UPDATE REPORT
ON THE APPIA ENERGY CORP. URANIUM-RARE EARTH PROPERTY, ELLIOT LAKE DISTRICT, NORTH-CENTRAL ONTARIO, CANADA

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1. SUMMARY

In a letter dated 1 February, 2007 **Canada Enerco Corporation** ("**CEC**"), of Toronto, Ontario, Canada retained **Watts, Griffis and McOuat Limited** ("**WGM**") to undertake an evaluation of a group of uranium properties (the "**Property**") located in Beange, Buckles, Bouck, Gunterman and Joubin Townships, near the town of Elliot Lake in north-central Ontario.

The above-mentioned townships cover much of the famous Elliot Lake mining camp which produced uranium during the period 1955 through 1996. By the time the last mines closed in the mid-1990s, a total of 362 million pounds of uranium oxide (313.5 Mlbs uranium metal) was produced from 13 underground mines having an average grade of approximately 0.106% U_3O_8 (898 ppm U or 2.12 lbs U_3O_8 per short ton). Many of the mines closed in the early 1980s at a time when uranium prices were falling due to rapidly accumulating uranium stockpiles in many countries, including Canada. At the same time, energy prices were rising, inflation rates were in excess of 10% and interest rates exceeded 15%. The mines closed due to simple economic pressures, and not as a result of resource depletion. The remaining uranium resources included several zones that had been discovered through drilling, but never developed.

The area is underlain by the Huronian Supergroup, a southward-thickening, mainly clastic succession which is well exposed north of Lake Huron. The uranium-rare earth deposits are hosted in quartz-pebble conglomerates that are located in the lower part of the Matinenda Formation, the basal unit of the Elliot Lake Group, and only a short distance above crystalline Archean basement rocks. Uranium and rare earth element ("**REE**") mineralization is stratabound, showing good consistency in grade and thickness over wide areas, both along strike and down dip. The uranium mineralization is predominantly uraninite, coffinite and uranothorite, most of which are readily leachable, and this facilitated high recoveries, typically 95%, in the operating mines. These minerals carry economically important amounts of yttrium which was historically co-produced at the uranium mines. The balance of the yttrium and the other REEs are dominantly contained in monazite.

In the late 1960s and early 1970s, Kerr McGee Corporation ("**Kerr McGee**"), a major US uranium producer, acquired a large group of mining claims covering key portions of the Elliot Lake basin. Ker McGee completed diamond drill holes in areas not formerly tested, or where earlier holes had failed to reach the uranium-bearing horizons. Following this drilling in the Banana Lake area, Mr. Doug Sprague, P.Eng., Chief Geologist for Rio Algom Ltd.

("**RIO**") estimated the potential resource in the Banana Lake area at 176 million short tons with a grade of 0.76 lbs U_3O_8/ton over an average thickness of 5.36 m (17.6 feet). REE contents were not determined at the time because the market for these metals was quite limited. Sprague's estimate was based on a limited number of widely spaced holes, and the fact that the uranium-bearing horizons demonstrated remarkable lateral and down-dip continuity over a very large area. Subsequent historical estimates thought to be authored by Sprague and shown on a RIO map (Rio Algom, 1979), were based on mine data as well as a series of deep drill holes completed by Kerr McGee and other exploration companies in widely separated areas of the Property. RIO's estimates increased the total remaining uranium resource to approximately 200 million pounds of U_3O_8. These historical resources, located in five separate zones down-dip from past-producing mines, are summarized as:

Zone	Quantity (tons)	Grade (lbs U_3O_8/ton)	Contained U_3O_8 (lbs)
Teasdale Lake	17,458,200	1.206	20,787,200
Gemico Block #3	42,800,000	0.38	16,264,000
Gemico Block #10	20,700,000	0.75	15,525,000
Banana Lake Zone	175,800,000	0.76	133,608,000
Canuc Zone	7,000,000	1.86	13,020,000
Total	**263,758,200**	**0.76**	**199,204,200**

The foregoing historical resources were not estimated in accordance with definitions and practices established for the estimation of Mineral Resources and Mineral Reserves by the Canadian Institute of Mining and Metallurgy ("**CIM**"). As such, the historical resources are not compliant with Canada's security rule National Instrument 43-101 ("**NI 43-101**"), and are unreliable for investment decisions. Neither Appia nor its Qualified Persons have done sufficient work to classify the historical resources as current mineral resources under current mineral resource terminology and are not treating the historical resources as current mineral resources. Nevertheless, most of the historical resources were estimated by mining companies active in the Elliot Lake camp using assumptions, methods and practices that were accepted at the time, and based on corroborative mining experience.

The purpose of WGM's initial review was to document the historical uranium resource estimates pertaining to the CEC claims, as well as to conceptually explore the conditions required in order for the resources to be economically viable. In an agreement dated 1 Nov., 2007, Appia Energy Corp. ("**Appia**") optioned the Property from CEC and WGM's contractual responsibilities were modified accordingly. Since that time, WGM has prepared several up-dates to the Mineral Resources. The current Mineral Resources, as stated herein and has been prepared in accordance with the terms of an agreement between Appia and WGM dated 31 October, 2012.

In accordance with WGM's recommendations, Appia carried out two programs of diamond drilling, the first during 2007-08 which confirmed the presence of uranium resources in the areas tested and provided the basis for initial resource estimates in the Teasdale Zone and the Banana Lake Zone, and the second during 2012 which focused on the Teasdale Zone and supported the current up-dated Mineral Resources estimates for that zone. Through the use of twinned, wedged and step-out drill holes, WGM and Appia have confirmed the borehole data on which the Banana Lake and Teasdale Lake estimates were based. WGM prepared its estimates for mineralized zones belonging to the Banana Lake and Teasdale Zones on the basis of sufficient data to show continuity of geology and grades. The current Mineral Resources estimates are contained in Workman and Breede (2011) and are summarized herein. The Banana Lake and Teasdale Zones are widely separated and are unlikely to be mined from a single point of entry (ramp or shaft).

The Banana Lake Mineral Resources estimate (Table 1) was prepared from a block model using a 0.6 lb U_3O_8/ton cut-off grade, a minimum vertical thickness of 5 m, and based on the assumption that material from this deposit would be refined in a central milling facility that would accommodate Appia's operations and, potentially, neighbouring mining operations in the Elliot Lake camp, thus significantly reducing capital and operating costs (Workman and Breede). The increased minimum thickness was imposed by WGM to provide a basis for the use of larger underground equipment as a cost-reduction strategy, however this restriction had little impact on the contained resources.

Table 1
Summary of Banana Lake Zone Mineral Resource Estimate
(using 0.6 lb U_3O_8 / ton Cut-Off Grade)

Category	Tons ('000)	S.G. (tons/m^3)	lbs U_3O_8/ton	Total lbs U_3O_8 ('000)
Inferred Resources	30,315	3.14	0.912	27,638

Notes:
1. Effective, 1 April, 2011
2. Mineral Resources which are not Mineral Reserves do not have demonstrated economic viability. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.
3. The quantity and grade of reported Inferred Resources in this estimation are uncertain in nature and there has been insufficient exploration to define these Inferred Resources as an Indicated or Measured Mineral Resource and it is uncertain if further exploration will result in upgrading them to an Indicated or Measured Mineral Resource category.
4. The Mineral Resources were estimated using the Canadian Institute of Mining, Metallurgy and Petroleum standards on Mineral Resources and Reserves, Definitions and Guidelines prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council December 11, 2005. These definitions are provided in Section 17.1 of this report.
5. S.G. of 2.85 tonnes/m^3 (or 3.14 tons/m^3) was used.
6. Indicated amounts may not precisely sum due to rounding.

The estimate of the Mineral Resources in the Teasdale Zone (Table 2) was initially reported in Workman and Breede (2011) and prepared from a polygonal model using a geological cut-off and a minimum bed thickness of 2.44 m (8 ft.) which takes into consideration the continuity of grade within the various mineralized beds and historical mining practices. The mineralized zone was geologically constrained by the well defined markers provided by the upper surface of the highest mineralized bed and the lower surface of the basal bed. The resources were reported for each of the three geological units that comprise the mineralized zone: Upper Reef ("**UR**"), Intermediate Quartzite ("**IQ**") and Lower Reef ("**LR**"). As a result of the inclusion of the UR to incorporate its significant REE content as well as the IQ, neither of which were mined historically, all drill hole intersections substantially exceeded the minimum thickness. No grade cut-off or high capping was used for this estimate as the grades were themselves quite robust and the utilization of a cut-off grade would require complex economic modelling of individual metals that is not required at this time.

This report encompasses the results of the most recent drilling on the Teasdale Zone and an up-dated Mineral Resources estimate for that zone. The report also incorporates the results of a recently completed metallurgical testing program that has identified beneficiation as a key step in the processing of Teasdale mineralization. The current Mineral Resources for the Teasdale Zone are stated as follows in Table 2 and as detailed in Table 3 for rare earth metals.

Table 2
Summary of Teasdale Zone Uranium and Rare Earth Mineral Resource Estimate

Zone	Tonnes ('000)	Tons ('000)	TREE (lbs/ton)	U_3O_8 (lbs/ton)	Average Thickness (m)	Contained TREE ('000 lbs)	Contained U_3O_8 ('000 lbs)
INDICATED RESOURCES							
UR	6,733	7,422	4.20	0.484	4.61	31,199	3,593
IQ	3,006	3,314	1.98	0.259	2.27	6,578	0.857
LR	3,355	3,699	2.68	0.958	2.60	9,912	3,544
Total	**13,095**	**14,435**	**3.30**	**0.554**	**9.48**	**47,689**	**7,995**
INFERRED RESOURCES							
UR	18,326	20,201	3.87	0.421	4.33	78,080	8,498
IQ	10,209	11,254	1.64	0.184	2.78	18,464	2,070
LR	9,972	10,992	3.33	0.869	2.71	36,631	9,564
Total	**38,507**	**42,447**	**3.14**	**0.474**	**9.82**	**133,175**	**20,115**

These resources are valid as of 30 July, 2013. Other qualifying notes for Mineral Resources are contained in Tables 1 and 3.

The individual REEs included in the TREE mineral resources are reported as follows in Table 3. The units are in PPM for comparison purposes (500 ppm = 1 lb/ton).

Table 3
Individual REE Resource Grade Composition Summary

Zone	Light REE (ppm)						**Heavy REE (ppm)**									
	La	Ce	Pr	Nd	Sm	Eu	**Gd**	**Tb**	**Dy**	**Ho**	**Er**	**Tm**	**Yb**	**Lu**	**Hf**	**Y**
INDICATED RESOURCES																
UR	540	951	93.9	313	51.7	1.9	32.8	3.9	17.2	2.7	7.0	0.9	5.5	0.8	6.8	72.9
IQ	256	452	44.9	148	24.4	1.0	14.7	1.8	7.7	1.2	3.1	0.4	2.5	0.4	3.6	30.6
LR	332	596	59.4	201	35.1	1.7	23.2	3.0	14.2	2.3	5.9	0.8	4.5	0.6	3.3	58.1
Total	**422**	**745**	**73.8**	**247**	**41.1**	**1.7**	**26.2**	**3.2**	**14.3**	**2.3**	**5.8**	**0.8**	**4.6**	**0.7**	**5.2**	**59.4**
INFERRED RESOURCES																
UR	498	876	85.9	285	47.2	1.8	29.3	3.5	15.9	2.5	6.5	0.9	5.3	0.8	6.8	67.9
IQ	213	374	37.0	122	20.0	0.8	12.3	1.4	6.4	1.0	2.6	0.4	2.2	0.3	3.3	26.5
LR	417	747	73.9	249	43.4	1.9	28.5	3.6	16.4	2.6	6.6	0.9	5.2	0.7	4.5	66.4
Total	**401**	**709**	**69.9**	**232**	**39.0**	**1.6**	**24.6**	**3.0**	**13.5**	**2.1**	**5.5**	**0.7**	**4.4**	**0.6**	**5.3**	**56.5**

Notes:
1. Mineral Resources effective 30 July, 2013
2. Mineral Resources are estimated at a cut-off value of $100 per tonne, using a uranium price of US$70/lb U_3O_8, a TREE price of $78/kg, and a C$:US$ exchange rate of 1:0.9.
3. Mineral Resources which are not Mineral Reserves do not have demonstrated economic viability. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.
4. The quantity and grade of reported Inferred Resources in this estimation are uncertain in nature and there has been insufficient exploration to define these Inferred Resources as an Indicated or Measured Mineral Resource and it is uncertain if further exploration will result in upgrading them to an Indicated or Measured Mineral Resource category.
5. The Mineral Resources were estimated using the Canadian Institute of Mining, Metallurgy and Petroleum standards on Mineral Resources and Reserves, Definitions and Guidelines prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council December 11, 2005. These definitions are provided in Section 17.1 of this report.
6. S.G. of 2.85 tonnes/m^3 (or 3.14 tons/m^3) was used.
7. Indicated amounts may not precisely sum due to rounding.

