1,597	10.00
Reaction 1			
Fe_2O_3	159.70	1,597	10.00

REAGENT CONSUMPTION SHEET

SULPHURIC ACID PRODUCTION FROM SO_2 OFFGAS OF PYRITE COMBUSTION

Reaction 3:	1 SO_2	+	0.5 O_2	⟶	1 SO_3	
kmoles:	40.01		20.00		40.01	
Reaction 4:	1 SO_3	+	1 H_2O	⟶	1 H_2SO_4	
kmoles:	40.01		40.01		40.01	

SO_2	64.07	2,563	40.01
O_2	32.00	1,760	55.01
FeS_2	119.99	2,400	20.00
Reaction 3			
SO_2	64.07	2,563	40.01
O_2	32.00	640	20.00
SO_3	80.07	3,204	40.01
Reaction 4			
SO_3	80.07	3,204	40.01
H_2O	18.02	721	40.01
H_2SO_4	98.09	3,924	40.01

Reaction	Moles	Reactant 1	Moles	Reactant 2	Moles	Reactant 3	Moles	Product 1	Moles	Product 2	Moles	Product 3	Moles	Product 4
PYRITE CONSUMPTION (POTENTIALLY USE AS SOURCE OF SULPHURIC ACID)														

Reaction 1:	3 FeS_2	+	8 O_2	⟶	1 Fe_3O_4	+	6 SO_2	
kmoles:	468.96						937.91	
Reaction 2:	2 SO_2	+	1 O_2	⟶	2 SO_3			
kmoles:	937.91				937.91			
Reaction 3:	1 SO_3	+	1 H_2O	⟶	1 H_2SO_4			
kmoles:					937.91			

Compound	M_r	Mass (kg)	kmoles
Reaction 1			
H_2SO_4	98.09	87,000	886.94
FeS_2	119.99	56,270	468.96

92 tph sulphuric acid required

92 tph Sulphuric = 937.91 kmoles/hr

468.96 kmoles/hr Pyrite = 56,270.16 kg/hr = 461,415.3 MTPA

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MASS BALANCE SUMMARY

Stream Description	Stream Units	ROM Material (Grizzly Screen Feed)	Grizzly Screen U/S	Grizzly Screen O/S (Primary Crusher Feed)	Primary Crusher Discharge	SAG Mill Feed (Fresh + Recycle)	SAG Mill Discharge (Screen Feed)	Screen U/S	Screen O/S (SAG Mill Recycle)	TOTAL Ball Mill Sump Feed	Ball Mill Sump Slurry Dilution Water	Ball Mill Sump Discharge(Cyclone Feed)	Cyclone U/F (Ball Mill Feed)	Cyclone O/F (Sulphuric Acid Leach Feed)	Ball Mill Discharge	Sulphuric Acid to Ferric Leach	Pyrolusite to Ferric Leach	Ferric Sulphate to Sulphuric Acid Leach	Sulphuric Acid Leach Discharge (CCD Feed)	CCD Wash Water	CCD Overflow(PLS to IX - Molybdenum)	CCD Underflow (Sulphuric Acid Leach Residue to Tails)
Stream Number		**001**	**002**	**003**	**004**	**005**	**006**	**007**	**008**	**009**	**010**	**011**	**012**	**013**	**014**	**015**	**016**	**017**	**018**	**019**	**020**	**021**
DRY SOLIDS MASS FLOW RATE	t/h	122.0	122.0	122.0	122.0	305.0	305.0	122.0	183.0	366.0	-	366.0	244.0	122.0	244.0	-	0.8	2.1	151.4	-	-	151.4
DRY SOLIDS VOL FLOW RATE	m^3/h	34.9	34.9	34.9	34.9	87.1	87.1	34.9	52.3	104.6	-	104.6	69.7	34.9	69.7	-	0.2	0.67	59.6	-	-	43.2
TOTAL																						
TOTAL (PULP) MASS FLOW RATE	t/h	135.6	135.6	135.6	135.6	338.9	338.9	135.6	203.3	484.1	-	484.1	348.6	348.6	348.6	87.5	1.7	3.5	408.7	168.0	257.3	319.4
TOTAL (PULP) VOL FLOW RATE	m^3/h	48.4	48.4	48.4	48.4	121.03	121.0	48.4	72.6	222.7	-	222.7	174.3	261.4	174.3	47.5	1.0	2.05	359.4	168.0	245.1	198.8
PULP DENSITY (%$^w/_w$ Solids)	% $^w/_w$	90.00%	90.00%	90.00%	90.00%	90.00%	90.00%	90.00%	90.00%	52.50%	0.00%	52.50%	70.00%	35.00%	70.00%	0.00%	50.00%	60.00%	37.03%	0.00%	0.00%	47.39%
LIQUIDS																						
URANIUM	t/h	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.120	-	0.120	-
VANADIUM	t/h	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.203	-	0.203	-
MOLYBDENUM	t/h	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.033	-	0.033	-
PHOSPHORUS	t/h	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	3.945	-	3.945	-
NICKEL	t/h	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.182	-	0.182	-
ZINC	t/h	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.401	-	0.401	-
MANGANESE	t/h	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.271	-	0.271	-
RE (Y & Nd)	t/h	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.047	-	0.047	-
ALL OTHERS	t/h	-	-	-	-	-	-	-	-	-	-	-	-	-	-	87.5	-	-	11.987	-	11.987	-
SOLIDS																						
URANIUM	t/h	0.122	0.122	0.122	0.122	0.305	0.305	0.122	0.183	0.366	-	0.366	0.244	0.122	0.244	-	-	-	0.024	-	-	0.024
VANADIUM	t/h	0.279	0.279	0.279	0.279	0.697	0.697	0.279	0.418	0.836	-	0.836	0.557	0.279	0.557	-	-	-	0.075	-	-	0.075
MOLYBDENUM	t/h	0.065	0.065	0.065	0.065	0.162	0.162	0.065	0.097	0.194	-	0.194	0.130	0.065	0.130	-	-	-	0.032	-	-	0.032
PHOSPHORUS	t/h	3.945	3.945	3.945	3.945	9.861	9.861	3.945	5.917	11.834	-	11.834	7.889	3.945	7.889	-	-	-	-	-	-	-
NICKEL	t/h	0.303	0.303	0.303	0.303	0.758	0.758	0.303	0.455	0.910	-	0.910	0.606	0.303	0.606	-	-	-	0.121	-	-	0.121
ZINC	t/h	0.409	0.409	0.409	0.409	1.023	1.023	0.409	0.614	1.228	-	1.228	0.819	0.409	0.819	-	-	-	0.008	-	-	0.008
MANGANESE	t/h	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.528	-	0.257	-	-	0.3
RE (Y & Nd)	t/h	0.054	0.054	0.054	0.054	0.135	0.135	0.054	0.081	0.162	-	0.162	0.108	0.054	0.108	-	-	-	0.007	-	-	0.007
ALL OTHERS	t/h	116.8	116.8	116.8	116.8	292.1	292.1	116.8	175.2	350.5	-	350.5	233.6	116.8	233.6	-	-	-	150.8	-	-	150.8
WATER																						
WATER MASS FLOW RATE	t/h	13.6	13.6	13.6	13.6	33.89	33.9	13.6	20.3	331.1	213.0	331.1	104.6	226.6	104.6	-	0.8	1.4	240.2	168.0	240.2	168.0
WATER VOL FLOW RATE	m^3/h	13.6	13.6	13.6	13.6	33.89	33.9	13.6	20.3	331.1	213.0	331.1	104.6	226.6	104.6	-	0.8	1.4	240.2	168.0	240.2	168.0
DRY SOLIDS S.G.	t/m^3	3.5	3.5	3.5	3.5	3.5	3.5	3.5	3.5	3.5	3.5	3.5	3.5	3.5	3.5	3.5	3.5	3.5	3.5	3.5	3.5	3.5
WATER S.G.	t/m^3	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0
PULP S.G.	t/m^3	3.25	3.25	3.25	3.25	3.25	3.25	3.25	3.25	2.31	1.00	2.31	2.75	1.88	2.75	1.84	1.67	1.68	1.14	1.00	1.05	1.61

MASS BALANCE SUMMARY

Stream Description	IX - Mo Discharge (to IX -U & V)	IX - Mo Water Wash Feed	IX - Mo Water Wash Discharge	IX - Mo 4% NH_4OH Elution Feed	IX - Mo 4% NH_4OH Elution Discharge	IX - Mo Water Wash 2 Feed	IX - Mo Water Wash 2 Discharge	Mo IX Elution Liquor Storage Tank Feed	Mo IX Elution Liquor Storage Tank Discharge (to Mo Precipitation Tank)	$Ca(OH)_2$ to Mo Precipitation Tank	Mo Precipitation - Discharge Slurry (Filter Feed Slurry)	Mo S/L Separation - Solids (Mo Product)	Mo S/L Separation - Liquids (to WWT)	IX -U & V Discharge (to Phosphoric Acid SX)	IX -U & V Water Wash Feed	IX -U & V Water Wash Discharge	IX -U & V Sulphuric Acid Elution Feed	IX -U & V Sulphuric Acid Elution Discharge	IX -U & V Water Wash 2 Feed	IX -U & V Water Wash 2 Discharge	U & V Elution Liquor Storage Tank Feed	U & V IX Elution Liquor Storage Tank Discharge (to U Precipitation Tank)
Stream Number	**022**	**023**	**024**	**025**	**026**	**027**	**028**	**029**	**030**	**031**	**032**	**033**	**034**	**035**	**036**	**037**	**038**	**039**	**040**	**041**	**042**	**043**
DRY SOLIDS MASS FLOW RATE	-	-	-	-	-	-	-	-	-	0.026	0.069	0.069	-	-	-	-	-	-	-	-	-	-
DRY SOLIDS VOL FLOW RATE	-	-	-	-	-	-	-	-	-	0.012	0.02	0.016	-	-	-	-	-	-	-	-	-	-
TOTAL																						
TOTAL (PULP) MASS FLOW RATE	257.3	4.5	5.5	2.2	2.4	4.5	4.5	11.2	11.2	0.13	11.4	0.070	11.3	256.6	13.5	13.5	7.1	7.4	13.5	13.5	21.06	21.06
TOTAL (PULP) VOL FLOW RATE	245.1	4.5	5.5	2.25	2.25	4.5	4.5	10.4	10.4	0.11	11.29	0.016	11.27	244.4	13.5	13.5	6.8	6.8	13.5	13.5	20.3	20.3
PULP DENSITY (%$^w/_w$ Solids)	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	20.00%	0.61%	98.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%
LIQUIDS																						
URANIUM	0.120	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.120	-	-	0.120	0.120
VANADIUM	0.203	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.203	-	-	0.203	0.203
MOLYBDENUM	-	-	-	-	0.033	-	-	0.033	0.033	-	-	-	-	-	-	-	-	-	-	-	-	-
PHOSPHORUS	3.945	-	-	-	-	-	-	-	-	-	-	-	-	3.945	-	-	-	-	-	-	-	-
NICKEL	0.182	-	-	-	-	-	-	-	-	-	-	-	-	0.182	-	-	-	-	-	-	-	-
ZINC	0.401	-	-	-	-	-	-	-	-	-	-	-	-	0.401	-	-	-	-	-	-	-	-
MANGANESE	0.271	-	-	-	-	-	-	-	-	-	-	-	-	0.271	-	-	-	-	-	-	-	-
RE (Y & Nd)	0.047	-	-	-	-	-	-	-	-	-	-	-	-	0.047	-	-	-	-	-	-	-	-
ALL OTHERS	11.987	-	-	0.090	0.087	-	-	0.087	0.087	-	0.041	-	0.041	11.987	-	-	0.7	0.7	-	-	0.7	0.7
SOLIDS																						
URANIUM	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
VANADIUM	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
MOLYBDENUM	-	-	-	-	-	-	-	-	-	-	0.033	0.033	-	-	-	-	-	-	-	-	-	-
PHOSPHORUS	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
NICKEL	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
ZINC	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
MANGANESE	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
RE (Y & Nd)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
ALL OTHERS	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
WATER																						
WATER MASS FLOW RATE	240.2	4.5	5.5	2.01	2.16	4.5	4.5	11.1	11.1	0.102	11.250	0.001	11.249	239.8	13.5	13.5	6.37	6.66	13.5	13.5	19.9	19.9
WATER VOL FLOW RATE	240.2	4.5	5.5	2.01	2.16	4.5	4.5	11.1	11.1	0.102	11.250	0.001	11.249	239.8	13.5	13.5	6.37	6.66	13.5	13.5	19.9	19.9
DRY SOLIDS S.G.	3.5	3.5	3.5	3.5	3.5	3.5	3.5	3.5	3.5	3.5	4.4	4.4	4.4	4.4	4.4	4.4	4.4	4.4	4.4	4.4	4.4	4.4
WATER S.G.	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0
PULP S.G.	1.05	1.00	1.00	0.99	0.99	1.00	1.00	1.08	1.08	1.77	1.01	4.35	1.00	1.05	1.00	1.00	1.05	1.05	1.00	1.00	1.04	1.04