As is clear from the foregoing estimate, the total rare metal content of the Teasdale Zone is approximately 6 times the uranium content if the resource is predominantly constrained by geology and is not constrained in respect to a uranium cut-off grade. While this approach results in the inclusion of lower grading uranium mineralization, it does incorporate large volumes of economically interesting REE mineralization.

Preliminary testwork and analysis carried out at the SGS Canada facility in Lakefield, Ontario indicates that a flotation-concentrate acid bake process route combined with a tailings leach can result in a high extraction of U (~90%) and most REEs (80-90%) from Appia’s Teasdale uranium-rare earth deposit. It is believed that planned additional testwork and data analysis will substantiate these data and probably lead to reduced overall processing costs and improved recovery.

WGM carried out site visits to the Elliot Lake property during 2007 and 2009 to review the geology, to locate historical drill hole collars, to examine and check sample drill core and to meet with company representatives concerning the project's accomplishments and future plans. WGM also met with officials of two key Ontario ministries: the Ministry of Northern Development and Mines and the Ministry of the Environment. These site visits and the work completed during each are described in Appia's more recent technical report that was authored by WGM in compliance with NI 43-101 (Workman and Breede, 2011). Although the most recent drilling was carried out under the close scrutiny of WGM and under the supervision of QP Alan MacEachern, P.Geo., WGM revisited the project site in July, 2013 for discussions with Mr. MacEachern and visits to a selection of drill sites for confirmation purposes. All work was found to have been completed in accordance with WGM's instructions and in accordance with industry best practices standards.

The commercial viability of REE mineralization was previously demonstrated by the historical recovery of yttrium as a by-product of uranium production at the Elliot Lake mines. These operations proved that separate facilities were not required to leach the REEs, and that once in solution, yttrium could be easily recovered. However the mine operators ignored the other REEs because the market was adequately served by deposits elsewhere. As noted in Workman and Breede (2011), Appia's drilling provided the opportunity to analyse the drill core for uranium and REEs and thereby estimate the size and potential of these resources present in that part of the Teasdale Zone drilled by Appia. To WGM's knowledge, the REE potential had never been investigated in the past. A significant number of historical drill holes completed by Conecho Corp. are known to have economically interesting LR intersections well beyond the limits of the area drilled by Appia at this time. The size of the Teasdale Zone is therefore likely to be increased with additional drilling which will test all horizons for both U and REEs.

At present, Appia plans to produce a high-value REE-uranium concentrate through beneficiation of the ore prior to leaching. Appia's assay data indicates that the value of the project vests mainly in uranium and some of the REEs present, predominantly in cerium, lanthanum, neodymium and yttrium which account for 86.3% of the total REEs present. Appia's initial review suggests that the project economics hinge on a higher uranium price than is present at this time, potentially as high as \$75/lb U_3O_8, however current market forecasts indicate that this price should be achievable given long-term supply-demand fundamentals. This price model has recently been used by Pele Mountain Resources Inc. ("**Pele**"), a junior exploration company that is exploring its Eco Ridge uranium project which is an "Adjacent Property" in the context of NI 43-101.

Pele provided an up-dated Mineral Resources estimate in a report publicly available on the SEDAR website dated 5 April, 2011 in which Indicated Resources of 14.31 Mt were reported grading 0.048% U_3O_8 (0.96 lbs U_3O_8 per ton) and 0.164% total rare earth elements ("**REE**"s) or 3.28 lbs/ton with additional Inferred Resources of 33.12 Mt grading 0.043% U_3O_8 (0.86 lbs U_3O_8 per ton) and 0.132% total REEs or 2.64 lbs/ton (Ciuculescu, 2011). The total contained metal was 15.2 million pounds of U_3O_8 and 51.9 Mlbs of REEs in the Indicated category and 31.4 Mlbs of U_3O_8 and 96.4 Mlbs of REEs in the Inferred category. The resources were based on a cut-off grade of 0.028% U_3O_8 and a long term uranium price of $60 per pound of uranium oxide (the current price is stable at $68). In July, 2011, Pele announced the results of a new Preliminary Economic Assessment ("**PEA**") for the Eco Ridge Project, including the following key findings based on a 9,400-tonne per day operation with life-of-mine production of 10.7 Mlbs of total rare earth oxides (REOs) and 24.9 Mlbs of U_3O_8 over a 14-year mine life:

- cumulative operating cash flow of US$1.72-billion
- cumulative pre-tax cash flow of US$1.31-billion
- positive NPV of $533 million (at a 10% discount rate)
- internal rate of return (IRR) of 47 percent (47%)
- operating cash cost of US $16 per pound U_3O_8, net of REO credits
- start-up capital costs of US $212 million and sustaining capital costs of US $195 million.

WGM and Appia assessed the Eco Ridge results and it believes that the foregoing costs offer guidance in respect to exploiting the Teasdale Zone resource which may potentially be much larger and higher grading. Historical production information for the Elliot Lake uranium mines does not include rare earth metal data other than some yttrium co-production data that is not specific to individual mines. Although it is likely that the major producers estimated the global rare metal content of the uranium ores, none of this information seems to be is in the public domain. As a result of the historical core being unavailable for reanalysis, Appia could rely only on its own drilling and REE assay data for resource estimation. WGM found that within the zone occupied by the uranium-bearing "reefs", REE mineralization was far more prevalent and blanketed across all of the uranium-bearing horizons.

An understanding of the dynamics of the uranium and REE markets is vital in opining on the future conditions under which the Appia project might be developed. Uranium prices have been affected on many occasions by the anti-nuclear sentiment that flared in the shadow of high-profile accidents such as Three Mile Island in the United States, Chernobyl in the Ukraine and most recently Fukishima in Japan. Media hype and political hyperbole did great, though mostly short-term, harm to the nuclear industry and uranium prices fell accordingly. The REE markets during the last 10 years have been as politicized as the uranium market, although for different reasons largely due to China's overwhelming control of REE production. China has used its clout to influence foreign REE manufacturers to transfer technology to China in return for assured supplies. This has resulted in a flurry of new REE projects and the development of REE-replacement alternatives where possible.

At about the time Appia's Elliot Lake project was initiated, the Spot Market price on 6 June, 2007 was US$135 per lb of U_3O_8 versus approximately $40 at the time of writing this report in June, 2013, a price retreat of approximately 70%. The less volatile Term Market price at that time was $95 versus $57 at this date, a retreat of 40%. However, it important to note that the Spot Price is the discount price and not the price at which most uranium is delivered to energy utilities. As of the date of this report, the Spot and Term prices are still recovering from the effects of the Fukushima Daiichi tsunami disaster in 2011 which was used by the media, anti-nuclear activists, and opposition governments to cast unjustifiable dispersions on the nuclear energy industry. Even Japan, having understandable concerns about reactor safety in an area with clear geohazards concerns, has re-stated its support of the industry and applications are being studied for the restart of 10 reactors idled since early 2011.

Utilities are intermittently buying in the Spot Market to supplement nuclear fuel supplies stockpiled at reactors and to satisfy the need for initial cores at new reactor sites. Substantial new electrical capacity development is going ahead at various nuclear sites and continuing upwards pressure on prices seems certain. The US-Russian HEU Agreement, officially named the "Megatons to Megawatts Program", terminates at the end of 2013. Under this agreement, 500 tonnes of highly enriched (weapons grade) uranium ("**HEU**") from approximately 20,000 Russian nuclear warheads is being down-blended to produce commercial-grade, low-enriched uranium ("**LEU**") for use in nuclear power plants. The LEU is shipped to the US where it is used as nuclear power plant fuel to generate electricity. Initiated in 1993, the 20-year program first shipped LEU to the US in June 1995. At this time the warhead-derived LEU from Russia provides over 10% of the electricity requirements of the US. No plans have been enunciated by either party to extend the agreement.

Despite anti-nuclear political rhetoric present in Europe, and until its recent reversal, Japan, many new nuclear energy projects are proceeding more or less on schedule. Both China and India, neither self-sufficient in fossil fuels, are building new reactors as quickly as possible due to rapidly escalating energy requirements – India's will triple within 25 years. At the same time, uranium mining projects are being slowed due to perceived social, political and capital risks. The permitting process for new uranium mining projects in greenfield areas is notorious for its delays, so sharp increases in production are unlikely. If the dynamics of the last uranium boom can be used as a model, lagging production will cause uranium prices to stabilize at a much higher price than has been seen recently. Evidence from various sources was reported by the World Nuclear Association to show that $40 was a marginal production price in 2010, illustrating that many producers were under stress at that time. RBC Capital Markets currently forecasts Term Market U_3O_8 prices rising to $75/lb in 2016, and an $80 price persisting beyond that to at least 2020. CIBC Metals & Minerals have forecast Term Market prices for U_3O_8 during 2014 and in the longer term of $65 and $70, respectively.

The search for REE deposits was energized by increasing demand for these metals in a wide range of 'high-tech' applications. China's decision to restrict REE exports to meet its own domestic needs initially caused price spikes for most REE products in the 3rd and 4th quarters of 2011, but at the end of 2011 and during 2012, prices fell in reflection of declining domestic consumption in the United States. Consumption generally decreased for cerium compounds used in automotive catalytic converters and in glass additives and glass-polishing compounds; REE chlorides used in the production of fluid-cracking catalysts for oil refining; REE compounds used in automotive catalytic converters and many other applications. Rare metals and their alloys used in armaments and base-metal alloys also saw price declines. Consumption was stable in lighter flints, permanent magnets, pyrophoric alloys and superalloys, but decreased for yttrium compounds. Demand remained stable for REEs in many other applications, especially in rechargeable batteries for electric and hybrid vehicles.

Molycorp's REE separation plant at Mountain Pass, California resumed operation in 2007 and operated throughout 2011 producing bastnäsite concentrates and other REE intermediates and refined products from mine stockpiles. Molycorp commenced its Project Phoenix in 2011, a plan to reopen mining operations and build new processing facilities. Mining operations were restarted during 2012. Molycorp also acquired Canadian REE technology company, Neo Material Technologies, a leading REE fabricator with global operations including facilities in China. The Mountain Pass processing plant commenced activity in the 1st quarter of 2013. US consumption decreased 25% in 2012 over 2011, partly due to substitution. MolyCorp reported that it sold 3,274 tonnes of REE product in the 1st Qtr, 2013 from its various global operations at an average sales price of US $44.71 per kg. The Mountain Pass REE product, of which approximately 47% is Ce, 32% is La, 21% is Nd and Pr and less than 5% is other REOs, was sold at US $23/kg. Although revenues were up 9% over 2012 levels, Molycorp posted a net loss of $0.33 per share due to soft REE prices.

Molycorp planned to ramp up to the facility's planned annual run rate of 19,050 tonnes of REO equivalent by mid-2013. It is forecast to develop profitability in 2015.

The considerable excitement that gripped rare earth metal explorers, a relatively recent phenomenon, forced governments to suddenly assess the strategic importance of REEs to key industrial applications. The Australian Government's review of national mineral activities for 2009 does <u>not</u> mention a single rare earth project even though several major discoveries were moving towards production. The Mount Weld Mine, owned by Lynas Corp., is now in production and the company's REO refinery in Malaysia is ramping up to an 11,000 tpa rate of REO production, to be doubled when its phase 2 program is commissioned later in 2013.

Figure 1 summarizes global REE production during the period from 1983 through 2012. Most of the growth on the production side of the industry, to the 133,000 tonnes produced in 2011 (110,000 t at this time), has resulted from increased output from China. However, both the USA and Australia returned to REE production during 2012, and hence any forecasts of future REE prices must incorporate the impact of alternative supplies in the marketplace.

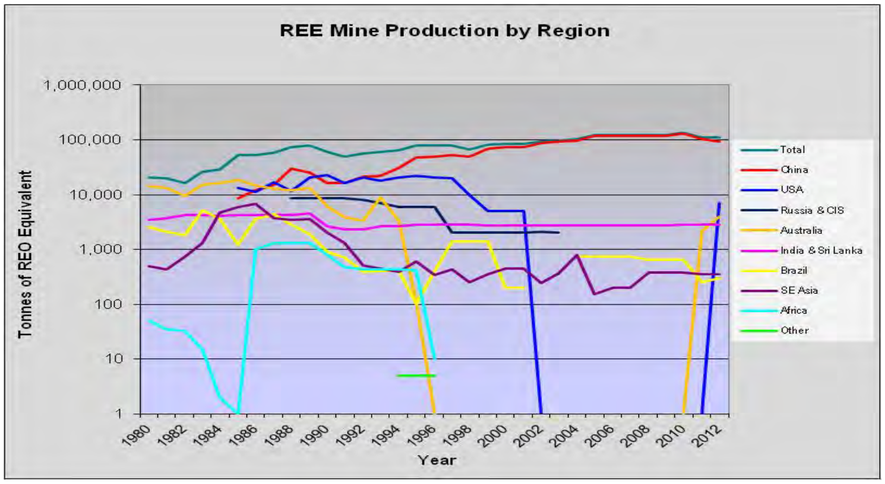

Figure 1: Graph showing global REE mine production. The return to production of mines in Australia and the USA is notable and must be taken into account in any forecast regarding future REE prices, especially the light REE metals.