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MASS BALANCE SUMMARY

Stream Description	NH_4OH to U Precipitation Tank	H_2O_2 to U Precipitation Tank	U Precipitation Discharge Slurry (Filter Feed)	Uranium S/L Separation - Solids (U Product)	Uranium S/L Separation - Liquids (to V Precipitation)	NH_4OH to V Precipitation Tank	V Precipitation Discharge Slurry (Filter Feed)	Vanadium S/L Separation - Solids (V Product)	Vanadium S/L Separation - Liquids (to WWT)	Phosphoric Acid SX - Aqueous Feed (PLS)	Phosphoric Acid SX - Mixed Organic Solvent	Phosphoric Acid SX - Loaded Organic Discharge	Phosphoric Acid SX - Raffinate Discharge (to RE Precipitation Tank)	Phosphoric Acid SX - Organic Stripping Agent (H_2O)	Phosphoric Acid SX - Barren Organic Discharge (recycled for re-use)	Phosphoric Acid SX - Organic Stripping Agent (H_2O + H_3PO_4 to Storage Tank)	H_3PO_4 Storage Tank Discharge	H_3PO_4 Evaporator Discharge Product	H_3PO_4 Evaporator Discharge Steam	RE Precipitation Reagent - NH_4OH	RE Precipitation Tank Discharge (to S/L Separation)	RE Precipitation - S/L Separation (Solids Discharge to Dryer)
Stream Number	**044**	**045**	**046**	**047**	**048**	**049**	**050**	**051**	**052**	**053**	**054**	**055**	**056**	**057**	**058**	**059**	**060**	**061**	**062**	**063**	**064**	**065**
DRY SOLIDS MASS FLOW RATE	-	-	0.170	0.170	-	-	0.467	0.467	-	-	-	-	-	-	-	-	-	-	-	-	0.072	0.072
DRY SOLIDS VOL FLOW RATE	-	-	0.036	0.052	-	-	0.200	0.200	-	-	-	-	-	-	-	-	-	-	-	-	0.015	0.015
TOTAL																						
TOTAL (PULP) MASS FLOW RATE	0.84	0.057	21.9	0.189	21.76	0.233	21.99	0.52	21.47	256.6	240.0	192.5	256.6	52.0	240.0	52.0	52.0	14.9	37.1	1.199	257.8	0.080
TOTAL (PULP) VOL FLOW RATE	0.90	0.052	21.2	0.099	21.14	0.252	21.32	0.25	21.07	244.4	180.0	186.6	250.2	52.0	180.0	46.2	46.2	9.1	-	1.297	240.6	0.023
PULP DENSITY (%$^w/_w$ Solids)	0.00%	0.00%	0.77%	90.00%	0.00%	0.00%	2.12%	90.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.03%	90.00%
LIQUIDS																						
URANIUM	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
VANADIUM	-	-	0.2	-	0.20	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
MOLYBDENUM	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
PHOSPHORUS	-	-	-	-	-	-	-	-	-	3.945	-	-	-	-	-	3.945	3.945	3.945	-	-	-	-
NICKEL	-	-	-	-	-	-	-	-	-	0.182	-	-	0.182	-	-	-	-	-	-	-	-	-
ZINC	-	-	-	-	-	-	-	-	-	0.401	-	-	0.401	-	-	-	-	-	-	-	-	-
MANGANESE	-	-	-	-	-	-	-	-	-	0.028	-	-	0.028	-	-	-	-	-	-	-	-	-
RE (Y & Nd)	-	-	-	-	-	-	-	-	-	0.047	-	-	0.047	-	-	-	-	-	-	-	-	-
ALL OTHERS	0.5	0.017	1.3	0.051	1.09	0.140	0.933	0.002	0.9	228.6	240.0	192.5	229.7	-	240.0	8.537	8.537	8.537	-	0.72	230.55	0.007
SOLIDS																						
URANIUM	-	-	0.120	0.120	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
VANADIUM	-	-	-	-	-	-	0.203	0.203	-	-	-	-	-	-	-	-	-	-	-	-	-	-
MOLYBDENUM	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
PHOSPHORUS	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
NICKEL	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
ZINC	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
MANGANESE	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
RE (Y & Nd)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.047	0.047
ALL OTHERS	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
WATER																						
WATER MASS FLOW RATE	0.3	0.040	20.3	0.018	20.46	0.093	20.6	0.050	20.5	13.8	-	-	26.3	52.0	-	39.5	39.5	2.4	37.1	0.48	27.16	0.001
WATER VOL FLOW RATE	0.3	0.0	20.3	0.018	20.46	0.093	20.6	0.050	20.5	13.8	-	-	26.3	52.0	-	39.5	39.5	2.4	37.1	0.48	27.16	0.001
DRY SOLIDS S.G.	4.4	4.4	3.3	3.3	3.3	3.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	4.7	4.7
WATER S.G.	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0
PULP S.G.	0.92	1.10	1.04	3.05	1.03	0.92	1.03	2.06	1.02	1.05	1.00	1.03	1.03	1.00	1.00	1.13	1.13	1.64	1.00	0.92	1.07	3.52

The Uranium Discovery Company

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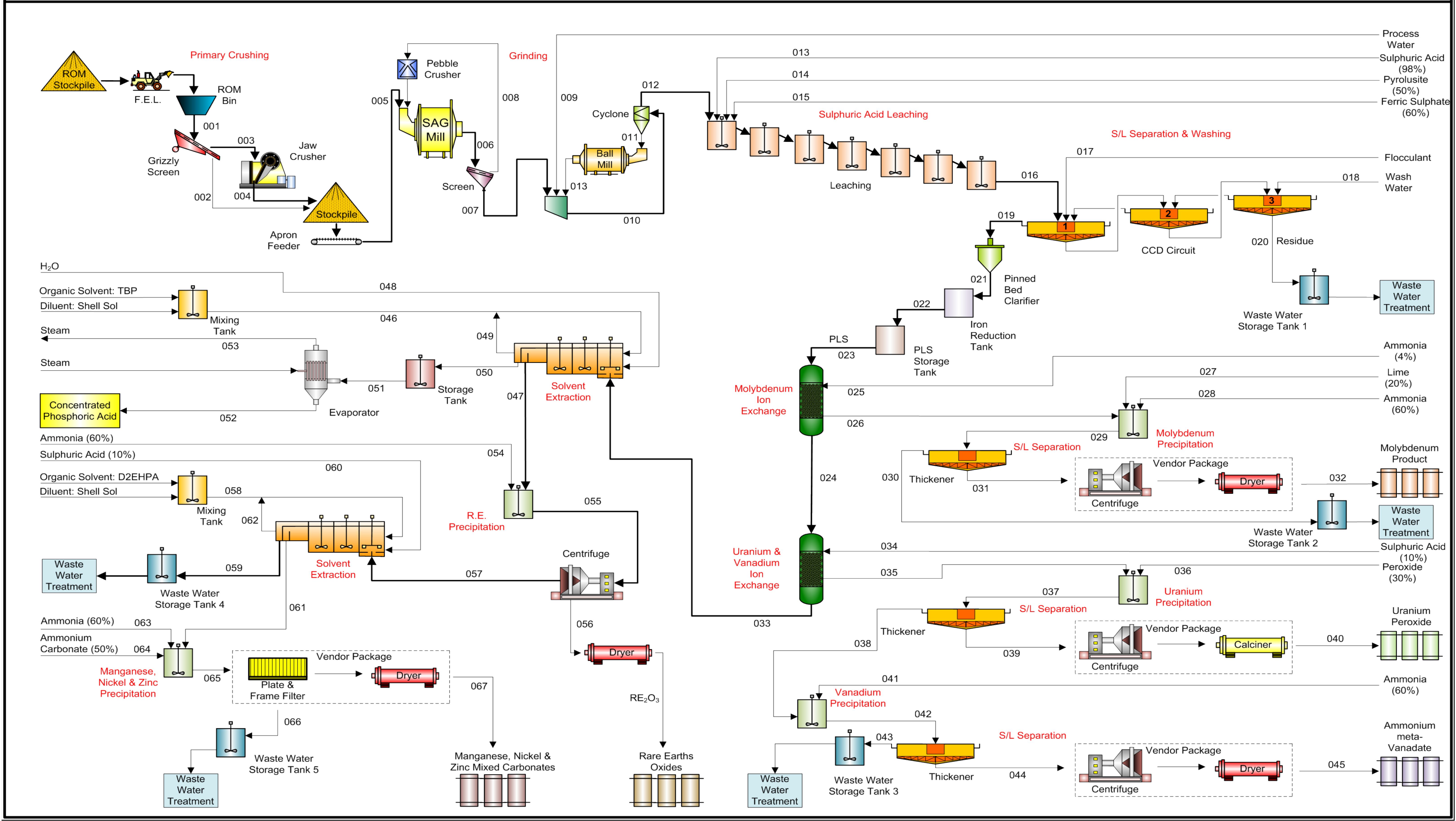
MINING & MINERALS

MASS BALANCE SUMMARY

Stream Description	RE Precipitation - S/L Separation (Liquids Discharge) (Mn, Ni & Zn SX Aqueous Feed (PLS))	Mn, Ni & Zn SX - Mixed Organic Solvent	Mn, Ni & Zn SX - Loaded Organic Discharge	Mn, Ni & Zn SX - Raffinate Discharge (to WWT Tank)	Mn, Ni & Zn SX - Organic Stripping Agent (H_2SO_4)	Mn, Ni & Zn SX - Loaded Organic Stripping Agent (H_2SO_4) (to Mn, Ni, Zn Precipitation Tank)	Mn, Ni & Zn SX - Barren Organic Discharge (recycled for re-use)	Mn, Ni & Zn Neutralizing Agent -NH_4OH (Ammonium Hydroxide)	Mn, Ni & Zn Precipitation Reagent -$(NH_4)_2CO_3$ (Ammonium Carbonate)	Mn, Ni & Zn Precipitation Tank - Discharge Slurry	Mn, Ni & Zn Precipitation Tank - Filter Feed Slurry	Mn, Ni & Zn Precipitation - S/L Separation (Solids Discharge) to Dryer	Mn, Ni & Zn Precipitation - S/L Separation (Liquids Discharge) to WWT	Mn, Ni & Zn Precipitation - Dryer Solids Discharge	Mn, Ni & Zn Precipitation - Dryer Steam Discharge	NET PROCESS WATER REQUIREMENT
Stream Number	066	067	068	069	070	071	072	073	074	075	076	077	078	079	080	081
DRY SOLIDS MASS FLOW RATE	-	-	-	-	-	-	-	-	-	1.19	1.19	1.19	-	1.195	-	-
DRY SOLIDS VOL FLOW RATE	-	-	-	-	-	-	-	-	-	0.28	0.28	0.28	-	0.275	-	-
TOTAL																
TOTAL (PULP) MASS FLOW RATE	257.7	240.0	241.5	255.2	21.0	22.1	240.0	2.44	3.92	25.44	25.44	1.33	24.11	1.219	0.11	-
TOTAL (PULP) VOL FLOW RATE	238.6	180.0	240.4	236.3	20.0	20.0	180.0	2.64	3.14	22.76	22.76	0.40	22.36	0.300	0.10	-
PULP DENSITY (%$^w/_w$ Solids)	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	4.70%	4.70%	90.00%	0.00%	98.00%	0.00%	0.00%
LIQUIDS																
URANIUM	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
VANADIUM	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
MOLYBDENUM	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
PHOSPHORUS	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
NICKEL	0.182	-	0.182	-	-	0.182	-	-	-	-	-	-	-	-	-	-
ZINC	0.401	-	0.401	-	-	0.401	-	-	-	-	-	-	-	-	-	-
MANGANESE	0.028	-	0.028	-	-	0.028	-	-	-	-	-	-	-	-	-	-
RE (Y & Nd)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
ALL OTHERS	229.9	240.0	241.5	227.7	2.1	2.987	240.0	1.46	2.35	4.05	4.05	0.02	4.0	-	0.02	-
SOLIDS																
URANIUM	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
VANADIUM	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
MOLYBDENUM	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
PHOSPHORUS	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
NICKEL	-	-	-	-	-	-	-	-	-	0.182	0.182	0.182	-	0.182	-	-
ZINC	-	-	-	-	-	-	-	-	-	0.401	0.401	0.401	-	0.401	-	-
MANGANESE	-	-	-	-	-	-	-	-	-	0.028	0.028	0.028	-	0.028	-	-
RE (Y & Nd)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
ALL OTHERS	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
WATER																
WATER MASS FLOW RATE	27.2	-	-	27.6	18.9	18.5	-	0.98	1.57	20.19	20.19	0.11	20.1	0.024	0.09	-
WATER VOL FLOW RATE	27.2	-	-	27.6	18.9	18.5	-	0.98	1.57	20.19	20.19	0.11	20.1	0.024	0.09	-
DRY SOLIDS S.G.	4.7	4.7	4.7	4.7	4.7	4.7	4.7	4.7	4.7	4.3	4.3	4.3	4.3	4.3	4.3	4.3
WATER S.G.	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0
PULP S.G.	1.08	1.00	1.00	1.08	1.05	1.10	1.00	0.92	1.25	1.12	1.12	3.33	1.08	4.07	1.10	1.00

Berlin Option B – Mass Balance Stream Numbers



U3O8 CORP. PRELIMINARY ECONOMIC ASSESSMENT ON THE BERLIN DEPOSIT, COLOMBIA

Report Prepared for

U3O8 Corp.

APPENDIX J: OPTION A PROCESS DESIGN CRITERIA

Report Prepared by

Tenova Mining & Minerals (Australia) Pty Ltd

M6088.A

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U3O8 CORPORATION

BERLIN, COLOMBIA

PRELIMINARY ECONOMIC ASSESSMENT STUDY

PROCESS DESIGN CRITERIA – OPTION A

M6088.A-P670-001

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0	20/11/12	ISSUED FOR IFU	J.SCHLOFFER	R.RAITER	P.NIEMANN	A.WALKER	
C	6/11/12	ISSUED FOR ICR	J.SCHLOFFER	R.RAITER	P.NIEMANN		
B	31/10/12	ISSUED FOR ICR	J.SCHLOFFER	R.RAITER	P.NIEMANN		
A	10/10/12	ISSUED FOR IDR	J.SCHLOFFER	R.RAITER			
REV	**DATE**	**REVISION HISTORY**	**ORGINATED**	**CHECKED**	**APPROVED**	**TENOVA APPROVAL**	**CLIENT APPROVAL**

Table of Contents

LIST OF TABLES

1 SOURCE OF DATA

Table 1-1 lists reference codes used in this document.

Table 1-1: Reference Codes

Code	Description
1	Client Supplied Data / Client Instruction
2	Previous Studies
3	Calculated Value
4	Bateman Data
5	Typical Data from Similar Operations
6	Mass Balance / Modelling
7	Test work / Consultant's Data
8	Literature, Engineering and Textbook Data
9	Regulatory Standards and Codes
10	Vendor Originated Data

2 DEFINITION OF TERMS IN TABLES

Table 1-1 lists the terms and abbreviations used in this document.

Table 2-1: Terms and Abbreviations

Nominal	Actual Operating Values
Design	Upper Limit to be Catered for in Design
Minimum	Lower Limit to be Catered for in Design
Ref	Reference to Source of Data
Rev	Revision Status
ASL	Above sea level
PLS	Pregnant liquor solution

3 SITE DATA

	Units	Design	Ref	Rev
3.1 General				
Site Location	-	Caldas Province, Columbia	1	A
Plant Site Elevation	m	800-900	1	C
Mine Site Elevation	m	Portal 861	1	C
Barometric Pressure	mbar	453.7	1	C
Latitude	deg	5 35 14.43N	1	C
Longitude	deg	74 58 0.7W	1	C
Design Life	years	20	1	C
3.2 Climate				
Rainfall				
Annual	mm	2653	1	C
Design Storm Values – 10 yr	mm / 24 h	220	1	C
Evaporation				
Annual	mm	17000	1	C
Humidity				
08:00 Average	%	73.9	1	C
14:00 Average	%	50.08	1	C
Wind				
Predominant Wind Direction	May-Oct	SSO – S – SSE	1	C
Predominant Wind Direction	Nov-Apr	-	1	C
Basic Wind Speed, Vu	m/s	23.9	1	C
Basic Wind Speed, Vp	m/s	1.24	1	C
Seismology				
Seismic Risk	-	-	1	C
Acceleration Co-efficient, a	-	-	1	C
Temperature				
Mean Annual	°C	22.24	1	C
Mean Annual Maximum	°C	26.9	1	C
Mean Annual Minimum	°C	18.74	1	C
Minimum	°C	16.2	1	C
Maximum	°C	34.7	1	C

3.3 Temperature					
Monthly Mean	**Monthly Mean Minimum** **°C**	**Monthly Mean Maximum** **°C**	**Average** **°C**	**Ref**	**Rev**
Jan	19.08	26.68	22.10	1	C
Feb	19.15	27.39	22.61	1	C
Mar	18.85	26.90	22.33	1	C
Apr	18.88	26.34	21.91	1	C
May	19.28	26.83	22.62	1	C
Jun	19.20	27.89	23.31	1	C
Jul	19.32	28.01	23.57	1	C
Aug	18.78	28.23	23.25	1	C
Sep	18.62	28.88	23.55	1	C
Oct	18.33	26.33	21.36	1	C
Nov	18.82	26.31	21.45	1	C
Dec	19.00	26.54	21.73	1	C

4 PRODUCTION CRITERIA

		Units	Nominal	Ref	Rev
4.1 ORE THROUGHPUT & MINERALOGY					
Ore Processing Flowrate		tpa	1,000,400	1	A
		tph	122.0	1	A
Ore Moisture Content		$\%^{w}/_{w}$	10.0	1	A
Ore Specific Gravity		-	3.50	1	A
Ore Bulk Density		kg/m^3	3.50	1	A
Resource Grades					
U_3O_8		ppm	1,179	1	A
V_2O_5		ppm	4,077	1	A
P_2O_5		ppm	73,950	1	A
Y_2O_3		ppm	461	1	A
Nd_2O_3		ppm	93	1	A
Elemental Grades					
U		ppm	929	1	A
V		ppm	2,284	1	A
P		ppm	32,300	1	A
Y		ppm	363	1	A
Nd		ppm	80	1	A
Re		ppm	5	1	A
Mo		ppm	531	1	A
Fe		ppm	8,130	1	A
Ni		ppm	2,485	1	A
Zn		ppm	2,561	1	A
S		ppm	9,700	1	A
Ag		ppm	2	1	A
Ore Mineralogy (1)					
Name	**Formula**				
Calcite	$CaCO_3$	%	49.80	1	A
Fluorapatite	$Ca_5(PO_4)_3F$	%	17.55	1	A
Quartz	SiO_2	%	16.30	1	A
Muscovite	$KAl_3Si_3O_{11}.H_2O$	%	8.50	1	A
Chlorite	$NaClO_2$	%	3.50	1	A
Pyrite	FeS_2	%	2.50	1	A
Dolomite	$CaMg(CO_3)_2$	%	1.35	1	A
Sphalerite	ZnS	%	0.50	1	A
Notes: (1) Mineralogy was approximated from mineralogical analysis.					