Global exploration efforts to develop rare earths projects surged in 2011, and investment and interest increased dramatically. Economic assessments continued in the U.S.A. at Bear Lodge (Wyoming); Diamond Creek (Idaho); and Elk Creek (Nebraska); as well as in Canada at Hoidas Lake (Saskatchewan), Kipawa (Quebec) and Nechalacho (Thor Lake) in the Nunavut Territory. None of these deposits are thought to be near a production decision.

Economic assessments in other locations around the world include Dubbo Zirconia in New South Wales, Australia; Kangankunde in Malawi; Nolan's Bore Project in the Northern Territory, Australia, and at Steenkampskraal in South Africa. Nolan's Bore is similar to Mount Weld in respect to REE content.

The mines of Elliot Lake are the only deposits in Canada which have seen rare metal production. During the 1970s and 1980s, yttrium was a major by-product of uranium mining at both the Denison and the Rio Algom operations. The Canadian Minerals yearbook documents production. Although significant concentrations of rare earth metals were recognized, exceeding even that of yttrium, global prices for such metals at the time did not favour a commercial operation. This report represents an up-date of the previous WGM work to take into account the considerable unrealized value of rare earth metal mineralization present in the Elliot Lake deposits. Although no previous resource estimates for these metals in the Elliot Lake deposits have ever been made, new interest in REEs has put the spotlight on the area as a potential producer. In November 2009, Montoro Resources Inc. confirmed that it had intersected significant concentrations of rare earth oxides on its Serpent River property located 13 km east of Elliot Lake.

WGM believes the Appia project offers the following positive factors:

1) the project is located in a brownfields area that has been impacted by a long history of uranium and REE mining and production;
2) the previous mining has demonstrated that uranium tailings can be stabilized and managed such that the impacts are successfully mitigated without serious adverse consequences for the local environment;;
3) Appia's Teasdale Deposit is close to existing electrical and water supplies as well as uranium infrastructure (the Cameco uranium refinery situated west of Elliot Lake near the town of Blind River);
4) the mineralization is stratabound with excellent lateral grade and thickness continuity;
5) mine rock conditions were favourable for underground development and their engineering properties are well understood;
6) the average thickness of the deposit (9.7 m) is approximately 3 times greater than the average thickness mined in the past which should allow Appia commensurate reductions in mining costs;
7) the uranium and REE mineralization are interrelated and can be mined without changes to the mining plan;
8) the associated REE production is another potential revenue generator that could exceed the value of uranium production;

9) the metallurgy of Elliot Lake uranium and REE mineralization is well known in respect to milling characteristics as well as its leaching characteristics, including the application of bacteria leach technology to uranium;
10) historical evidence proves that REE recovery can utilize much of the same processing technology as that used for uranium production;
11) Appia bears no responsibility for potential environmental legacy issues in the future arising out of previous mining activities;
12) other companies delineating new uranium and REE resources in the area, such as Pele Mountain Resources, as well as companies such as Denison Mines and Rio Algom which continue to hold significant Mineral Resources, may present opportunities for joint mill ownership or toll milling;
13) proximity to the Panel Mine which offers the potential for shaft revitalization and underground openings for mine infrastructure as well as in-stope leaching and waste disposal;
14) the project is located in Ontario, Canada, in an area that has a long mining tradition and in an investor-friendly jurisdiction that supports flow-through financing;
15) a hard-working and knowledgeable local labour force lives in northern Ontario;
16) excellent year-round project access and close to infrastructure and supply centres;
17) the property is large, extending over an area adjacent to and down-dip of former mine workings; and,
18) the potential to develop a large scale thorium resource from Elliot Lake tailings is a potential future consideration since India and some other countries are now working on the engineering challenges of using thorium fuel.

Based on Appia's results to date, additional exploration is clearly warranted.

In Workman and Breede (2011), WGM previously identified a staged exploration program budgeted at $14.6 million that, over time, would minimize technical risk by building slowly from the established facts concerning the historical work. For planning purposes, three phases of drilling were proposed on each of the Teasdale and Banana Lake zones for budgetary and cash flow reasons. The drilling was divided between 15,405 m on the Teasdale Lake Zone and 17,600 m on the Banana Lake Zone. During 2012, Appia completed approximately half of the drilling proposed at a significantly lower cost that estimated. The savings are largely attributed to Appia avoiding the cost of a helicopter for moving the drilling rig between sites. During 2012-13, Appia also completed a detailed metallurgical testing program that provided positive outcomes for the use of beneficiation to process Elliot Lake ores, thereby reducing capital costs, as well as positive outcomes concerning the recovery of uranium and REEs.

As a first priority exploration task, WGM recommends that Appia continue its drilling to enlarge the Teasdale Zone in all directions. In completing this review, WGM assessed the results of the drilling to date and herein proposes that Appia complete an additional 14 drill holes totalling 7,750 m at a total estimated cost of $2,192,000. These costs are detailed in Table 35 in the Budget section of this report.

Although the amount of mineralization present in the Banana Lake Zone is substantial, its continuing exploration must be rated as a second priority exploration target at this time because WGM believes that the greatest benefit to Appia is in completing the Teasdale drilling and its economic assessment as a matter of the highest priority. Continued exploration drilling on the Banana Lake Zone must be contingent on funding, however exploration costs have declined as a result of the general conditions which have negatively impacted many gold and base metals projects in the Province of Ontario. Therefore WGM believes that some of the cost savings realized during the 2012 drilling program may be realized in the Banana Lake area despite the fact that the drilling is much deeper. These cost reductions have been applied to the recommended drilling for this zone. Whereas the previously budgeted drilling was estimated to cost $7.6 million for three phases totalling 17,600 m, this drilling might now be completed for as little as $4.96 million (Table 35).

WGM also recommends that Appia continue its metallurgical testing program to optimise the beneficiation and leaching flowsheet and to provide key information for a NI 43-101 compliant Preliminary Economic Assessment (“**PEA**”) in the near future. Based on the results of Pele’s PEA on its Eco Ridge deposit, WGM believes that the Teasdale PEA will provide ample justification for continuing investment in the Elliot Lake uranium and rare metal-bearing zones.

WGM has also recommended that Appia pursue a program of community engagement and information sessions to ensure that local residents and their representatives are aware of the company’s activities and the long term benefits to the community if uranium mining resumes at Elliot Lake.

2. INTRODUCTION

2.1 GENERAL

In a letter dated 1 February, 2007 Canada Enerco Corporation ("**CEC**"), of Toronto, Ontario, Canada retained Watts, Griffis and McOuat Limited ("**WGM**") to undertake an evaluation of a group of uranium properties located in Buckles, Bouck, Beange, Gunterman and Joubin Townships, near the town of Elliot Lake in north-central Ontario (Figure 2). The aforementioned townships enclose portions of the famous Elliot Lake Mining Camp which produced uranium during the period 1955 through 1996. Senior WGM Geologist and Vice-President, Al Workman, P.Geo. and Qualified Person ("**QP**") for the project, visited key Ontario ministry offices in Sault Ste. Marie and visited the Elliot Lake area on 15-16 May, 2007 and during 3-4 June, 2009. In a subsequent agreement dated 1 November, 2007, Appia Energy Corp. ("**Appia**") optioned the Property from CEC, and then requested that WGM up-date its previous report to encompass the results of recent diamond drilling carried out by Appia on the Property.

Between 18 November, 2007 and 12 March, 2008 Appia completed a Stage 1 exploration program comprising a total of 10 diamond drill holes using 2 drilling rigs (6 new holes and 4 wedged holes totalling 3,885.2 m). Exploration expenditures totalled approximately C$ 2,000,000. The drilling confirmed previously reported intersections and provided the basis for WGM to complete a NI 43-101 compliant resource estimate on the Banana Lake and Teasdale Lake zones. At this time, Appia was a private company and consequently the results were not reported publicly. During its revisit of the project site during June, 2009 WGM had discussions with Appia's QP, Mr. Alan MacEachern, formerly Chief Mine Geologist for Denison Mines Ltd. resulting in valuable insights into the local geology that are not available in published literature. During this site visit WGM confirmed that the reported exploration work had been completed, and re-examined key intervals of mineralized drill core.

From October to December 2008, Appia carried out a second program of diamond drilling on the Banana Lake Zone. This exploration entailed new step-out drilling within the resource area of the zone previously identified by Rio Algom based on widely separated Kerr McGee drill holes. Using the new data, WGM up-dated the previous NI 43-101 compliant resource estimate, the results of which showed a substantial increase in tonnage over WGM's initial estimate, as well as a 20% increase in grade over historical estimates.

The current WGM-Appia agreement is dated 1 October, 2009 and under this agreement WGM has provided on-going advisory services and up-dates to its earlier reports. These assignments for Appia have included geological modelling of the Banana Lake and Teasdale Zones drilled by Appia, and Mineral Resource estimates for each. All reports and estimates have been completed in compliance with Canadian securities rule National Instrument 43-101 ("**NI 43-101**").

WGM's report dated 18 July, 2011 included a report of Mineral Resources for the Appia Project. It was filed during November, 2012 and it provides details of historical uranium production in the Elliot Lake mining camp, historical resource estimates on Appia's claims, WGM's observations concerning the geology and Appia's exploration, as well as WGM's data confirmation procedures and evaluation of the project together with recommendations for additional exploration.

On-going discussions between WGM and Mr. MacEachern have been extremely valuable in determining the parameters used in modelling the rare earth mineralization. The initial drilling in 2006-07, as well as the current drilling program, have confirmed the earlier intersections and contributed to substantially increasing the size of the area drill-tested in the Teasdale Zone. It has also increased the level of knowledge in the continuity of the geology and the geological model to allow WGM to confidently recommend additional exploration drilling to further enlarge the area of Mineral Resources. Further metallurgical testing is warranted to refine the processing flowsheet.

WGM's most recent review of Appia's Elliot Lake Project, dated 18 July, 2011, included a detailed history of mining in the Elliot Lake area. In the Elliot Lake camp, a total of 156,750 tons of uranium metal (313.5 million pounds) was produced from 13 underground mines from approximately 177 million tons of ore having an average grade of approximately 0.106% U_3O_8 (898 ppm U), equivalent to 2.12 lbs U_3O_8 per short ton. This production equals approximately 362 million pounds of U_3O_8.[1] The primary mining method used was room and pillar mining with significant resource losses in pillar support (50-70% ore extraction). The ore was hoisted and transported to a number of central mills for crushing, grinding and leaching using sulphuric acid to dissolve the uranium. Uranium was then stripped from the solution using sulphuric acid (solvent) extraction and ion exchange processes to produce a uranium-oxide precipitate (yellowcake) which was then dried and shipped for further refining. Yttrium, rare earth metals, thorium and some other metals (Fe, Al…) were also present in the pregnant solution from which yttrium was recovered

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[1] 1 million lbs of U_3O_8 are equivalent to approximately 385 metric tonnes of uranium metal.

separately. The remaining REEs were not recovered at that time because the market had little demand for these metals.

In addition to the primary mining production, a number of alternative techniques were pioneered both at Elliot Lake as well as at Kerr Addison's Agnew Lake Mine, 70 km to the east, to enhance uranium recovery. Principal amongst the secondary techniques was the use of bio-leaching and ion exchange columns to recover small amounts of uranium from mine waters being pumped to surface. Denison Mines Limited established an underground bacterial leaching program in the mid-1960s, and initiated a research program in 1980 to expand bio-leach production. In addition, underground leaching was carried out on broken ore packed into the open stopes following the primary mining of ore. The stopes were sealed and then flooded for leaching. Oxygen was introduced into the stope to accelerate the leaching. Spray leaching was also used to extract the uranium from underground pillars and a portion of the broken ore, and to some degree the wet-dry-wet cycling of the spraying program resulted in higher recoveries. At the Agnew Lake Mine, the steeply dipping geometry of the ore zones allowed Kerr Addison to 'long-hole' stopes which were similarly sealed and flooded with leachate. This proved to recover only half of the estimated uranium present in the ore. Kerr Addison also attempted heap-leaching of ore on surface, but this again failed to produce anticipated recoveries.

During the 20-year period ending in the 1980s, the accumulation of significant uranium stockpiles far exceeding market demands led to a prolonged slump in uranium prices beginning in 1981. Relative to the open-pitable high-grade ores in the Athabasca Basin of Saskatchewan, the lower grading ores of Elliot Lake fell out of favour and one by one the mines closed. Most of the mines contained significant, readily leachable uranium resources at the time that uranium prices declined, but the deep underground workings resulted in operating costs that made the resources uneconomic. That factor combined with lower demand for new uranium production resulted in the closing of the Elliot Lake camp, and the decommissioning of the mines.