	Units	Nominal	Ref	Rev
4.2 OPERATING SCHEDULES				
Overall				
Operating Hours per Year	h	8200	4	A
Operating Hours per Day	d	24	4	A
Design Availability	%	93.6	4	A
Crushing				
Availability	%	85.0	4	A
Operating Hours	h/annum	7446	4	A
Grinding				
Availability	%	93.6	4	A
Operating Hours	h/annum	8200	4	A
Acetic Acid Leach & Regeneration				
Availability	%	93.6	4	A
Operating Hours	h/annum	8200	4	A
Ferric Leach				
Availability	%	93.6	4	A
Operating Hours	h/annum	8200	4	A
Ion Exchange				
Availability	%	93.6	4	A
Operating Hours	h/annum	8200	4	A
Solvent Extraction				
Availability	%	93.6	4	A
Operating Hours	h/annum	8200	4	A
Metals Recovery				
Availability	%	93.6	4	A
Operating Hours	h/annum	8200	4	A

5 CRUSHING

	Units	Nominal	Ref	Rev
Annual throughput	t/a (dry)	1,000,400	1	A
Operating throughput	t/h (dry)	134.3	4	A
5.1 ROM Feed				
Ore Delivery Method	-	Truck / FEL	4	A
Delivery Location	-	ROM Bin	4	B
Truck Size	t	30-60	4	B
Ore Moisture	$\%^{w}/_{w}$	5-10	1	A
ROM P_{100}	mm	500	1	A
ROM P_{80}	mm	110	1	A
ROM Bin Capacity	min	30	4	A
5.2 Product Screening 1				
Number of Screens	-	1	4	
Type	-	Open circuit	4	A
Screen Type	-	Static Grizzly	4	A
Aperture Size	mm	500	4	B
Number of Decks	-	1	4	A
Circulating Load	%	0	4	A
5.3 Product Screening 2				
Number of Screens	-	1	4	A
Type	-	Open circuit	4	A
Screen Type	-	Vibrating Grizzly Feeder	4	A
Aperture Size	mm	35	4	B
Number of Decks	-	1	4	A
Circulating Load	%	0	4	A
5.4 Primary Crusher				
Crusher Type	-	Jaw	4	A
Crusher Feed Method	-	Vibrating Grizzly	4	B
Feed Material Impact Work Index	kWh / t	10.0	1	A
Crusher Product Size – P_{100}	mm	70	6	B
Crusher Product Size – P_{80}	mm	35	6	A
Crusher Product Stockpile	hrs	24	4	A
Crusher Stockpile Discharge	-	Apron Feeder	4	A

6 GRINDING

	Units	Nominal	Ref	Rev
6.1 Primary Grinding				
Equipment Type	-	SAG Mill	4	A
Throughput per annum	tpa (dry)	1,000,400	1	A
Throughput per hour	tph (dry)	122	1	A
Solids fraction in mill	$\%^v/_v$	50.0	4	B
Recirculating Load	%	150	2	A
Ore Delivery Method	-	Conveyor	5	A
Feed Size - P_{100}	mm	70	6	A
Feed Size – P_{80}	mm	35	6	A
Grinding Work Index	kWh / t	10.0	1	A
Mill Product Size – P_{100}	mm	5	6	A
Mill Product Size – P_{80}	mm	1.5	6	A
6.2 Pebble Crushing				
Equipment Type	-	Cone Crusher	4	A
Expected throughput per hour	tph (dry)	24.4	4	A
6.3 Product Classification				
Classification Method	-	Screen	4	A
Number of Screens	-	1	5	A
Type	-	Closed circuit	5	A
Screen Type	-	Wet	5	B
Number of Decks	-	1	4	A
Screen Cut Size	mm	2.0	4	A
Screen Oversize – P_{80}	mm	5.0	4	A
Screen Undersize – P_{80}	mm	1.5	4	A
6.4 Secondary Grinding				
Equipment Type	-	Ball Mill	4	A
Throughput per annum	t/a (dry)	1,000,400	1	A
Throughput per hour	t/h (dry)	122	1	A
Recirculating Load	%	200.0	5	A
Feed Slurry Solids Density	%	50.0	5	A
Slurry Dilution Source 1	-	Process Water	1	A
Slurry Dilution Source 2	-	Leach Wash Water	1	A
Ore Delivery Method	-	Slurry Pump	4	A
Feed Size – P_{80}	mm	1.5	4	B
Bond Work Index	kWh / t	10.0	4	B
Grinding Product Size – P_{80}	mm	0.05	4	B

	Units	Nominal	Ref	Rev
6.5 Product Classification				
Classification Method	-	Cyclone	4	A
Number of Cyclones	-	2	5	A
Type	-	Closed circuit	5	A
Cyclone Type	-	Wet	5	A
Cyclone Feed Density	$\%^{w}/_{w}$	35.0	1	A
Cyclone Cut Size	mm	0.125	4	B
Cyclone O/F Product – P_{80}	mm	0.05	4	B
Cyclone U/F Product – P_{80}	mm	0.8 – 1.0	4	B

7 ACETIC ACID LEACH

	Units	Nominal	Ref	Rev
7.1 Leach Conditions				
Leach pulp density	$\%^w/_w$	12.0	1	A
Leach temperature	°C	25	1	A
Leach residence time	h	0.5	1	A
7.2 Leach Reagents				
Acetic Acid composition	g/L	86.6	1	A
Solids content	$\%^w/_w$	38.0	1	A
Total Acetic Acid applied	kg/t	615	1	A
Fresh Acetic Acid applied	kg/t	16	1	A
Regenerated Acetic Acid applied	kg/t	599	1	A
7.3 Extent Of Reactions				
Calcite to Calcium Acetate	%	100.0	1	A
Fluorapatite reaction	%	0	1	A
Uranium leaching	%	5.0	1	A
Nickel leaching	%	10.0	1	A
7.4 Acetic Acid Regeneration				
Sulphuric Acid required for 100% regeneration	kg/t	488	1	A
Sulphuric Acid concentration	$\%^w/_w$	98.0	5	A
Regeneration time	mins	30	7	A
Regeneration temperature	°C	25	1	A
Extent of Calcium Acetate conversion	%	100.0	7	A
S/L separation method	-	Thickeners	4	A
Number of thickeners	-	2	4	A
Thickener configuration	-	Parallel	4	A
Thickener U/F density	$\%^w/_w$	55.0	5	A
Acetic Acid regenerated	kg/t	599	1	A
7.5 Gypsum Production				
Gypsum production rate	kg/t	857	1	A
Washed moisture content of cake	$\%^w/_w$	10.0	1	A
Solid liquid separation method 1				
Thickener Diameter	M	15.0	4	B
Thickener feed solids density	$\%^w/_w$	10.9	4	B
Thickener U/F solids density	$\%^w/_w$	50.0	4	B

	Units	Nominal	Ref	Rev
Solid liquid separation method 2				
Belt Filter belt	-	Belt filter	4	B
Belt Filter Feed solids density	$\%^{w}/_{w}$	50.0	4	B
Wash water mass flowrate	t/h	100.0	4	B

8 FERRIC LEACH

	Units	Nominal	Ref	Rev
8.1 Leaching Conditions				
Leach pulp density	$\%^{w}/_{w}$	35.0	1	A
Leach temperature	°C	65	1	A
Leach residence time	h	10	7	A
Leach tank configuration (rows x number of tanks)	-	2 x 7	4	B
Sulphuric Acid concentration	$\%^{w}/_{w}$	98.0	5	A
Total Sulphuric Acid applied	kg/t	55	1	A
Total Ferric Sulphate applied per volume of leach slurry	kg/m^3	8.0	1	B
Total Pyrolusite applied	kg/t	1.4	4	A
PLS S.G.	-	1.05	7	A
8.2 Leaching Extent of Reactions				
Ca	%	14.0	1	A
Mg	%	34.0	1	A
U	%	93.0	1	A
V	%	69.0	1	A
P	%	100.0	1	A
Y	%	86.0	1	A
Nd	%	61.0	1	A
Re	%	18.0	1	A
Mo	%	46.0	1	A
Fe	%	96.0	1	A
Ni	%	51.0	1	A
Zn	%	88.0	1	A
Ag	%	0	1	A
8.3 Solid Liquid Separation				
S/L separation method	-	CCD Circuit	4	A
Number of CCDs	-	2	5	A
Configuration	-	Series	4	A
Solids loading	$t/h.m^2$	1.0	5	A
Thickener U/F density	$\%^{w}/_{w}$	50.0	5	A
Wash water volumetric flow rate	m^3/h	95.0	6	B
8.4 Iron Reduction				
Iron reduction material	-	Raw, washed Fe	4	A

9 MOLYBDENUM ION EXCHANGE

	Units	Nominal	Ref	Rev
9.1 Molybdenum Ion Exchange				
Number of columns	-	3	4	A
Resin type	-	Chelating	5	A
Loading equivalent	eqv/L	1.0	4	A
Mo recovery	$\%^{w}/_{w}$	98.0	5	A
Stripping fluid	-	NH_4OH (4%)	5	A
Washing fluid	-	H_2O	5	A
Cycle duration (load, backup, strip)	h	8.0	4	A
Operating Temperature	°C	25	4	B

10 MOLYBDENUM RECOVERY

	Units	Nominal	Ref	Rev
10.1 Molybdenum Precipitation				
Recovery method	-	Precipitation	4	A
Precipitation product	-	$CaMoO_4$	5	A
Precipitation pH	-	4	8	A
Precipitation temperature	°C	25	8	A
Alkaline fluid for pH control	-	NH_4OH (60% $^w/_w$)	5	A
10.2 Solid Liquid Separation				
Solid/liquid separation method 1	-	Thickener	5	A
Solids loading	$t/h.m^2$	1.0	4	A
Thickener U/F density	% $^w/_w$	50.0	5	A
Solid/liquid separation method 2	-	Centrifuge	5	A
Product solids density	% $^w/_w$	90.0	5	A
Solid/liquid separation method 3	-	Dryer	5	A
Dryer temperature	°C	100	5	A
Product solids density	% $^w/_w$	99.0	5	A

11 VANADIUM AND URANIUM ION EXCHANGE

	Units	Nominal	Ref	Rev
11.1 Vanadium and Uranium Ion Exchange				
Number of columns	-	9	4	A
Resin type	-	Tertiary amine	5	A
Loading equivalent (V_2O_5)	eqv/L	1.0	4	A
Loading equivalent (UO_2)	eqv/L	1.0	4	A
V recovery	$\%^w/_w$	98.0	5	A
U recovery	$\%^w/_w$	98.0	5	A
Stripping fluid	-	H_2SO_4 (10%$^w/_w$)	4	A
Washing fluid	-	H_2O	4	A
Cycle duration (load, backup, strip)	h	8.0	4	A
Operating Temperature	°C	25	4	B

12 URANIUM RECOVERY

	Units	Nominal	Ref	Rev
12.1 Uranium Precipitation				
Recovery method	-	Precipitation	4	A
Precipitation product	-	$UO_4.2H_2O$	5	A
Precipitation pH	-	4	5	A
Precipitation temperature	°C	25	5	A
Alkaline fluid for pH control	-	NH_4OH (60%$^w/_w$)	4	A
Precipitation reagent	-	H_2O_2 (30%$^w/_w$)	5	A
12.2 Solid Liquid Separation				
Solid/liquid separation method 1	-	Thickener	5	A
Solids loading	$t/h.m^2$	1.0	4	A
Thickener U/F density	%$^w/_w$	50.0	4	A
Solid/liquid separation method 2	-	Centrifuge	5	A
Product solids density	%$^w/_w$	90.0	4	A
Solid/liquid separation method 3	-	Calciner	5	A
Dryer temperature	°C	700	5	A
Product solids density	%$^w/_w$	99.0	4	A
Calciner product	-	U_3O_8	4	A

13 VANADIUM RECOVERY

	Units	Nominal	Ref	Rev
13.1 Vanadium Precipitation				
Recovery method	-	Precipitation	4	A
Precipitation product	-	NH_4VO_3	5	A
Precipitation pH	-	8	5	A
Precipitation temperature	°C	25	5	A
Alkaline fluid for pH control	-	NH_4OH (60%$^w/_w$)	4	A
13.2 Solid Liquid Separation				
Solid/liquid separation method 1	-	Thickener	5	A
Solids loading	$t/h.m^2$	1.0	4	A
Thickener U/F density	%$^w/_w$	50.0	4	A
Solid/liquid separation method 2	-	Centrifuge	5	A
Product solids density	%$^w/_w$	90.0	5	A
Solid/liquid separation method 3	-	Dryer	5	A
Dryer temperature	°C	100	5	A
Product solids density	%$^w/_w$	99.0	4	A

14 PHOSPHORIC ACID SOLVENT EXTRACTION

	Units	Nominal	Ref	Rev
14.1 Phosphoric Acid SX Circuit				
Extractant				
Name	-	TBP	5	A
Concentration	$\%^{v}/_{v}$	75.0	4	A
Diluent				
Name	-	ShellSol	5	A
Concentration	$\%^{v}/_{v}$	25.0	4	A
Entrainment Losses				
Organic losses	%	10.0	8	A
Plant Configuration				
Number of trains	-	1	4	A
Number of extract stages	per train	7	4	A
Number of wash stages	per train	3	4	A
Number of strip stages	per train	5	4	A
O:A mixing ratio	-	1 : 1	5	A
Retention time	min/ stage	2.0	4	A
Overall extraction	%	99.0	4	A
Settling retention time	min	2.0	5	A
Stripping agent	-	H_2O	4	A
Scrubbing agent	-	H_2O	4	A
Organic product compounds	-	Phosphoric acid (18$\%^{w}/_{w}$)	4	A
Raffinate product compounds	-	Rare earths	4	A
	-	Manganese, nickel & zinc	4	A

15 PHOSPHORIC ACID UPGRADE

	Units	Nominal	Ref	Rev
15.1 Phosphoric Acid Concentration				
Phosphoric acid feed concentration	$\%^{w}/_{w}$	24.3	6	A
Equipment type	-	Evaporator	4	A
Heat source	-	Steam/Diesel/Electric - TBD	5	A
Evaporator temperature	°C	>110	4	A
Phosphoric acid product concentration	$\%^{w}/_{w}$	83.7	4	A
Packaging container details	-	20.0 m^3 shipping container tank	1	A
Packaging container volume	m^3	20.0	5	A

16 RARE EARTH RECOVERY

	Units	Nominal	Ref	Rev
16.1 Rare Earth Precipitation				
Precipitation pH	-	6	8	A
pH control	-	NH_4OH (60%$^w/_w$)	4	A
Precipitation temperature	°C	25	8	A
Rare earth recovery	%	100	4	A
16.2 Solid Liquid Separation				
Solid/liquid separation method 1	-	Pressure filter	4	A
Product solids density	%$^w/_w$	90.0	5	A
Solid/liquid separation method 2	-	Dryer	4	A
Dryer temperature	°C	>110	5	A
Product solids density	%$^w/_w$	99.0	4	A
Product description	-	Rare earth oxides	4	A

17 MANGANESE, NICKEL & ZINC SOLVENT EXTRACTION

	Units	Nominal	Ref	Rev
17.1 Manganese, Nickel & Zinc SX Circuit				
Extractant				
Organic Name	-	D2EHPA	5	A
Concentration	$\%^{v}/_{v}$	30.0	4	A
Diluent				
Diluent Name		ShellSol	5	A
Concentration	$\%^{v}/_{v}$	70.0	4	A
Entrainment Losses				
Organic losses	%	10.0	8	A
Plant Configuration				
Number of trains		1	4	A
Number of extract stages	per train	7	4	A
Number of wash stages	per train	3	4	A
Number of strip stages	per train	5	4	A
O:A mixing ratio		1:1	5	A
Retention time	min/ stage	2.0	4	A
Overall extraction	%	98.0	4	A
Settling retention time	min	2.0	5	A
Stripping agent	-	Sulphuric acid ($10\%^{w}/_{w}$)	4	A
Scrubbing agent	-	H_2O	4	A
Organic product compounds	-	Waste water treatment	4	A
Raffinate product compounds	-	Manganese, nickel & zinc	4	A

18 MANGANESE, NICKEL & ZINC RECOVERY

	Units	Nominal	Ref	Rev
18.1 Manganese, Nickel & Zinc Precipitation				
Precipitation pH	-	8	8	A
pH control	-	NH_4OH (60%$^w/_w$)	4	A
Precipitating agent	-	$(NH_4)_2CO_3$ (50%$^w/_w$)	8	A
Precipitation temperature	°C	25	4	A
Mn, Ni, Zn recovery	%	98.0	4	A
18.2 Solid/Liquid Separation				
Solid/liquid separation method 1	-	Centrifuge	5	A
Product solids density	%$^w/_w$	90.0	4	A
Solid/liquid separation method 2	-	Dryer	4	A
Dryer temperature	°C	>110	4	A
Product solids density	%$^w/_w$	99.0	4	A
Product description	-	Manganese, nickel & zinc mixed carbonate	4	A

19 ACID PRODUCTION PLANT

	Units	Nominal	Ref	Rev
19.1 Sulphuric Acid Production Plant				
Production Method	-	Solid sulphur combustion	5	A
Conversion efficiency	%	91.0	8	A
Sulphur consumption*	t/d	637.0	6	A
Sulphuric acid production	t/d	1,947.0	6	A
* Note: potential exists to reduce sulphur consumption by utilising SO_2 off gas from ferric production to produce 4.0 t/h of sulfuric acid.				

20 WASTE WATER TREATMENT

	Units	Nominal	Ref	Rev
Target pH	-	8.0	5	A
20.1 Ferric Leach Wash Residue Liquid				
pH	-	8.0	5	A
Neutralising agent	-	Hydrated Lime	4	A
20.2 Molybdenum Recovery Solid/Liquid Separation Liquid				
pH	-	8.0	5	A
Neutralising agent	-	Hydrated Lime	4	A
20.3 Vanadium Precipitation Solid/Liquid Separation Liquid				
pH	-	8.0	5	A
Neutralising agent	-	Hydrated Lime	4	A
20.4 Mn, Ni & Zn Solvent Extraction Waste Liquid				
pH	-	8.0	5	A
Neutralising agent	-	Hydrated Lime	4	A
20.5 Mn, Ni & Zn Precipitation Solid/Liquid Separation Liquid				
pH	-	8.0	5	A
Neutralising agent	-	Hydrated Lime	4	A

21 TAILINGS NEUTRALISATION AND/OR PRECIPITATION

	Units	Nominal	Ref	Rev
21.1 Process Information				
Number of Stages	-	TBD	5	A
Design Residence Time	h / stage	1.0	4	A
Neutralising Source	-	Hydrated Lime	5	A
Target Final pH	-	8.0	5	A
21.2 Process Chemistry				
Waste water impurity precipitation	%	100	4	A

22 REAGENTS

	Units	Nominal	Ref	Rev
22.1 Acetic Acid (Glacial)				
Supply Form	-	Liquid	8	A
Supply Packaging	-	1 t Bulki Box	8	A
Supply Concentration	$\%^{w}/_{w}$	99.85	8	A
pH	-	2.4	8	A
Bulk Density	t / m^3	1.05	8	A
Storage Concentration	$\%^{w}/_{w}$	99.85	8	A
Addition Concentration	$\%^{w}/_{w}$	99.85	8	A
Viscosity at 25°C	cP	1.22	8	A
Flash Point	°C	40.0	8	A
22.2 Ammonium Carbonate				
Supply Form	-	Solids Powder	8	A
Supply Packaging	-	1 t bag	8	A
Supply Concentration	$\%^{w}/_{w}$	99.99	8	A
Bulk Density	t / m^3	1.50	8	A
Storage Concentration	$\%^{w}/_{w}$	99.99	-	A
Addition Concentration	$\%^{w}/_{w}$	99.99	-	A
Viscosity at 25°C	cP	Not applicable	-	A
Flash Point	°C	Not applicable	-	A
22.3 Ammonium Hydroxide (60%$^{w}/_{w}$)				
Supply Form	-	Liquid	8	A
Supply Packaging	-	1 t Bulki Box	8	A
Supply Concentration	$\%^{w}/_{w}$	60.00	8	A
pH	-	11.7	8	A
Bulk Density	t / m^3	0.92	8	A
Storage Concentration	$\%^{w}/_{w}$	60.00	8	A
Addition Concentration	$\%^{w}/_{w}$	60.00	8	A
Viscosity at 25°C	cP	1.3	8	A
Flash Point	°C	Not applicable	-	A
22.4 Diluent (ShellSol)				
Supply Form	-	Liquid	8	A
Supply Packaging	-	1 t Bulki Box	8	A
Supply Concentration	$\%^{w}/_{w}$	100.0	8	A
Bulk Density	t / m^3	0.75	8	A
Storage Concentration	$\%^{w}/_{w}$	100.0	8	A
Addition Concentration	$\%^{w}/_{w}$	100.0	8	A
Viscosity at 25°C	cP	0.91	8	A

	Units	Nominal	Ref	Rev
Flash Point	°C	27	8	A
22.5 Ferric Sulphate (60%$^{w}/_{w}$)				
Supply Form	-	Liquid	8	A
Supply Packaging	-	1 t Bulki Box	8	A
Supply Concentration	%$^{w}/_{w}$	60.00	8	A
Bulk Density	t / m^3	1.69	8	A
Storage Concentration	%$^{w}/_{w}$	60.00	8	A
Addition Concentration	%$^{w}/_{w}$	60.00	8	A
Viscosity at 25°C	cP	20	8	A
Flash Point	°C	Not applicable	8	A
22.6 Hydrated Lime (90%$^{w}/_{w}$)				
Supply Form	-	Solids powder	8	A
Supply Packaging	-	1 t bag	8	A
Supply Concentration	%$^{w}/_{w}$	90.0 – 95.0	8	A
pH	-	12	8	A
Bulk Density	t / m^3	0.8	8	A
Storage Concentration	%$^{w}/_{w}$	40.0	8	A
Addition Concentration	%$^{w}/_{w}$	90.0	8	A
Viscosity at 25°C	cP	20	8	A
Flash Point	°C	Not applicable	8	A
22.7 Hydrogen Peroxide (35%$^{w}/_{w}$)				
Supply Form	-	Liquid	8	A
Supply Packaging	-	1 t Bulki Box	8	A
Supply Concentration	%$^{w}/_{w}$	35.0	8	A
Bulk Density	t / m^3	1.11	8	A
Storage Concentration	%$^{w}/_{w}$	35.0	8	A
Addition Concentration	%$^{w}/_{w}$	35.0	4	A
Viscosity at 25°C	cP	1.1	8	A
Flash Point	°C	Not applicable	8	A
22.8 Pyrite				
Supply Form	-	Solids powder	8	A
Supply Packaging	-	1 t bag	8	A
Supply Concentration	%$^{w}/_{w}$	100.0	8	A
Bulk Density	t / m^3	5.0	8	A
Storage Concentration	%$^{w}/_{w}$	100.0	8	A
Addition Concentration	%$^{w}/_{w}$	100.0	8	A
Viscosity at 25°C	cP	Not applicable	8	A
Flash Point	°C	Not applicable	8	A