Due to the significant hiatus between the closure of the mines and the present, no drill core was available for check sampling. As part of its on-going work for Appia, WGM reviewed core logs and various technical reports that were prepared by previous mine operators. WGM believes that the information in these files is an accurate representation of the state of knowledge at the time the mines closed. Furthermore, in every instance, Appia's drilling programs have confirmed the previous uranium intersections which lends considerable veracity to the historical resource estimates.

Figure 2.

APPIA ENERGY CORP.

Elliot Lake Property

Ontario, Canada

Project Location Map

In previous reports, WGM has included the use of several historical resource estimates. The estimates are based on widely spaced drilled holes completed by several of the major companies operating in the Elliot Lake area during the 1970s and 1980s. The resources are deep, and are located down-dip of previous mine workings. The use of widely spaced holes was common practice in the Blind River district due to the uniformity of the stratabound mineralization. The historical resources, which were estimated by companies well acquainted with both the area and with uranium mining, are inferred, and WGM included them in its discussion because it believes that the resources are material to the exploration potential and future economic value of the Appia mineral claims. Given these facts however, WGM cautions that the historical resource estimates do not meet current standards as defined by the Canadian Institute of Mining and Metallurgy ("**CIM**") and implemented under Canadian Securities Regulators' Rule National Instrument 43-101 ("**NI 43-101**").

This report encompasses the results of the most recent drilling on the Teasdale Zone and up-dated estimates of the Mineral Resources in that zone. The report also incorporates the results of a recently completed metallurgical testing program that has identified beneficiation as a key step in the processing of Teasdale mineralization. Hydrometallurgical testing has also shed light on the extraction of uranium and REEs from the mineralization. The continuing exploration work recommended in this report includes additional exploration drilling on the Teasdale Zone to add additional resources and to up-grade the confidence level of the existing Inferred Resources in some areas. Continued metallurgical testing is also recommended to improve the beneficiation approach, to fine-tune the extraction flowsheet and to refine the costs related to the recovery of uranium and REEs from Teasdale ore. Community out-reach is also recommended to ensure that Elliot Lake residents and their representatives are well informed concerning Appia's plans and the benefits of renewed mining in the area. As a second priority item, contingent on available funds, WGM has also recommended additional drilling on the much deeper and significantly larger mineralization in the Banana Lake Zone.

On completion of the next phase of exploration on the Teasdale Zone, Appia should complete a Preliminary Economic Assessment ("**PEA**") which includes a preliminary mine plan to exploit the Mineral Resources and a cash flow model incorporating the Mineral Resource estimates, capital cost and operating cost estimates, metallurgical recovery data, metal market data and the mining plan. The PEA would include an up-date of the Mineral Resources based on the exploration drilling recommended herein which should result in a significant increase in resources based on the current well-established geological model in which WGM has a high degree of confidence. The PEA will conceptually examine the conditions that might allow the resources in the Teasdale Zone to be brought into production. In this respect, the assessment is a material fact that is relevant to Appia's longer term plans.

2.2 TERMS OF REFERENCE

During early 2007, **Canada Enerco Corporation** ("**CEC**"), of Toronto, Ontario, Canada retained **Watts, Griffis and McOuat Limited** ("**WGM**") to undertake a review of a group of its uranium properties (the "**Property**") located in Buckles, Bouck, Beange, Gunterman and Joubin Townships, north-central Ontario. CEC is an Ontario-registered private corporation based in Toronto, Canada. Also based in Toronto, WGM is a consulting firm of geologists and engineers which has been providing high quality technical services to the mineral industry since 1962.

As a result of a vending agreement dated 1 November, 2007 between CEC and a related company, Appia Energy Corp. ("**Appia**") of Toronto whereby Appia acquired the exploration and mining rights to the Property, WGM was retained by Appia to prepare an up-date of its previous work including initial Mineral Resources estimates for the Banana Lake and Teasdale Lake zones carried out during the fourth quarter of 2008 and early 2009. WGM has been retained since that time as Appia's independent QP for the project. WGM has undertaken several resource estimates when Appia was a private company, and was asked by Appia to provide this report incorporating new and up-dated technical data.

The Property comprises a group of staked mining claims which cover the extensions of uranium ore zones from past-producing mines located on the north and south limbs of the Quirke Lake Syncline. As room and pillar mining was the favoured mining method, the mines contain substantial uranium resources left in pillars and undeveloped mineralized zones. The extensions of these zones are inferred to contain a considerable uranium resource based on previous drilling.

This report includes the previous findings, the results of the most recent drilling program, WGM's latest resource estimates and new data acquired as a result of a recently completed program of metallurgical testing carried by SGS-XRAL Laboratories at its testing facility in Lakefield, Ontario.

WGM included detailed discussions of the uranium and REE markets in its previous NI 43-101 compliant technical report filed on SEDAR. These discussions, which are summarized herein, included the conclusions expressed by the World Nuclear Association in its biennial meeting in London, UK, held on 9-11 September 2009 concerning uranium pricing. Despite the recent Fukushima nuclear accident, an incident borne not out of a technological failure but out of a natural disaster, most countries have not abandoned plans to expand their nuclear-electric generating capacities. Whereas the uranium market is forecast to grow markedly over the next several decades based on supply-demand fundamentals, REE markets are currently influenced by a high degree of political control exercised by the Government of China, and are therefore less easily predicted.

WGM understands that it is Appia's intent to use WGM's report as a supporting document to support a future Initial Public Offering "**IPO**".

2.3 SOURCES OF INFORMATION

The historical exploration information reviewed during this assignment, and incorporated into this NI 43-101 compliant report, was largely collected from the public records of the Ministry of Northern Development and Mines ("**MNDM**") offices located in Sault Ste. Marie, Ontario. WGM also relied upon its own library and research resources as well as the expertise of its personnel. WGM spot tested the information given for reliability against MNDM files. Other information was gleaned from authoritative internet sources such as the World Nuclear Association, the Canadian Nuclear Safety Commission, the Canadian Nuclear Association, the International Atomic Energy Association, and the World Information Service on Energy ("**WISE**") Uranium Project.

The site visits by WGM Senior Geologist Al Workman on 15-16 May, 2007 and during 3-4 June, 2009 were used as opportunities to collect additional public information from the records of the MNDM regional office in Sault Ste. Marie, to discuss exploration policy with ministry representatives and to meet with Ministry of the Environment officials. A recent site visit on 15 July, 2013 was made to examine drill core and to confirm selected drill site locations.

WGM has reviewed various licence documents (abstracts), but did not carry out a detailed audit of the certificates in order to verify title to any of the properties described herein. Efforts were made through discussions with MNDM personnel to understand the nature of any potential challenges which might arise in respect to resuming uranium production in the Elliot Lake area. Similar discussions were held with Ministry of the Environment (MOE) representatives. Importantly, WGM ascertained that the Ontario Government would fully uphold the rights of any mineral claim owner to undertake such development in compliance with existing laws and regulations.

Given the long period of time that has elapsed since the Elliot Lake area was actively being explored and mined, substantial sections of drill core from the key holes put down by Kerr McGee and other companies were not available for examination. Although a few representative sections of mineralized drill core have been maintained in the MNDM core library in Sault Ste. Marie, Ontario, such material is of limited quantity and great historical significance, and is therefore not available for resampling. No surface exposures were sampled as the results of such samples would have been irrelevant given the scope of the undertaking under consideration by Appia. Check samples of Appia drill core were selected for independent analysis by WGM and sent to the SGS laboratory on Don Mills (Toronto) for check assaying. The results were previously reported in detail in Workman and Breede (2011) which found that the Appia assays were representative of the uranium contained in the samples within a reasonable range of error. Routine check analysis by Appia has up-held the validity of its assay database.

This report is the responsibility of WGM which alone has been in charge of its overall presentation.

2.4 RELIANCE ON OTHER EXPERTS

The metallurgical testing program carried out by Appia and summarized herein was performed under a contract between Appia and SGS Mineral Services based in Lakefield Ontario. The work was supervised by Mr. James Brown, MASc, P.Eng., Senior Metallurgist-Hydrometallurgy and carried out under the direction of Senior WGM Associate Metallurgist, Mr. John Goode, P.Eng.

For metallurgical test data, WGM has relied on the technical report prepared by SGS Mineral Services entitled “An Investigation Into the Recovery of Uranium and Rare Earth Elements from the Teasdale Property prepared for Appia Energy Corporation, Project 13849-001, Final Report” dated 19 July, 2013 and a second SGS report entitled ‘An Investigation Into Beneficiation Testwork to assess the Recoverability of Uranium and Rare Earth Elements from the Teasdale Property prepared for Appia Energy Corporation, Project 13849-002, Final Report” dated 5 June, 2013. Both reports are appended hereto in the Attachments sections of this report. In the preparation of this WGM report, the authors have also relied on a technical report concerning the mineralogy of the Teasdale Zone prepared by SGS Mineral Services entitled "An Investigation by High Definition Mineralogy into the Mineralogical Characteristics of Three Composite Samples from the Elliot Lake Project" dated 10 May, 2013 and authored by Mr. Bernie C. Yeung, Mineralogist and Dr. Huyun Zhou, Senior Mineralogist.

During its previous site visits, WGM met with various QPs who have provided in-put to the Appia project including Mr. Robert MacGregor, P.Eng., a mining engineer resident in the city, and a former employee at the Denison Mine and Mr. Alan MacEachern, a consulting geologist and Elliot lake resident who has 40 years of experience in the mining camp, originally managing drilling programs and logging drill core during his tenure with Denison Mines Ltd. Both have contributed intimate details of their knowledge of the uranium-bearing horizons ("reefs") mined at Elliot Lake. Significant insight and additional information concerning underground uranium leaching and bioleaching was also supplied by Mr. Richard Swider, P.Eng., a former metallurgical engineer at the Denison Mine. Mr. Swider acted in his role as a WGM Associate whereas Mr. MacGregor and Mr. MacEachern are QPs who are independent of WGM.

WGM relied on the MNDM offices to provide accurate land title information and did not ascertain or confirm the legal status of the Appia mining claims beyond downloading from the MNDM a list of current claim owners in the project area. This data indicates that the claims are in good standing insofar as the office of the Mining recorder is concerned.

One key goal of the earlier WGM site visit was to locate the position of some of the Kerr McGee drill holes. Although the locations provided to WGM by the MNDM from the ministry database proved to be inaccurate, WGM was able to reliably re-locate holes using the sketch maps contained within individual drill hole records filed for assessment. To this extent, WGM's reliance on the Ministry records was conditional, and done solely as a means of establishing the search area within which the hole was likely to be found [2].

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[2] Subsequent to WGM's efforts, Appia reported that it was able to locate all of the historical holes that it searched for.

2.5 UNITS AND CURRENCY

All monetary sums relating to uranium prices are reported in United States dollars (US$) unless stated otherwise. A conversion rate of 0.90 Canadian dollars (C$) to the United States dollar (US $) was used in respect to developing and qualifying Mineral Resource estimates and cost models.

Measurements in this report are stated in the SI (metric) system. In keeping with norms in the industry, uranium grades are reported as pounds triuranium octoxide (commonly referred to as "yellowcake" or U_3O_8) per short ton. Less commonly, uranium content is reported as per cent uranium oxide (%U_3O_8).[3] Assay data may be reported as parts per million (ppm).

Uranium supply pricing is predominantly established during direct contract negotiations between producers and energy utilities, and the quantities involved are typically several million pounds or more of U_3O_8. This is commonly referred to as the "Term Market". Smaller quantities of uranium, measured in hundreds of thousands of pounds of U_3O_8, may be offered for sale on one of several "Spot Markets". The prices on the Spot Market bid by utilities seeking to increase on-site fuel supplies is generally discounted with respect to the prices established under Term contracts, however in times of uncertainty regarding future pricing, the Spot price can exceed the Term price resulting in an inverted market. Prices on both markets are quoted in US $ per pound of U_3O_8.

The classification of Mineral Resources and Mineral Reserves in Canada follows the codification established by the CIM. The CIM system, which must be followed in order to assure NI 43-101 compliance, ranks Mineral Resources and Mineral Reserves in terms of confidence level which in turn is a reflection of the types and amounts of exploration work completed. The conversion of resources to reserves is based on a study of mineral economics that establishes the economic viability of the existing resources under a specific set of conditions. The Mineral Resources estimated by WGM based on recent Appia drilling programs are in compliance with NI 43-101.

The historical resources mentioned herein this report cannot be precisely confirmed by the authors and are not suitable for investment decisions.

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[3] Units conversions :

1% U metal = 1.18% U_3O_8

2 lb U_3O_8/ton = 1 kg U_3O_8/tonne

2.6 RISK FACTORS

As is generally the case in the world at large, natural resources including mineral commodities are the property of the sovereign State, and the right to develop and exploit mineral deposits is conveyed to private interests via permitting and licensing procedures and agreements. Mineral projects must therefore meet certain statutory requirements to be permitted to go into production.