	Units	Nominal	Ref	Rev
22.9 Pyrolusite				
Supply Form	-	Solids powder	8	A
Supply Packaging	-	1 t bag	8	A
Supply Concentration	$\%^w/_w$	100.0	8	A
Bulk Density	t / m^3	5.0	8	A
Storage Concentration	$\%^w/_w$	100.0	8	A
Addition Concentration	$\%^w/_w$	100.0	8	A
Viscosity at 25°C	cP	Not applicable	8	A
Flash Point	°C	Not applicable	8	A
22.10 Organic Solvent - TBP				
Supply Form	-	Liquid	8	A
Supply Packaging	-	1 t Bulki Box	8	A
Supply Concentration	$\%^w/_w$	100.0	8	A
Bulk Density	t / m^3	0.98	8	A
Storage Concentration	$\%^w/_w$	100.0	8	A
Addition Concentration	$\%^w/_w$	100.0	8	A
Viscosity at 25°C	cP	3.80	8	A
Flash Point	°C	120	8	A
22.11 Organic Solvent – D2EHPA				
Supply Form	-	Liquid	8	A
Supply Packaging	-	1 t Bulki Box	8	A
Supply Concentration	$\%^w/_w$	>95%	8	A
Bulk Density	t / m^3	0.97	8	A
Storage Concentration	$\%^w/_w$	>95%	8	A
Addition Concentration	$\%^w/_w$	>95%	8	A
Viscosity at 25°C	cP	42	8	A
Flash Point	°C	196	8	A
22.12 Sulphur				
Supply Form	-	Solids powder	8	A
Supply Packaging	-	1 t bag	8	A
Supply Concentration	$\%^w/_w$	99.5	8	A
Bulk Density	t / m^3	2.05	8	A
Flash Point	°C	168	8	A
22.13 Sulphuric Acid				
Supply Form	-	Liquid – produced on site	8	A
Supply Packaging	-	Not applicable – stored on site	8	A
Supply Concentration	$\%^w/_w$	98%	8	A
Bulk Density	t / m^3	1.84	8	A
Storage Concentration	$\%^w/_w$	98%	8	A

	Units	Nominal	Ref	Rev
Addition Concentration	$\%^{w}/_{w}$	98% & 10%	8	A
Viscosity at 25°C	cP	26.7	8	A
Flash Point	°C	Not applicable	8	A
22.14 Washed Scrap Iron				
Supply Form	-	Solids	8	A
Supply Packaging	-	Not required	8	A
Bulk Density	t / m^3	5.2	8	A

23 SERVICES

	Units	Nominal	Ref	Rev
23.1 Water				
Raw Water				
Supply Water Quality	-	As supplied	4	A
Filtered Water				
Filter Supply Water Quality	-	Raw water	4	A
Gland Water				
Supply Water Quality	-	Filtered	4	A
Potable Water				
Supply Water Quality	-	Filtered	4	A
Product Water Quality Standard	-	Drinking Water Guidelines	4	A
Fire Water				
Supply Water Quality	-	Raw water	4	A
23.2 Power				
Plant Power Source	-	Grid electricity	1	A

The Uranium Discovery Company

U3O8 CORP. PRELIMINARY ECONOMIC ASSESSMENT ON THE BERLIN DEPOSIT, COLOMBIA

Report Prepared for

U3O8 Corp.

APPENDIX K: OPTION B PROCESS DESIGN CRITERIA

Report Prepared by

Tenova Mining & Minerals (Australia) Pty Ltd

M6088.A

U3O8 CORPORATION

BERLIN, COLOMBIA

PRELIMINARY ECONOMIC ASSESSMENT STUDY

PROCESS DESIGN CRITERIA – OPTION B

M6088.A-P670-003

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0	31/01/13	ISSUED FOR USE	J.SCHLOFFER	R.RAITER	P.NIEMANN	A.WALKER
B	25/01/13	ISSUED FOR ICR	J.SCHLOFFER	R.RAITER	P.NIEMANN	
A	20/11/12	ISSUED FOR IDR	J.SCHLOFFER	R.RAITER		
REV	**DATE**	**REVISION HISTORY**	**ORGINATED**	**CHECKED**	**APPROVED**	**TENOVA APPROVAL**

The Uranium Discovery Company U3O8 CORP

tenova MINING & MINERALS

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LIST OF TABLES

1 SOURCE OF DATA

Table 1-1 lists reference codes used in this document.

Table 1-1: Reference Codes

Code	Description
1	Client Supplied Data / Client Instruction
2	Previous Studies
3	Calculated Value
4	Bateman Data
5	Typical Data from Similar Operations
6	Mass Balance / Modelling
7	Test work / Consultant's Data
8	Literature, Engineering and Textbook Data
9	Regulatory Standards and Codes
10	Vendor Originated Data

2 DEFINITION OF TERMS IN TABLES

Table 2-1 lists the terms and abbreviations used in this document.

Table 2-1: Terms and Abbreviations

Nominal	Actual Operating Values
Design	Upper Limit to be Catered for in Design
Minimum	Lower Limit to be Catered for in Design
Ref	Reference to Source of Data
Rev	Revision Status
ASL	Above sea level
PLS	Pregnant liquor solution

3 SITE DATA

	Units	Design	Ref	Rev
3.1 General				
Site Location	-	Caldas Province, Columbia	1	A
Plant Site Elevation	m	800-900	1	A
Mine Site Elevation	m	Portal 861	1	A
Barometric Pressure	mbar	453.7	1	A
Latitude	deg	5 35 14.43N	1	A
Longitude	deg	74 58 0.7W	1	A
Design Life	years	20	1	A
3.2 Climate				
Rainfall				
Annual	mm	2653	1	A
Design Storm Values – 10 yr	mm / 24 h	220	1	A
Evaporation				
Annual	mm	17000	1	A
Humidity				
08:00 Average	%	73.9	1	A
14:00 Average	%	50.08	1	A
Wind				
Predominant Wind Direction	May-Oct	SSO – S – SSE	1	A
Predominant Wind Direction	Nov-Apr		1	A
Basic Wind Speed, Vu	m/s	23.9	1	A
Basic Wind Speed, Vp	m/s	1.24	1	A
Seismology				
Seismic Risk	-	-	1	A
Acceleration Co-efficient, a	-	-	1	A
Temperature				
Mean Annual	°C	22.24	1	A
Mean Annual Maximum	°C	26.9	1	A
Mean Annual Minimum	°C	18.74	1	A
Minimum	°C	16.2	1	A
Maximum	°C	34.7	1	A

3.3 Temperature					
Monthly Mean	Monthly Mean Minimum °C	Monthly Mean Maximum °C	Average °C	Ref	Rev
Jan	19.08	26.68	22.10	1	A
Feb	19.15	27.39	22.61	1	A
Mar	18.85	26.90	22.33	1	A
Apr	18.88	26.34	21.91	1	A
May	19.28	26.83	22.62	1	A
Jun	19.20	27.89	23.31	1	A
Jul	19.32	28.01	23.57	1	A
Aug	18.78	28.23	23.25	1	A
Sep	18.62	28.88	23.55	1	A
Oct	18.33	26.33	21.36	1	A
Nov	18.82	26.31	21.45	1	A
Dec	19.00	26.54	21.73	1	A

4 PRODUCTION CRITERIA

		Units	Nominal	Ref	Rev
4.1 ORE THROUGHPUT & MINERALOGY					
Ore Processing Flowrate		tpa	1,000,400	1	A
		tph	122.0	1	A
Ore Moisture Content		$\%^{w}/_{w}$	10.0	1	A
Ore Specific Gravity		-	3.50	1	A
Ore Bulk Density		kg/m^3	3.50	1	A
Resource Grades					
U_3O_8		ppm	1,179	1	A
V_2O_5		ppm	4,077	1	A
P_2O_5		ppm	73,950	1	A
Y_2O_3		ppm	461	1	A
Nd_2O_3		ppm	93	1	A
Elemental Grades					
U		ppm	929	1	A
V		ppm	2,284	1	A
P		ppm	32,300	1	A
Y		ppm	363	1	A
Nd		ppm	80	1	A
Re		ppm	5	1	A
Mo		ppm	531	1	A
Fe		ppm	8,130	1	A
Ni		ppm	2,485	1	A
Zn		ppm	2,561	1	A
S		ppm	9,700	1	A
Ag		ppm	2	1	A
Ore Mineralogy (1)					
Name	**Formula**				
Calcite	$CaCO_3$	%	49.80	1	A
Fluorapatite	$Ca_5(PO_4)_3F$	%	17.55	1	A
Quartz	SiO_2	%	16.30	1	A
Muscovite	$KAl_3Si_3O_{11}.H_2O$	%	8.50	1	A
Chlorite	$NaClO_2$	%	3.50	1	A
Pyrite	FeS_2	%	2.50	1	A
Dolomite	$CaMg(CO_3)_2$	%	1.35	1	A
Sphalerite	ZnS	%	0.50	1	A
Notes: (1) Mineralogy was approximated from mineralogical analysis.					

	Units	Nominal	Ref	Rev
4.2 OPERATING SCHEDULES				
Overall				
Operating Hours per Year	h	8200	4	A
Operating Hours per Day	d	24	4	A
Design Availability	%	93.6	4	A
Crushing				
Availability	%	85.0	4	A
Operating Hours	h/annum	7446	4	A
Grinding				
Availability	%	93.6	4	A
Operating Hours	h/annum	8200	4	A
Ferric Leach				
Availability	%	93.6	4	A
Operating Hours	h/annum	8200	4	A
Ion Exchange				
Availability	%	93.6	4	A
Operating Hours	h/annum	8200	4	A
Solvent Extraction				
Availability	%	93.6	4	A
Operating Hours	h/annum	8200	4	A
Metals Recovery				
Availability	%	93.6	4	A
Operating Hours	h/annum	8200	4	A

5 CRUSHING

	Units	Nominal	Ref	Rev
Annual throughput	t/a (dry)	1,000,400	1	A
Operating throughput	t/h (dry)	134.3	4	A
5.1 ROM Feed				
Ore Delivery Method	-	Truck / FEL	4	A
Delivery Location	-	ROM Bin	4	A
Truck Size	t	30-60	4	A
Ore Moisture	$\%^{w}/_{w}$	5-10	1	A
ROM P_{100}	mm	500	1	A
ROM P_{80}	mm	110	1	A
ROM Bin Capacity	min	30	4	A
5.2 Product Screening 1				
Number of Screens	-	1	4	
Type	-	Open circuit	4	A
Screen Type	-	Static Grizzly	4	A
Aperture Size	mm	500	4	A
Number of Decks	-	1	4	A
Circulating Load	%	0	4	A
5.3 Product Screening 2				
Number of Screens	-	1	4	A
Type	-	Open circuit	4	A
Screen Type	-	Vibrating Grizzly Feeder	4	A
Aperture Size	mm	35	4	A
Number of Decks	-	1	4	A
Circulating Load	%	0	4	A
5.4 Primary Crusher				
Crusher Type	-	Jaw	4	A
Crusher Feed Method	-	Vibrating Grizzly	4	A
Feed Material Impact Work Index	kWh / t	10.0	1	A
Crusher Product Size – P_{100}	mm	70	6	A
Crusher Product Size – P_{80}	mm	35	6	A
Crusher Product Stockpile	hrs	24	4	A
Crusher Stockpile Discharge	-	Apron Feeder	4	A

6 GRINDING

	Units	Nominal	Ref	Rev
6.1 Primary Grinding				
Equipment Type	-	SAG Mill	4	A
Throughput per annum	tpa (dry)	1,000,400	1	A
Throughput per hour	tph (dry)	122	1	A
Solids fraction in mill	$\%^v/_v$	50.0	4	A
Recirculating Load	%	150	2	A
Ore Delivery Method	-	Conveyor	5	A
Feed Size - P_{100}	mm	70	6	A
Feed Size – P_{80}	mm	35	6	A
Grinding Work Index	kWh / t	10.0	1	A
Mill Product Size – P_{100}	mm	5	6	A
Mill Product Size – P_{80}	mm	1.5	6	A
6.2 Pebble Crushing				
Equipment Type	-	Cone Crusher	4	A
Expected throughput per hour	tph (dry)	24.4	4	A
6.3 Product Classification				
Classification Method	-	Screen	4	A
Number of Screens	-	1	5	A
Type	-	Closed circuit	5	A
Screen Type	-	Wet	5	A
Number of Decks	-	1	4	A
Screen Cut Size	mm	2.0	4	A
Screen Oversize – P_{80}	mm	5.0	4	A
Screen Undersize – P_{80}	mm	1.5	4	A
6.4 Secondary Grinding				
Equipment Type	-	Ball Mill	4	A
Throughput per annum	t/a (dry)	1,000,400	1	A
Throughput per hour	t/h (dry)	122	1	A
Recirculating Load	%	200.0	5	A
Feed Slurry Solids Density	%	50.0	5	A
Slurry Dilution Source 1	-	Process Water	1	A
Slurry Dilution Source 2	-	Leach Wash Water	1	A
Ore Delivery Method	-	Slurry Pump	4	A
Feed Size – P_{80}	mm	1.5	4	A
Bond Work Index	kWh / t	10.0	4	A
Grinding Product Size – P_{80}	mm	0.05	4	A

	Units	Nominal	Ref	Rev
6.5 Product Classification				
Classification Method	-	Cyclone	4	A
Number of Cyclones	-	2	5	A
Type	-	Closed circuit	5	A
Cyclone Type	-	Wet	5	A
Cyclone Feed Density	$\%^{w}/_{w}$	35.0	1	A
Cyclone Cut Size	mm	0.125	4	A
Cyclone O/F Product – P_{80}	mm	0.05	4	A
Cyclone U/F Product – P_{80}	mm	0.8 – 1.0	4	A

7 SULPHURIC ACID LEACH

	Units	Nominal	Ref	Rev
7.1 Leaching Conditions				
Leach pulp density	$\%^w/_w$	35.0	1	A
Leach temperature	°C	65	1	A
Leach residence time	h	10	7	A
Leach tank configuration (rows x number of tanks)	-	2 x 7	4	A
Sulphuric Acid concentration	$\%^w/_w$	98.0	5	A
Total Sulphuric Acid applied	kg/t	55	1	A
Total Ferric Sulphate applied per volume of leach slurry	kg/m^3	8.0	1	A
Total Pyrolusite applied	kg/t	1.4	4	A
PLS S.G.	-	1.05	7	A
7.2 Leaching Extent of Reactions				
Ca	%	14.0	1	A
Mg	%	34.0	1	A
U	%	93.0	1	A
V	%	69.0	1	A
P	%	100.0	1	A
Y	%	86.0	1	A
Nd	%	61.0	1	A
Re	%	18.0	1	A
Mo	%	46.0	1	A
Fe	%	96.0	1	A
Ni	%	51.0	1	A
Zn	%	88.0	1	A
Ag	%	0	1	A
7.3 Solid Liquid Separation				
S/L separation method	-	CCD Circuit	4	A
Number of CCDs	-	3	5	A
Configuration	-	Series	4	A
Solids loading	$t/h.m^2$	1.0	5	A
Thickener U/F density	$\%^w/_w$	50.0	5	A
Wash water volumetric flow rate	m^3/h	95.0	6	A
7.4 Iron Reduction				
Iron reduction material	-	Raw, washed Fe	4	A

8 MOLYBDENUM ION EXCHANGE

	Units	Nominal	Ref	Rev
8.1 Molybdenum Ion Exchange				
Number of columns	-	3	4	A
Resin type	-	Chelating	5	A
Loading equivalent	eqv/L	1.0	4	A
Mo recovery	$\%^{w}/_{w}$	98.0	5	A
Stripping fluid	-	NH_4OH (4%)	5	A
Washing fluid	-	H_2O	5	A
Cycle duration (load, backup, strip)	h	8.0	4	A
Operating Temperature	°C	25	4	A

9 MOLYBDENUM RECOVERY

	Units	Nominal	Ref	Rev
9.1 Molybdenum Precipitation				
Recovery method	-	Precipitation	4	A
Precipitation product	-	$CaMoO_4$	5	A
Precipitation pH	-	4	8	A
Precipitation temperature	°C	25	8	A
Alkaline fluid for pH control	-	NH_4OH (60%$^w/_w$)	5	A
9.2 Solid Liquid Separation				
Solid/liquid separation method 1	-	Thickener	5	A
Solids loading	t/h.m^2	1.0	4	A
Thickener U/F density	%$^w/_w$	50.0	5	A
Solid/liquid separation method 2	-	Centrifuge	5	A
Product solids density	%$^w/_w$	90.0	5	A
Solid/liquid separation method 3	-	Dryer	5	A
Dryer temperature	°C	100	5	A
Product solids density	%$^w/_w$	99.0	5	A

10 VANADIUM AND URANIUM ION EXCHANGE

	Units	Nominal	Ref	Rev
10.1 Vanadium and Uranium Ion Exchange				
Number of columns	-	9	4	A
Resin type	-	Tertiary amine	5	A
Loading equivalent (V_2O_5)	eqv/L	1.0	4	A
Loading equivalent (UO_2)	eqv/L	1.0	4	A
V recovery	$\%^w/_w$	98.0	5	A
U recovery	$\%^w/_w$	98.0	5	A
Stripping fluid	-	H_2SO_4 (10%$^w/_w$)	4	A
Washing fluid	-	H_2O	4	A
Cycle duration (load, backup, strip)	h	8.0	4	A
Operating Temperature	°C	25	4	A

11 URANIUM RECOVERY

	Units	Nominal	Ref	Rev
11.1 Uranium Precipitation				
Recovery method	-	Precipitation	4	A
Precipitation product	-	$UO_4.2H_2O$	5	A
Precipitation pH	-	4	5	A
Precipitation temperature	°C	25	5	A
Alkaline fluid for pH control	-	NH_4OH (60%$^w/_w$)	4	A
Precipitation reagent	-	H_2O_2 (30%$^w/_w$)	5	A
11.2 Solid Liquid Separation				
Solid/liquid separation method 1	-	Thickener	5	A
Solids loading	t/h.m^2	1.0	4	A
Thickener U/F density	%$^w/_w$	50.0	4	A
Solid/liquid separation method 2	-	Centrifuge	5	A
Product solids density	%$^w/_w$	90.0	4	A
Solid/liquid separation method 3	-	Calciner	5	A
Dryer temperature	°C	700	5	A
Product solids density	%$^w/_w$	99.0	4	A
Calciner product	-	U_3O_8	4	A

12 VANADIUM RECOVERY

	Units	Nominal	Ref	Rev
12.1 Vanadium Precipitation				
Recovery method	-	Precipitation	4	A
Precipitation product	-	NH_4VO_3	5	A
Precipitation pH	-	8	5	A
Precipitation temperature	°C	25	5	A
Alkaline fluid for pH control	-	NH_4OH (60%$^w/_w$)	4	A
12.2 Solid Liquid Separation				
Solid/liquid separation method 1	-	Thickener	5	A
Solids loading	$t/h.m^2$	1.0	4	A
Thickener U/F density	%$^w/_w$	50.0	4	A
Solid/liquid separation method 2	-	Centrifuge	5	A
Product solids density	%$^w/_w$	90.0	5	A
Solid/liquid separation method 3	-	Dryer	5	A
Dryer temperature	°C	100	5	A
Product solids density	%$^w/_w$	99.0	4	A

13 PHOSPHORIC ACID SOLVENT EXTRACTION

	Units	Nominal	Ref	Rev
13.1 Phosphoric Acid SX Circuit				
Extractant				
Name	-	TBP	5	A
Concentration	$\%^v/_v$	75.0	4	A
Diluent				
Name	-	ShellSol	5	A
Concentration	$\%^v/_v$	25.0	4	A
Entrainment Losses				
Organic losses	%	10.0	8	A
Plant Configuration				
Number of trains	-	1	4	A
Number of extract stages	per train	7	4	A
Number of wash stages	per train	3	4	A
Number of strip stages	per train	5	4	A
O:A mixing ratio	-	1 : 1	5	A
Retention time	min/ stage	2.0	4	A
Overall extraction	%	99.0	4	A
Settling retention time	min	2.0	5	A
Stripping agent	-	H_2O	4	A
Scrubbing agent	-	H_2O	4	A
Organic product compounds	-	Phosphoric acid (18$\%^w/_w$)	4	A
Raffinate product compounds	-	Rare earths	4	A
	-	Manganese, nickel & zinc	4	A

14 PHOSPHORIC ACID UPGRADE

	Units	Nominal	Ref	Rev
14.1 Phosphoric Acid Concentration				
Phosphoric acid feed concentration	$\%^{w}/_{w}$	24.3	6	A
Equipment type	-	Evaporator	4	A
Heat source	-	Steam/Diesel/Electric - TBD	5	A
Evaporator temperature	°C	>110	4	A
Phosphoric acid product concentration	$\%^{w}/_{w}$	83.7	4	A
Packaging container details	-	20.0 m^3 shipping container tank	1	A
Packaging container volume	m^3	20.0	5	A

15 RARE EARTH RECOVERY

	Units	Nominal	Ref	Rev
15.1 Rare Earth Precipitation				
Precipitation pH	-	6	8	A
pH control	-	NH_4OH (60%$^w/_w$)	4	A
Precipitation temperature	°C	25	8	A
Rare earth recovery	%	100	4	A
15.2 Solid Liquid Separation				
Solid/liquid separation method 1	-	Pressure filter	4	A
Product solids density	%$^w/_w$	90.0	5	A
Solid/liquid separation method 2	-	Dryer	4	A
Dryer temperature	°C	>110	5	A
Product solids density	%$^w/_w$	99.0	4	A
Product description	-	Rare earth oxides	4	A