Due to a combination of legitimate concerns and irrational fears, uranium projects receive special attention which can prolong the permitting process and certain additional conditions associated with a "social licence" to operate can be imposed. This is especially true for mining projects that are located close to settlements as is the case with some of the Appia mining claims, although the main Mineral Resource areas are located at some distance from the town of Elliot Lake. WGM understands from its conversations with Mr. Bob McGregor, P.Eng. (Mining), that Elliot Lake municipal leaders are generally in favour of renewed exploration and mining activity as a means of increasing tax revenue to the city. Mr. McGregor had previously met with two town officials in his capacity as an independent consultant. This sentiment has been confirmed recently in discussions with the aforementioned Mr. MacEachern, P.Geo., a long-standing resident of Elliot Lake. WGM's meeting with Ministry of the Environment officials in Sault Ste Marie also found assurance that Appia's project would not face any unusual permitting hurdles.

There are no land use restrictions of which WGM is aware which might restrict the ability of Appia to access the project areas, or which might restrict its ability to bring its uranium property into production. The economic viability of the estimated Mineral Resources is dependent on a higher priced uranium-REE market than exists as of the date of this report. Metal market researchers at RBC Capital Markets, CIBC and at many other financial and research institutions are forecasting a shortage of U_3O_8 and substantially higher prices in 2014 and beyond that will prevail at or during the timeframe that the Appia requires for completion of its exploration and project permitting for mine development.

WGM's own analysis favours higher uranium prices based on data from such organizations as the World Nuclear Organization, the World Information Service on Energy (WISE) Uranium Project, the International Atomic Energy Agency and the US Energy Information Agency. WGM is aware that future reactor builds can be affected, and likely will be affected by alternative energy sources, especially low-cost natural gas, however the predicted uranium shortages are based on the number of reactors currently in service and under construction.

The predictions are not based on the US or Europe resuming or embracing a robust nuclear reactor construction program. Uranium pricing is a reflection of supply-demand fundamentals, and demand is easily forecast on reactor operations and construction (actual and planned). While nuclear fuel consumption is easily forecast, shocks to demand have occurred as a collateral result of major accidents, and these have affected uranium prices in the short term. Political posturing in the wake of such accidents has certainly played a role in exacerbating a decline in uranium prices. However, these shocks have proven to be transient largely because of growing global energy demand. The long-term linkage between oil and uranium prices, which is in an imbalance as of mid-2013 due a natural event in 2011 (Japan's tsunami) and not by a reactor malfunction at Fukushima, indicates that uranium prices should rise but this is not a certainty.

Balancing the forgoing caution are several factors which are favourable for the Appia project:

- new mine infrastructure development would be in brownfields areas already disturbed by industrial and mining activity;
- water, electrical, transportation and communications infrastructure is in place or close at hand;
- the processing of Elliot Lake ore is well known and faces no significant technical uncertainties;
- no First Nations land claims affect the project area;
- Appia bears no responsibility (liability) in any manner for potential future impacts arising out of historical mining operations and waste disposal; and,
- the Cameco uranium refinery is located approximately 50 km away near Blind River.

As far as WGM knows, all of the Appia mining claims that are the subject of this report are presently held by Appia without legal encumbrances to other parties or by the Government which would relate to previous mining activities. The reader is also directed to the section in this report entitled "Other Relevant Data and Information", specifically the sub-section on environmental policy and issues.

3. PROPERTY DESCRIPTION AND LOCATION

3.1 GENERAL LOCATION

The Appia uranium-REE property comprises a group of unpatented mineral claim units located in Buckles, Bouck, Beange, Gunterman, Joubin and Lehman Townships and near the town of Elliot Lake in north-central Ontario (Figure 3). Elliot Lake is located on Highway 108 approximately 26 km north of Highway 17, also known as the Trans-Canada Highway.

The project area is situated in UTM zone 17. The geographic co-ordinates of the town of Elliot Lake are 46^{o}23'N latitude and 82^{o}39'W longitude. The Appia claims have not been surveyed. As is typical for exploration properties, Appia does not own the surface rights to the underlying mineral claims. The surface rights to the claims belong to the Crown and some belong to the City of Elliot Lake. Surface rights can be acquired and there is sufficient area to construct the infrastructure necessary for mining and processing operations.

3.2 PROPERTY ADMINISTRATION AND STATUS

Mineral claim titles in the Province of Ontario are administered by the Ministry of Northern Development and Mines ("**MNDM**"). The Elliot Lake area is administered by the regional office located in the city of Sault Ste. Marie, a major government centre in north-central Ontario. Other government offices in the city include those charged with administering regulations pertaining to the environment, inland waterways, transportation and communications. The Appia Property consists of 100 staked mining claim units with original anniversary dates ranging from 19 October, 2004 to 11 December, 2009 (Table 4). Originally, 58 of the claims were held by Canada Enerco Corporation ("**CEC**"), a company related to Appia, however 100% ownership in these claims was transferred to Appia on 27 July, 2009. The terms of this agreement are discussed in Section 3.4 Nature of Appia's Interest. As can be seen from the following table, excess expenditures have been filed against the claims ensuring that they remain in good standing. Appia maintains a large positive balance in its reserves sufficient to carry the claims for another five years without being compelled to carry out additional exploration.

Table 4
Location and status of Appia Mining Claims

Township [1]	Claim Number	Recording Date	Due Date	Status	Ownership [2]	Work Required	Total Applied	Total Reserve
BEANGE	4201498	2005-May-02	2014-May-02	Active	100% *	$4,800	$33,600	$0
BEANGE	4201499	2005-May-02	2014-May-02	Active	100% *	$4,000	$28,000	$0
BEANGE	4201500	2005-May-02	2014-May-02	Active	100% *	$6,400	$44,800	$0
BEANGE	4201501	2005-May-02	2014-May-02	Active	100% *	$6,400	$44,800	$39,293
BEANGE	4201502	2005-May-02	2014-May-02	Active	100% *	$6,400	$44,800	$0
BEANGE	4201503	2005-May-02	2014-May-02	Active	100% *	$6,000	$42,000	$0
BEANGE	4201504	2005-May-02	2014-May-02	Active	100% *	$6,000	$42,000	$0
BEANGE	4205717	2005-Jun-28	2014-Jun-28	Active	100% *	$2,400	$16,800	$0
BEANGE	4207326	2005-May-02	2014-May-02	Active	100% *	$6,400	$44,800	$0
BEANGE	4219904	2007-Mar-27	2014-Mar-27	Active	100% *	$800	$4,000	$0
BEANGE	4219907	2007-Mar-27	2014-Mar-27	Active	100% *	$1,600	$8,000	$0
BEANGE	4219941	2007-Mar-27	2014-Mar-27	Active	100% *	$1,600	$8,000	$0
BEANGE	4219969	2007-Mar-27	2014-Mar-27	Active	100% *	$1,200	$6,000	$0
BEANGE	4219977	2007-Mar-27	2014-Mar-27	Active	100% *	$1,600	$8,000	$0
BEANGE	4243832	2008-Sep-12	2014-Sep-12	Active	100% *	$1,600	$6,400	$0
BEANGE	4248859	2009-Dec-11	2014-Dec-11	Active	100%	$1,600	$4,800	$0
BEANGE	4248860	2009-Dec-11	2014-Dec-11	Active	100%	$6,400	$19,200	$0
BOLGER	4219968	2007-Mar-27	2014-Mar-27	Active	100% *	$2,400	$12,000	$0
BOLGER	4248857	2009-Dec-11	2014-Dec-11	Active	100%	$1,600	$4,800	$0
BOLGER	4248858	2009-Dec-11	2014-Dec-11	Active	100%	$3,200	$9,600	$0
BOUCK	3019176	2006-Dec-21	2013-Dec-21	Active	100% *	$3,600	$18,000	$0
BOUCK	3019177	2006-Dec-21	2013-Dec-21	Active	100% *	$3,200	$16,000	$290,466
BOUCK	3019230	2006-Dec-21	2014-Dec-21	Active	100% *	$6,400	$38,400	$310
BOUCK	3019231	2006-Dec-21	2014-Dec-21	Active	100% *	$6,400	$38,400	$0
BOUCK	3019232	2006-Dec-21	2014-Dec-21	Active	100% *	$4,800	$28,800	$0
BOUCK	3019233	2006-Dec-21	2014-Dec-21	Active	100% *	$3,200	$19,200	$0
BOUCK	3019234	2006-Dec-21	2014-Dec-21	Active	100% *	$4,800	$28,800	$421,123
BOUCK	4205718	2005-Jun-28	2014-Jun-28	Active	100% *	$400	$2,800	$0
BOUCK	4207259	2006-Dec-21	2014-Dec-21	Active	100% *	$4,000	$24,000	$0
BOUCK	4207262	2006-Dec-21	2014-Dec-21	Active	100% *	$6,000	$36,000	$0
BOUCK	4215011	2007-Feb-27	2014-Feb-27	Active	100% *	$400	$2,000	$0
BOUCK	4215012	2007-Feb-27	2014-Feb-27	Active	100% *	$3,200	$16,000	$0
BOUCK	4215013	2007-Feb-27	2014-Feb-27	Active	100% *	$1,200	$6,000	$0
BOUCK	4215302	2006-Dec-29	2014-Dec-29	Active	100% *	$1,600	$9,600	$0
BOUCK	4218619	2007-Aug-01	2014-Aug-01	Active	100%	$4,000	$20,000	$0
BOUCK	4219908	2007-Mar-30	2014-Mar-30	Active	100% *	$400	$2,000	$0
BOUCK	4221243	2004-Oct-19	2014-Oct-19	Active	100% *	$5,200	$41,600	$0
BOUCK	4221244	2004-Oct-19	2014-Oct-19	Active	100% *	$2,800	$22,400	$0
BOUCK	4221245	2004-Oct-19	2014-Oct-19	Active	100% *	$6,400	$51,200	$0
BOUCK	4248854	2009-Dec-11	2014-Dec-11	Active	100%	$400	$1,200	$0
BOUCK	4248855	2009-Dec-11	2014-Dec-11	Active	100%	$400	$1,200	$0
BUCKLES	04216870*[3]	2007-Nov-13	2014-Nov-13	Active	100% *	$6,400	$32,000	$0
BUCKLES	3009193	2004-Oct-19	2014-Oct-19	Active	100% *	$1,200	$9,600	$0
BUCKLES	4201526	2004-Nov-16	2014-Nov-16	Active	100% *	$800	$6,400	$0
BUCKLES	4202357	2004-Oct-19	2014-Oct-19	Active	100% *	$800	$6,400	$191,069
BUCKLES	4202381	2004-Oct-19	2014-Oct-19	Active	100% *	$6,400	$51,200	$141,815
BUCKLES	4205719	2005-Jun-28	2014-Jun-28	Active	100% *	$4,800	$33,600	$0
BUCKLES	4215303	2006-Dec-29	2014-Dec-29	Active	100% *	$5,200	$31,200	$0
BUCKLES	4215314	2006-Dec-21	2014-Dec-21	Active	100% *	$2,000	$12,000	$710
BUCKLES	4215315	2006-Dec-21	2016-Dec-21	Active	100% *	$400	$3,200	$349
BUCKLES	4216851	2007-Nov-13	2014-Nov-13	Active	100% *	$6,000	$30,000	$0
BUCKLES	4216852	2007-Nov-13	2014-Nov-13	Active	100% *	$6,400	$32,000	$0
BUCKLES	4216869	2007-Nov-13	2014-Nov-13	Active	100% *	$6,400	$32,000	$0