16 MANGANESE, NICKEL & ZINC SOLVENT EXTRACTION

	Units	Nominal	Ref	Rev
16.1 Manganese, Nickel & Zinc SX Circuit				
Extractant				
Organic Name	-	D2EHPA	5	A
Concentration	$\%^{v}/_{v}$	30.0	4	A
Diluent				
Diluent Name		ShellSol	5	A
Concentration	$\%^{v}/_{v}$	70.0	4	A
Entrainment Losses				
Organic losses	%	10.0	8	A
Plant Configuration				
Number of trains		1	4	A
Number of extract stages	per train	7	4	A
Number of wash stages	per train	3	4	A
Number of strip stages	per train	5	4	A
O:A mixing ratio		1:1	5	A
Retention time	min/ stage	2.0	4	A
Overall extraction	%	98.0	4	A
Settling retention time	min	2.0	5	A
Stripping agent	-	Sulphuric acid (10%$^{w}/_{w}$)	4	A
Scrubbing agent	-	H_2O	4	A
Organic product compounds	-	Waste water treatment	4	A
Raffinate product compounds	-	Manganese, nickel & zinc	4	A

17 MANGANESE, NICKEL & ZINC RECOVERY

	Units	Nominal	Ref	Rev
17.1 Manganese, Nickel & Zinc Precipitation				
Precipitation pH	-	8	8	A
pH control	-	NH_4OH (60%$^w/_w$)	4	A
Precipitating agent	-	$(NH_4)_2CO_3$ (50%$^w/_w$)	8	A
Precipitation temperature	°C	25	4	A
Mn, Ni, Zn recovery	%	98.0	4	A
17.2 Solid/Liquid Separation				
Solid/liquid separation method 1	-	Centrifuge	5	A
Product solids density	%$^w/_w$	90.0	4	A
Solid/liquid separation method 2	-	Dryer	4	A
Dryer temperature	°C	>110	4	A
Product solids density	%$^w/_w$	99.0	4	A
Product description	-	Manganese, nickel & zinc mixed carbonate	4	A

18 ACID PRODUCTION PLANT

	Units	Nominal	Ref	Rev
18.1 Sulphuric Acid Production Plant				
Production Method	-	Solid sulphur combustion	5	A
Conversion efficiency	%	91.0	8	A
Sulphur consumption*	t/d	637.0	6	A
Sulphuric acid production	t/d	1,947.0	6	A
* Note: potential exists to reduce sulphur consumption by utilising SO_2 off gas from ferric production to produce 4.0 t/h of sulfuric acid.				

19 WASTE WATER TREATMENT

	Units	Nominal	Ref	Rev
Target pH	-	8.0	5	A
19.1 Ferric Leach Wash Residue Liquid				
pH	-	8.0	5	A
Neutralising agent	-	Hydrated Lime	4	A
19.2 Molybdenum Recovery Solid/Liquid Separation Liquid				
pH	-	8.0	5	A
Neutralising agent	-	Hydrated Lime	4	A
19.3 Vanadium Precipitation Solid/Liquid Separation Liquid				
pH	-	8.0	5	A
Neutralising agent	-	Hydrated Lime	4	A
19.4 Mn, Ni & Zn Solvent Extraction Waste Liquid				
pH	-	8.0	5	A
Neutralising agent	-	Hydrated Lime	4	A
19.5 Mn, Ni & Zn Precipitation Solid/Liquid Separation Liquid				
pH	-	8.0	5	A
Neutralising agent	-	Hydrated Lime	4	A

20 TAILINGS NEUTRALISATION AND/OR PRECIPITATION

	Units	Nominal	Ref	Rev
20.1 Process Information				
Number of Stages	-	TBD	5	A
Design Residence Time	h / stage	1.0	4	A
Neutralising Source	-	Hydrated Lime	5	A
Target Final pH	-	8.0	5	A
20.2 Process Chemistry				
Precipitation	%	100	4	A

21 REAGENTS

	Units	Nominal	Ref	Rev
21.1 Acetic Acid (Glacial)				
Supply Form	-	Liquid	8	A
Supply Packaging	-	1 t Bulki Box	8	A
Supply Concentration	$\%^{w}/_{w}$	99.85	8	A
pH	-	2.4	8	A
Bulk Density	t / m^3	1.05	8	A
Storage Concentration	$\%^{w}/_{w}$	99.85	8	A
Addition Concentration	$\%^{w}/_{w}$	99.85	8	A
Viscosity at 25°C	cP	1.22	8	A
Flash Point	°C	40.0	8	A
21.2 Ammonium Carbonate				
Supply Form	-	Solids Powder	8	A
Supply Packaging	-	1 t bag	8	A
Supply Concentration	$\%^{w}/_{w}$	99.99	8	A
Bulk Density	t / m^3	1.50	8	A
Storage Concentration	$\%^{w}/_{w}$	99.99	-	A
Addition Concentration	$\%^{w}/_{w}$	99.99	-	A
Viscosity at 25°C	cP	Not applicable	-	A
Flash Point	°C	Not applicable	-	A
21.3 Ammonium Hydroxide (60%$^{w}/_{w}$)				
Supply Form	-	Liquid	8	A
Supply Packaging	-	1 t Bulki Box	8	A
Supply Concentration	$\%^{w}/_{w}$	60.00	8	A
pH	-	11.7	8	A
Bulk Density	t / m^3	0.92	8	A
Storage Concentration	$\%^{w}/_{w}$	60.00	8	A
Addition Concentration	$\%^{w}/_{w}$	60.00	8	A
Viscosity at 25°C	cP	1.3	8	A
Flash Point	°C	Not applicable	-	A
21.4 Diluent (ShellSol)				
Supply Form	-	Liquid	8	A
Supply Packaging	-	1 t Bulki Box	8	A
Supply Concentration	$\%^{w}/_{w}$	100.0	8	A
Bulk Density	t / m^3	0.75	8	A
Storage Concentration	$\%^{w}/_{w}$	100.0	8	A
Addition Concentration	$\%^{w}/_{w}$	100.0	8	A
Viscosity at 25°C	cP	0.91	8	A

	Units	Nominal	Ref	Rev
Flash Point	°C	27	8	A
21.5 Ferric Sulphate (60%$^w/_w$)				
Supply Form	-	Liquid	8	A
Supply Packaging	-	1 t Bulki Box	8	A
Supply Concentration	%$^w/_w$	60.00	8	A
Bulk Density	t / m^3	1.69	8	A
Storage Concentration	%$^w/_w$	60.00	8	A
Addition Concentration	%$^w/_w$	60.00	8	A
Viscosity at 25°C	cP	20	8	A
Flash Point	°C	Not applicable	8	A
21.6 Hydrated Lime (90%$^w/_w$)				
Supply Form	-	Solids powder	8	A
Supply Packaging	-	1 t bag	8	A
Supply Concentration	%$^w/_w$	90.0 – 95.0	8	A
pH	-	12	8	A
Bulk Density	t / m^3	0.8	8	A
Storage Concentration	%$^w/_w$	40.0	8	A
Addition Concentration	%$^w/_w$	90.0	8	A
Viscosity at 25°C	cP	20	8	A
Flash Point	°C	Not applicable	8	A
21.7 Hydrogen Peroxide (35%$^w/_w$)				
Supply Form	-	Liquid	8	A
Supply Packaging	-	1 t Bulki Box	8	A
Supply Concentration	%$^w/_w$	35.0	8	A
Bulk Density	t / m^3	1.11	8	A
Storage Concentration	%$^w/_w$	35.0	8	A
Addition Concentration	%$^w/_w$	35.0	4	A
Viscosity at 25°C	cP	1.1	8	A
Flash Point	°C	Not applicable	8	A
21.8 Pyrite				
Supply Form	-	Solids powder	8	A
Supply Packaging	-	1 t bag	8	A
Supply Concentration	%$^w/_w$	100.0	8	A
Bulk Density	t / m^3	5.0	8	A
Storage Concentration	%$^w/_w$	100.0	8	A
Addition Concentration	%$^w/_w$	100.0	8	A
Viscosity at 25°C	cP	Not applicable	8	A
Flash Point	°C	Not applicable	8	A

	Units	Nominal	Ref	Rev
21.9 Pyrolusite				
Supply Form	-	Solids powder	8	A
Supply Packaging	-	1 t bag	8	A
Supply Concentration	$\%^w/_w$	100.0	8	A
Bulk Density	t / m^3	5.0	8	A
Storage Concentration	$\%^w/_w$	100.0	8	A
Addition Concentration	$\%^w/_w$	100.0	8	A
Viscosity at 25°C	cP	Not applicable	8	A
Flash Point	°C	Not applicable	8	A
21.10 Organic Solvent - TBP				
Supply Form	-	Liquid	8	A
Supply Packaging	-	1 t Bulki Box	8	A
Supply Concentration	$\%^w/_w$	100.0	8	A
Bulk Density	t / m^3	0.98	8	A
Storage Concentration	$\%^w/_w$	100.0	8	A
Addition Concentration	$\%^w/_w$	100.0	8	A
Viscosity at 25°C	cP	3.80	8	A
Flash Point	°C	120	8	A
21.11 Organic Solvent – D2EHPA				
Supply Form	-	Liquid	8	A
Supply Packaging	-	1 t Bulki Box	8	A
Supply Concentration	$\%^w/_w$	>95%	8	A
Bulk Density	t / m^3	0.97	8	A
Storage Concentration	$\%^w/_w$	>95%	8	A
Addition Concentration	$\%^w/_w$	>95%	8	A
Viscosity at 25°C	cP	42	8	A
Flash Point	°C	196	8	A
21.12 Sulphur				
Supply Form	-	Solids powder	8	A
Supply Packaging	-	1 t bag	8	A
Supply Concentration	$\%^w/_w$	99.5	8	A
Bulk Density	t / m^3	2.05	8	A
Flash Point	°C	168	8	A
21.13 Sulphuric Acid				
Supply Form	-	Liquid – produced on site	8	A
Supply Packaging	-	Not applicable – stored on site	8	A
Supply Concentration	$\%^w/_w$	98%	8	A
Bulk Density	t / m^3	1.84	8	A
Storage Concentration	$\%^w/_w$	98%	8	A

	Units	Nominal	Ref	Rev
Addition Concentration	$\%^{w}/_{w}$	98% & 10%	8	A
Viscosity at 25°C	cP	26.7	8	A
Flash Point	°C	Not applicable	8	A
21.14 Washed Scrap Iron				
Supply Form	-	Solids	8	A
Supply Packaging	-	Not required	8	A
Bulk Density	t / m^3	5.2	8	A

22 SERVICES

	Units	Nominal	Ref	Rev
22.1 Water				
Raw Water				
Supply Water Quality	-	As supplied	4	A
Filtered Water				
Filter Supply Water Quality	-	Raw water	4	A
Gland Water				
Supply Water Quality	-	Filtered	4	A
Potable Water				
Supply Water Quality	-	Filtered	4	A
Product Water Quality Standard	-	Drinking Water Guidelines	4	A
Fire Water				
Supply Water Quality	-	Raw water	4	A
22.2 Power				
Plant Power Source	-	Grid electricity	1	A