Table 4
Location and status of Appia Mining Claims

Township [1]	Claim Number	Recording Date	Due Date	Status	Ownership [2]	Work Required	Total Applied	Total Reserve
BUCKLES	4216871	2007-Nov-13	2014-Nov-13	Active	100% *	$4,800	$24,000	$0
BUCKLES	4216872	2007-Nov-13	2014-Nov-13	Active	100% *	$1,200	$6,000	$0
BUCKLES	4219974	2007-Apr-13	2014-Apr-13	Active	100% *	$400	$2,000	$0
BUCKLES	4219978	2004-Oct-19	2014-Oct-19	Active	100% *	$400	$3,200	$0
BUCKLES	4219979	2004-Oct-19	2014-Oct-19	Active	100% *	$400	$3,200	$0
BUCKLES	4219980	2004-Oct-19	2014-Oct-19	Active	100% *	$400	$3,200	$0
BUCKLES	4221246	2004-Oct-19	2014-Oct-19	Active	100% *	$6,000	$48,000	$0
BUCKLES	4221249	2004-Oct-19	2014-Oct-19	Active	100% *	$6,000	$48,000	$0
BUCKLES	4221250	2004-Oct-19	2014-Oct-19	Active	100% *	$6,400	$51,200	$183,798
BUCKLES	4221251	2004-Oct-19	2014-Oct-19	Active	100% *	$4,000	$32,000	$68,188
BUCKLES	4221252	2004-Oct-19	2014-Oct-19	Active	100%	$6,400	$51,200	$0
BUCKLES	4222197	2008-Feb-19	2014-Feb-11	Active	100%	$4,800	$9,600	$0
BUCKLES	4222202	2008-Feb-19	2014-Feb-11	Active	100%	$6,000	$12,000	$0
BUCKLES	4222203	2008-Feb-19	2014-Feb-11	Active	100%	$800	$1,600	$0
BUCKLES	4226849	2008-Aug-21	2014-Aug-21	Active	100%	$1,600	$6,400	$0
BUCKLES	4226852	2008-Aug-21	2014-Aug-21	Active	100%	$1,600	$6,400	$0
BUCKLES	4228612	2008-Jan-24	2015-Jan-24	Active	100%	$1,200	$6,000	$0
BUCKLES	4228970	2008-Feb-19	2014-Feb-19	Active	100% *	$1,600	$6,400	$0
BUCKLES	4228971	2008-Feb-19	2014-Feb-19	Active	100% *	$400	$1,600	$0
GUNTERMAN	3019178	2006-Dec-21	2014-Dec-21	Active	100%	$1,200	$7,200	$0
GUNTERMAN	3019179	2006-Dec-21	2014-Dec-21	Active	100%	$4,400	$26,400	$0
GUNTERMAN	3019180	2006-Dec-21	2014-Dec-21	Active	100%	$2,400	$14,400	$0
GUNTERMAN	4215008	2007-Feb-27	2014-Feb-27	Active	100%	$4,800	$24,000	$0
GUNTERMAN	4215009	2007-Feb-27	2014-Feb-27	Active	100%	$800	$4,000	$0
GUNTERMAN	4215010	2007-Feb-27	2014-Feb-27	Active	100%	$800	$4,000	$0
GUNTERMAN	4215014	2007-Feb-27	2014-Feb-27	Active	100%	$4,800	$24,000	$0
GUNTERMAN	4215015	2007-Feb-27	2014-Feb-27	Active	100%	$1,600	$8,000	$0
GUNTERMAN	4217961	2007-Feb-07	2014-Feb-07	Active	100%	$1,200	$2,400	$0
GUNTERMAN	4218458	2007-Feb-07	2014-Feb-07	Active	100%	$1,200	$2,400	$0
GUNTERMAN	4218459	2007-Feb-19	2014-Feb-07	Active	100%	$1,600	$3,200	$0
GUNTERMAN	4218461	2007-Feb-19	2014-Feb-07	Active	100%	$1,200	$2,400	$0
GUNTERMAN	4218620	2007-Aug-01	2014-Aug-01	Active	100%	$2,400	$12,000	$0
GUNTERMAN	4218621	2007-Aug-01	2014-Aug-01	Active	100%	$4,000	$20,000	$0
GUNTERMAN	4248851	2009-Dec-11	2014-Dec-11	Active	100%	$3,200	$9,600	$0
GUNTERMAN	4248852	2009-Dec-11	2014-Dec-11	Active	100%	$4,000	$12,000	$0
GUNTERMAN	4248853	2009-Dec-11	2014-Dec-11	Active	100%	$1,600	$4,800	$0
JOUBIN	3019312	2006-Dec-21	2014-Dec-21	Active	100%	$6,000	$36,000	$0
JOUBIN	3019313	2007-Feb-02	2014-Feb-02	Active	100%	$3,600	$18,000	$0
JOUBIN	4205720	2005-Jun-28	2014-Jun-28	Active	100%	$3,600	$25,200	$0
JOUBIN	4214928	2007-Feb-27	2014-Feb-27	Active	100%	$400	$2,000	$0
JOUBIN	4215016	2007-Feb-27	2014-Feb-27	Active	100%	$1,600	$8,000	$0
JOUBIN	4215309	2006-Dec-29	2014-Dec-29	Active	100%	$4,800	$28,800	$0
JOUBIN	4215313	2007-Feb-02	2014-Feb-02	Active	100%	$3,600	$18,000	$0
JOUBIN	4226850	2008-Aug-21	2014-Aug-21	Active	100%	$3,600	$14,400	$0
JOUBIN	4226862	2008-Aug-21	2014-Aug-21	Active	100%	$3,600	$14,400	$0
JOUBIN	4226863	2008-Aug-21	2014-Aug-21	Active	100%	$3,600	$14,400	$0
LEHMAN	4243828	2008-Sep-12	2014-Sep-12	Active	100%	$6,400	$25,600	$0
					Totals	$324,400	$1,843,200	$1,337,121

NOTES:

(1) The township is designated as per the location of the #1 claim post.

(2) Ownership marked by an asterisk (*) were subject to a transfer (option) agreement between Appia Energy and CEC.

(3) This claim number was issued by the MNDM twice and the Ministry determined that the Appia claim should receive a “0” prefix to reduce confusion rather than issue a replacement claim number.

Mineral claim titles in the Province of Ontario are administered by the Ministry of Northern Development and Mines ("**MNDM**"). The Elliot Lake area is administered by the regional office located in the city of Sault Ste. Marie, a major government centre in north-central Ontario. Other government offices in the city include those charged with administering regulations pertaining to the environment, inland waterways, transportation and communications.

The Appia claims are unpatented, staked claims which are subject to annual exploration expenditure requirements. These requirements, presented in Table 1, are assessed on a per-claim basis, and must be met in order to maintain the claims in good standing. The MNDM monitors the completion of assessment work through a reporting system that demands the claim holder file an annual assessment report by the anniversary date for each claim, or group of claims. Work credits may be spread over blocks of contiguous claims.

The anniversary date for the individual claim comprising the Appia Property (Figure 3) are shown in Table 1. The total work commitment required to maintain the claims in good standing is C $324,400. At this time, Appia has filed excess expenditures and retains C$1,337,121 in reserves for meeting future requirements (Table 4). Appia has additional exploration expenditures in the amount of $1,974,263 relating to the 2012 drilling which has not yet been filed against its work requirements, and this amount will substantially increase the reserves held against future work obligations. Appia has no relinquishment plans at this time, and does not see a need to relinquish any claims in the future.

Certain of the mining claims (#4214928, 4221249 and 4228612), while valid, are currently subject to a Decommissioning Licence issued under the Nuclear Safety and Control Act and relating to previous mining activities. The licence holder, Denison Mines Inc., is obligated to undertake a work program relating to control of environmental impacts and restoration of the land. Appia is required to avoid exploration activities that might interfere with the execution of Denison's environmental programs. It is clear from correspondence received by Energy Metals Corp. (see following section) that Denison does not have the authority to grant access to these claims for the purpose of exploration drilling. Appia has no plans to carry out exploration on these licences at this time.

3.3 NATURE OF APPIA'S INTEREST

Appia holds its mineral titles as a result of having acquired the claims under the terms of an agreement dated 1 November, 2007 with Canada Enerco Corporation ("**CEC**"), a related company, which originally staked the claims in accordance with the Mining Act of Ontario R.S.O. 1990. The claims are now held 100% by Appia.

Under the Vending Agreement with CEC, Appia paid 35 million common shares to CEC in exchange for the claims and Appia was granted two options by CEC. Under the 1st option, Appia had the right to buy back 1 million of its shares at C $1 per share at any time prior to 31 August, 2008. Appia exercised this option. The 2nd option is conditional on Appia spending at least $10,000,000 on exploration on the Elliot Lake properties prior to 2 November, 2012, to define a NI 43-101 compliant uranium mineral resource on the property. This option granted Appia the right, prior that date, to buy back 9 million shares in tranches of 1 million shares at C $2 per share, subject to a price adjustment. The adjustment governed the maximum purchase price for the block of shares as follows:

> $0.10 times the NI 43-101 compliant Mineral Resources in pounds of U_3O_8.

In the event that the purchase price is less than $20 million, the option price of the 9 million shares will be adjusted to equal the maximum purchase price divided by 10 million. CEC retains a 1% uranium production payment royalty on uranium sold at a price equal to or in excess of US $130 per lb U_3O_8, as well as a 1% net smelter royalty on any precious and base metal co-production when the price of uranium equals or exceeds US $130 per lb U_3O_8.

Appia did not meet the minimum expenditure requirement imposed in the forgoing and could not exercise its 2nd option.

Under the agreement with CEC, Appia is required to maintain the Property in good standing, including any claims returned (surrendered) to CEC. In turn, CEC is required to supplement the Property with any additional claim units that it acquires within 20 km of the Property boundaries subject only to Appia's acceptance of such new claim units. Appia is responsible for paying the acquisition (staking) costs of any claim units that it acquires from CEC.

Under an agreement dated 14 February, 2008, Appia purchased a group of claims from Dan Patrie Exploration Ltd. which retains a 1% net smelter royalty on the production and sale of any uranium from the subject claims at a time when the realized price for the uranium equals or exceeds US $130 per lb U_3O_8. The claims covered in this purchase agreement are:

- 3019312 and 3019313;
- 4215309; and,
- 4215313 to 4125315 inclusive.

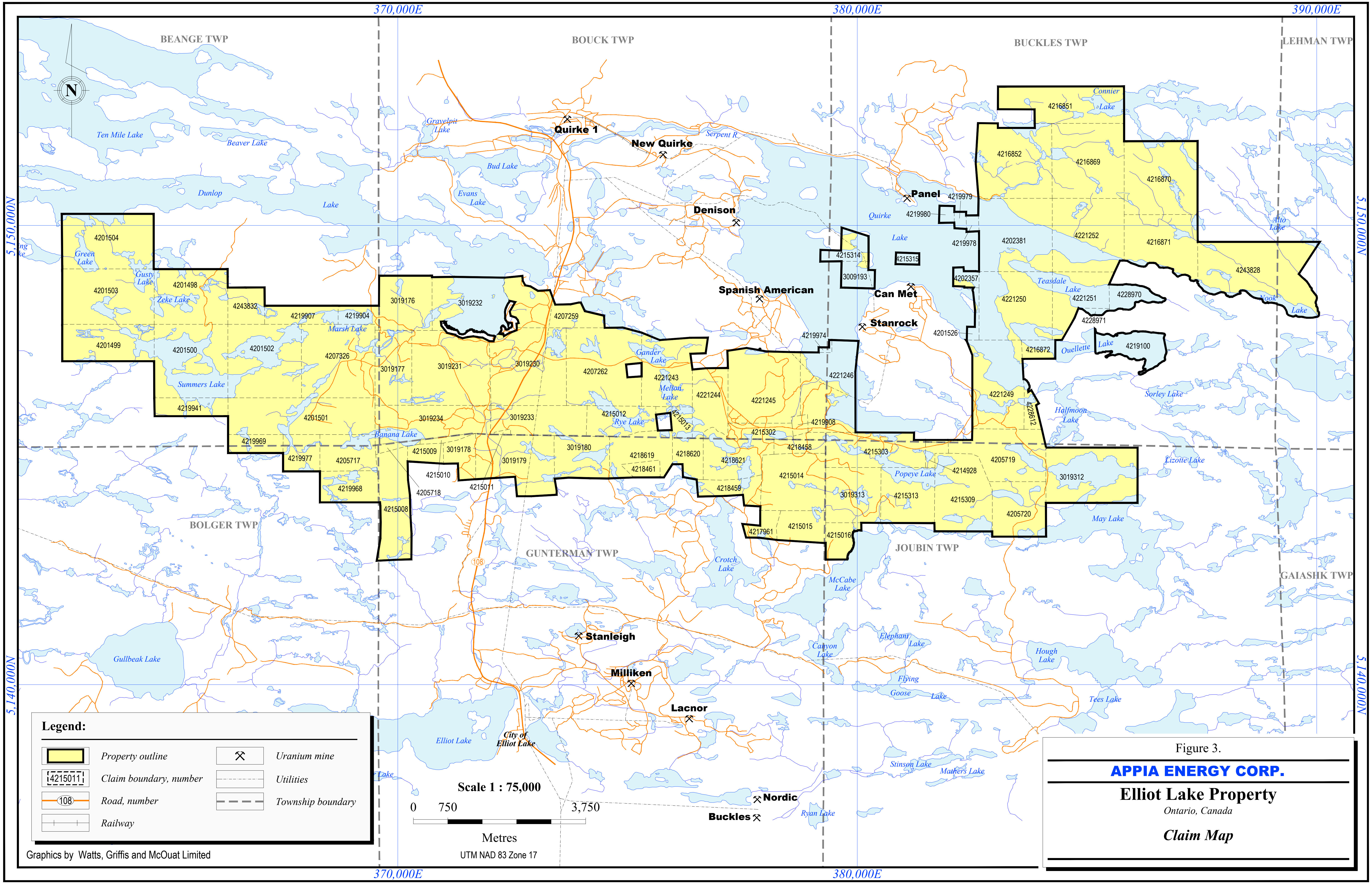
One half of the royalty payable to Dan Patrie Exploration Ltd. may be purchased by Appia for C $1 million.

CEC transferred some surface rights to the City of Elliot Lake under an agreement dated 1 November 2005, and executed 22 November, 2005, to allow the construction of a road. The mining claims so affected were 4221245 and 4221246 (formerly 3009176 and 3009177). A similar agreement on 12 January, 2009 transferred the surface rights for road construction to the City of Elliot Lake on claims 4215303, 4215313 and 4215303.

A block of claims in Buckles Township at the eastern end of the Appia Property were previously the subject of an option agreement between CEC and Energy Metals Corporation (“**EMC**”) which had an option to earn a 50% interest. That option has been relinquished, and all outstanding shares of EMC have been purchased by Uranium One Inc. In exchange for terminating the option agreement, CEC issued C $250,000 worth of stock (250,000 common shares) of Appia to EMC (now Uranium One). In turn, CEC and now Appia, must maintain in good standing those claims that were subject to the original agreement until such time as Appia completes an initial public offering (“**IPO**”). In addition, Uranium One retains the right to participate in any Appia financing (for up to 9.9%) until and including an Appia IPO or reverse take-over. No Appia claim units are under option to a second party at this time or affected by a royalty agreement.

Lastly, an agreement on 22 July, 2009 conveyed Denison the right to construct a new tailings infrastructure on claims numbered 4221247 and 4221248 in exchange for a 3% net smelter royalty on any uranium production from the subject claims. Denison also granted Appia the right of access onto claims held by Denison in the Elliot Lake area as well as the right to use former Denison mine workings to facilitate the exploration and development of Appia's Elliot Lake Project.

Watts, Griffis and McOuat



Legend:

🟨	Property outline	⚒	Uranium mine
4215011	Claim boundary, number		Utilities
108	Road, number		Township boundary
	Railway		

Graphics by Watts, Griffis and McOuat Limited

Figure 3.

APPIA ENERGY CORP.

Elliot Lake Property

Ontario, Canada

Claim Map

4. ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

4.1 ACCESS

The Appia Property is located approximately mid-way between the city of Sudbury 126 km by road to the east and the city of Sault Ste. Marie 181 km to the west. It can be reached via the Trans-Canada Highway (#17), and then via Highway #108 approximately 26 km north to the town of Elliot Lake. The town can be reached by regular northern Ontario bus service, but it is not currently serviced by air. Regularly scheduled air travel from Toronto is available on a daily basis into both Sudbury and Sault Ste. Marie.

4.2 CLIMATE

The Elliot Lake area has a northern boreal climate, moderated by its proximity to Lake Huron, with warm summers and cold dry winters. The coldest months are January and February which average -17° to -18°C. The summers are hottest during July and August with maximum temperatures of 22° to 24°C, however, summer nights tend to be cool with minimum temperatures of 11° to 12°C.

Most of the precipitation in Elliot Lake falls during the spring months of April through May and during September-October. Absolute summer and winter temperatures are moderated by the areas proximity to Lake Huron, one of the largest of the Great Lakes. Although on a latitude equal to that of Kirkland Lake, the Elliot Lake area does not experience the cold weather that the former centre receives.

4.3 LOCAL RESOURCES

Elliot Lake with a 2006 population of 11,549 is a small fraction of its former size during the uranium boom of the 1970's when its population exceeded 30,000. It is now a local supply centre for recreation areas in the north, offering a wide variety of food sources as well as general mechanical supplies and services (equipment repair, welding, auto maintenance….etc). All the major Canadian banks are represented in the city: Royal Bank of Canada, TD-Canada Trust, Bank of Nova Scotia, CIBC and the Bank of Montreal.

The Ontario government maintains two offices in Elliot Lake: the Office of the Worker Advisor which operates under the Ministry of Labour, and an office of the Ministry of Northern Development and Mines (50 Hillside Dr. North, Elliot Lake ON P5A 1X4 - Telephone (705) 848-7133. The latter office is also a "Service Ontario" office which provides a broad range of administrative services for other ministries such as transportation and health (renewal of driver's licences and health cards).

A new integrated health centre has been constructed in Elliot Lake that houses the community's doctors and other health care professionals. The city is serviced with 24-hour 911-response ambulance service provided by the Algoma District Services Administration Board. The board provides one on-site ambulance and crew 24 hours a day and an additional crew on weekdays from 8 to 4 pm for transfers to service the other outlying areas. For emergency transportation to other centres, a helicopter landing pad is located at the Elliot Lake Hospital. Air Bravo Corporation operates an air ambulance service, servicing all of north-eastern Ontario and provides charter services. Policing services in Elliot Lake are provided by the Elliot Lake detachment of the Ontario Provincial Police (OPP). Officers patrol the streets and are on duty 24 hours a day 7 days a week. The Elliot Lake fire service provides 24 hour service with a complement 34 firefighters. They have a fully equipped fire hall with an aerial pumper and a complement of rescue vehicles.

Elliot Lake is located near the northern margin of the developed corridor along the Trans-Canada Highway. As a result, there are no paved roads extending more than 20 km north of the city. Elliot Lake Municipal Airport has no regularly scheduled flights, and is currently being used for occasional auto racing.

Local and long-distance communication facilities are well developed in Elliot Lake, and many hotels can provide internet services.

Most types of field supplies and equipment are readily available in Elliot Lake, although the selection is not as complete as might be found from major suppliers in the south. Outdoor recreation equipment is generally in good supply in order to support the local recreational community. Other supplies such as office equipment and materials are readily available.

4.4 INFRASTRUCTURE

The project is situated in the famous Elliot Lake uranium mining camp. Located at the end of a regional highway, the city of Elliot Lake contains a full complement of local Government, health, education and other services. The town has good drinking water, sewage treatment, communications and electrical services which are sufficient to support mining operations. A 4,500 ft (1,385 m) paved runway (46°21’06”N 82°33’40”W) is located about 6.4 km ESE of the town although it is not serviced by regular flights at this time and its surface condition is unknown. The runway has an ESE-WNW direction striking approximately 112 degrees. The location of other infrastructure relevant to mining is shown in Table 5.

Table 5

GPS Co-ordinates for Mine Infrastructure in the Appia Project Area Near Elliot Lake

Location	Geographic Location		UTM Location			Elevation
	Latitude	Longitude	Zone	Easting	Northing	(metres)
MINE WORKINGS						
Buckles Mine	46° 22.5865’ N	82° 35.3287’ W	17T	377812	5137101	345
Can-Met Mine	46° 28.8693’ N	82° 32.8897’ W	17T	381166	5148674	n.a.
Denison Mine	46° 29.5777’ N	82° 35.8808’ W	17T	377366	5150062	n.a.
Lacnor Mine	46° 23.7363’ N	82° 36.5087’ W	17T	376343	5139261	n.a.
Milliken Mine	46° 24.1363’ N	82° 37.5085’ W	17T	375077	5140027	n.a.
New Quirke Mine	46° 30.3595’ N	82° 37.1497’ W	17T	375773	5151543	n.a.
Nordic Mine	46° 22.8030’ N	82° 35.3248’ W	17T	377825	5137501	n.a.
Panel Mine	46° 29.9053’ N	82° 32.9840’ W	17T	381083	5150595	n.a.
Quirke 1 Mine	46° 30.7528’ N	82° 38.7920’ W	17T	373688	5152315	n.a.
Spanish American Mine	46° 28.6867’ N	82° 35.4585’ W	17T	377873	5148401	n.a.
Stanleigh Mine	46° 24.6828’ N	82° 38.4148’ W	17T	373937	5141063	n.a.
Stanrock Mine	46° 28.3820’ N	82° 33.7012’ W	17T	380110	5147792	n.a.

Note: WGM measured the location of the Buckles Mine shaft. All other co-ordinates were taken from MNDM records.

4.5 AGRICULTURE

There is relatively little agriculture in the project area due to the thin soils and the short growing period having only 112 frost-free days (versus 160 days for Toronto), both representing major obstacles to market-oriented agricultural development. Some private gardens are grown locally to produce vegetables for local consumption.

Silvaculture is a major industry in the area which produces pine and spruce for the construction industry, as well as cedar and a few hardwoods such as birch as a specialty woods. Some renewed cutting is expected in the area of the Appia property during 2013.

4.6 PHYSIOGRAPHY

Located in the Canadian Shield, the project area is gently rolling with occasional bedrock scarps as much as 25 m in height (Plate 1). Elevations range from approximately 300 to 500 metres above sea level. The city of Elliot Lake is situated at 312 m above sea level. The area is dotted with a great number of lakes which is typical of the shield. The largest of these is Quirke Lake. The lakes drain towards the south into the North Channel, a body of water which forms part of Lake Huron.

Soils in the project area are generally thin as a result of protracted periods of glaciation during the Pleistocene. Areas between bedrock ridges are generally filled with glacial till with an upper muskeg or peat-covered surface. Drainage may be poor locally.

Plate 1: Winter view of terrain in the Blind River Area.

5. HISTORY

5.1 REGIONAL EXPLORATION HISTORY

The history of the Elliot Lake uranium discovery is described in some detail in Workman and Breede (2011) which can be found on the SEDAR site at www.sedar.com under company profiles for Appia Energy Corp. dated 12 Dec., 2012. The history is briefly summarized here as follows.

A modest staking rush occurred in the area now known as the Blind River District during 1948. Several samples from Long Township, 122 km east of Sault Ste. Marie contained low but measurable amounts of uranium. This led geologists into the Elliot Lake area and a high proportion of the early samples taken demonstrated economically interesting uranium values averaging 0.11% U_3O_8. Over a period of a month or so, most of the major claim blocks were staked. During the subsequent period ending in 1954, diamond drilling outlined deposits totalling more than 200 million tons grading an estimated 2.5 lbs U_3O_8 per ton. On-going surface exploration work and diamond drilling that followed in the Elliot Lake area during the 1960s and 1970s, and in areas now held by the Appia, was completed as part of the deposit evaluation and ore definition process that gave rise to the historical mines. Little work was done during the 1980s as the uranium price was in decline due to reduced reactor builds and accumulating stockpiles in the major producing countries.

5.2 MINING HISTORY

During the mid-1980s, more than half of Canada's reasonably assured uranium resources, though expensive to develop and mine, were contained in the Quirke Lake Syncline despite the addition of high-grade deposits found in the Athabasca Basin of northern Saskatchewan (International Atomic Energy Agency, 1987). The mining history of the Elliot Lake uranium camp is described in detail in Workman and Breede (2011) which can be found on the SEDAR site at www.sedar.com under company profiles for Appia Energy Corp. dated 12 Dec., 2012. This report described the production from each individual mine. What follows is a brief overview taken from that report.

During 1956, the Quirke Mine at Quirke Lake and the Nordin Mine near Elliot Lake commenced operations under the new companies Preston East Dome and Algom Uranium Mines Limited. Eldorado, the federal Crown corporation which was the sole buyer of

Canadian uranium production, gave a \$206M uranium supply contract to Algom and a \$55M contract to Pronto. However, before the Algom Mine could begin production, the company was taken over by the U.K.'s Rio Tinto Limited (Rio Tinto). By the end of 1957, Rio Tinto had also bought control of Nordic Uranium Mines Ltd. and merged its interests in three additional mines into Northspan Uranium Mines Limited. Finally Rio Tinto acquired Milliken Lake Uranium Mines from Hirshhorn. By the end of 1958, Rio Tinto had seven mines in operation supplying 40% of Canada's uranium concentrate production: Algom Quirke, Nordic, Pronto, Milliken Lake and three Northspan mines.

At this same time, another small explorer named Stanrock Uranium Mining Ltd. commissioned its mill in 1958 and started production. Realizing the value of high yttrium contents in the Elliot Lake ores, Stanrock began producing yttrium as a by product in 1965. The production was quite simple as the metal went into solution together with uranium. During this same period, Denison sank two shafts on its discovery, one a 5-compartment and the other a 7-compartment shaft, and a mill was constructed to process 6,000 tons per day. Eventually reorganized as Denison Mines Ltd., the company negotiated a \$280,600,000 contract to supply 28 M lbs of U_3O_8 to the United States between 1957 and November, 1963. When supply contracts to the United Kingdom were added to this, nearly \$500M of uranium was sold by 1963.

Having observed the Stanrock yttrium operation, Denison decided to capitalize on the growing market for yttrium which had previously been identified as a potential by-product in Elliot Lake ores. In 1966 a yttrium circuit was added to Denison's mill and production started later that year with 10,307 kg (22,724 lbs) of Y_2O_3 produced. The following year, the Elliot Lake camp reached a zenith in its output with 78,268 kg (172,551 lbs) of Y_2O_3 produced (Canadian Minerals Yearbook). The camp's output gradually diminished as the US market turned more and more to lower cost production from its own mines, including the Mountain Pass Mine in California, a major producer of cerium and lanthanum. By 1970, the output was only 33,112 kg (73,000 lbs). No production was recorded in 1971 or '72. Stanrock merged with Denison Mines Ltd. in 1973, a year that saw only 181 kg (400 lbs) of Y_2O_3 produced, but the yttrium market revived the following year which saw a collective output of 39,366 kg (86,787 lbs) of Y_2O_3 from the Elliot Lake mines. During the period 1975 through 1977, output from the Denison mine alone averaged 30,545 kg (67,340 lbs) of yttrium oxide, however by 1978 yttrium production became uneconomic due to increased reagent costs.

The oil shocks that occurred during the mid-1970s combined with rapidly growing energy demands led to significant increases in uranium demand during the 1970s and the international price for uranium oxide rose above all previous highs, reaching $43.40 per pound during the summer of 1978. This up-swing in commodity prices enabled many of the Elliot Lake uranium mines to resume production, including the Agnew Lake Mine to the east, and fuelled a second uranium exploration boom in the Elliot Lake area.

At one time, 13 uranium mines operated at Elliot Lake, most of which were owned by Rio Algom Limited as follows in Table 6. However the largest mine was the Denison Mine, and its production served as the foundation of the company that bore its name.

Table 6
Summary of Elliot Lake Mining Operations

Mine	Period of Operation	Production
Denison Mines Limited Operations		
Can-Met Mine	1957-1960	2.6 M tons of ore
Denison Mine	1957-1992	59 M tons of ore
Stanrock Mine *[1]	1958-1964; 1964-1970*[2]	+/- 6.9 M tons of ore
Rio Algom Limited Operations		
Algom (Buckles) Mine *[3]	1955-1958	124,890 tons of ore
Lacnor Mine	1956-1960	3.4 M tons of ore
Nordic Mine	1957-1968	13 M tons of ore
Milliken Mine	1957-1964	6.3 M tons of ore
Panel Mine	1957-1961; 1978-1990	15 M tons of ore
Pronto Mine *[4]	1955-1970	2.3 M tons of ore
Quirke Mine 1	1955-1961; 1965-1990	44 M tons of ore
Quirke Mine 2		production uncertain
Spanish-American Mine	1956-1960	276,000 tons of ore
Stanleigh Mine	1956-1960; 1982-1996	14 M tons of ore

*1 Amalgamated with the Denison Mine in 1973
*2 Post-1964 production was from bio-leaching
*3 Ore was milled at the Spanish American and Lacnor Mills
*4 Pronto Mill changed over to copper processing from 1960-1970

Collectively, the foregoing Denison mines produced some 156 Mlbs of U_3O_8 from 75 M tons of ore grading approximately 2.1 lbs U_3O_8 per ton. The Rio Algom mines produced approximately 206 Mlbs of U_3O_8 from 92 M tons of ore grading approximately 2.3 lbs U_3O_8 per ton. The total production was approximately 362 Mlbs of U_3O_8.

In the late 1980s, the main contractor for uranium from Elliot Lake mines was Ontario's public energy utility Ontario Hydro. Political pressure on the government and softening international uranium prices forced the government to renegotiate its contracts with Denison Mines Ltd. Faced with high mining costs, the last remaining uranium mines in the Blind River Area were forced to close. Before closure, the Denison and Agnew Lake mines attempted various innovative means to drastically reduce mining costs, such as flood leading in sealed stopes, and heap leaching, but recoveries failed to meet expectations. In 1985, Denison assessed the potential of supplying 300,000 pounds of Y_2O_3 per year to Japan, however this was never realized due to the mine's inability to sustain uranium operations.

Much can be learned from the mining history of the camp. Most of this mining was completed using conventional room and pillar methods, and whole ore was crushed, ground and leached in a conventional sulphuric acid solution. The initial mills lacked beneficiation circuits to separate the predominantly silica-bearing gangue from the pyrite-rich interstitial phase that carried the uranium and REE minerals. As production was expended, the mills simply added additional circuits of the same design. No effort was made to achieve economies through reducing the volume of material leached presumably because electrical energy used for heating was cheap in Ontario at the time. Over time this changed and costs rose both for heating the leachate and for heating the large quantities of air needed for ventilation during the winter months.

Denison Mines was also an innovator in respect to the application of bio-leach technology and in-situ (underground) leaching. This is described in detail in Workman and Breede (2011).

5.3 HISTORICAL RESERVES AND RESOURCES

5.3.1 General Overview

As mentioned in the foregoing, the closure of the Elliot Lake Mines was triggered by a collapse in uranium prices due to a tremendous over-supply of uranium on the world market far exceeding any demands from the military or from energy utilities. The inventory of uranium in various forms had been building for more than 20 years, and the fall in prices came as no great surprise to those working in the industry at the time. Mining in the Elliot Lake camp continued despite the new economic conditions due largely to long-term supply contracts that Rio Algom and Denison had negotiated with Ontario Hydro and a few other energy utilities. As these contracts were satisfied or, in the case of Ontario Hydro, cancelled through a buy-out negotiation, the mines were faced with the reality of substantially lower revenue and ever escalating costs. As a result, the mines closed leaving considerable lower grading uranium resources in the ground.

With the run-up in prices seen during 2006 and 2007 when they were poised to exceed the inflation-adjusted record uranium market prices established during the period 1977-1979, the Elliot Lake area enjoyed a renaissance. The exploration drilling by Pele Mountain Resources (“**Pele**”) that defined new Mineral Resources in Pecors Township is proof of both renewed interest and the potential for success. The mineralization on Pele’s property was known previously from considerable exploration work that outlined the uranium-bearing zone several decades ago. Pele’s so-called Eco Ridge deposit is situated approximately 10 km south of the Appia claim block. Pele have announced a NI 43-101 compliant Mineral Resources as follows in Table 7.

Table 7
Summary of Mineral Resources, Pele Mountain Resources, Eco Ridge Project
(16 April, 2012)

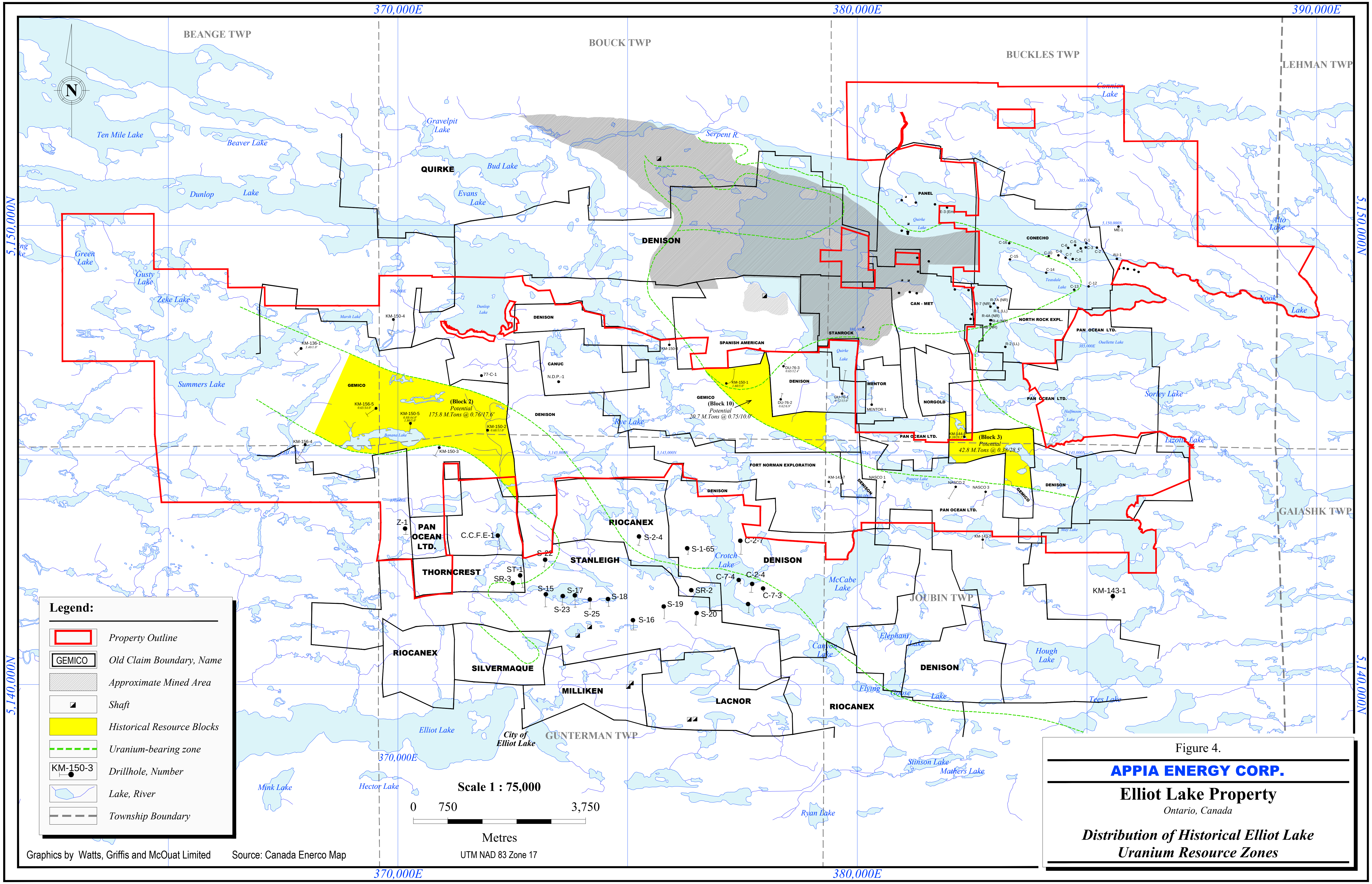
Zone & Classification	Tonnes (000s)	U3O8 (%)	U3O8 (000s lbs)	LREO	HREO	TREO	TREO
				------ (ppm) ------		---- (000s lbs) ----	
INDICATED							
MCB	20,514	0.045	20,447	1,426	193	1,618	73,184
HWZ	28,223	0.012	7,214	733	88	821	51,111
Total	**48,737**	**0.026**	**27,661**	**1,025**	**132**	**1,157**	**124,295**
INFERRED							
MCB	16,906	0.043	15,940	1,279	183	1,463	54,515
HWZ	20,956	0.013	5,822	713	95	808	37,329
Total	**37,863**	**0.026**	**21,762**	**966**	**134**	**1,100**	**91,843**

Notes:

1) Mineral Resources were estimated at a cut-off value of $100 per tonne for the Main Conglomerate Bed (“**MCB**”), and $50 per tonne for the Hanging wall Zone (“**HWZ**”) based on prices and Pele’s anticipated recoveries of uranium and rare earths, net of off-site rare earth separation costs. Pele used an average uranium price of US$70 per lb U3O8 and a rare earth “basket price” of $78 per kg (net of separation charges), and a C$:US$ exchange rate of 1.00:1.00.
2) A minimum mining thickness of 1.8 m was used for the MCB whereas the thickness of the HWZ was based on geology.
3) Scandium (Sc2O3) is also included in HREO, as it occurs in low concentrations and carries high unit values like an HREO.

APE REP / APE_09a_Historical_U_Resources_20k.dwg (Layout: 75k)
Last revision date: Thursday 25 July 2013

Watts, Griffis and McOuat



Legend:

- ▭ Property Outline
- GEMICO Old Claim Boundary, Name
- ▨ Approximate Mined Area
- ◪ Shaft
- 🟨 Historical Resource Blocks
- - - - Uranium-bearing zone
- KM-150-3 Drillhole, Number
- Lake, River
- - - - Township Boundary

Scale 1 : 75,000

0 750 3,750

Metres

UTM NAD 83 Zone 17

Graphics by Watts, Griffis and McOuat Limited

Source: Canada Enerco Map

Figure 4.

APPIA ENERGY CORP.

Elliot Lake Property

Ontario, Canada

Distribution of Historical Elliot Lake Uranium Resource Zones

Prior to Appia's exploration work in the Elliot Lake basin, historical uranium resources were known to occur in five main areas of the Appia property as shown in Figure 4. Based on drill hole evidence, they are summarized as follows:

Teasdale Lake Zone	located in Buckles Township approximately 1 km east of the former Can-Met Mine and situated obliquely on strike (and down dip) about 4 km southeast of the Panel Mine.
Gemico Block 3	located on boundary between Buckles and Joubin Townships and situated obliquely down-plunge from the Stanrock Mine
Gemico Block 10	located in south-eastern Bouck Township and down-dip of the Spanish American Mine
Banana Lake Zone	located in Beange Township and western Bouck Township, and situated in the centre of the Quirke Lake Syncline.
The Canuc Zones	located in west-central Bouck Township, and situated southwest of the Spanish American Mine in an area not intensively drilled.

Appia's exploration drilling has focused on the Teasdale and Banana Lake zones, the historical information for which is detailed in the following section. Whereas the up-dip portions of the Teasdale Zone are near the rim of the Elliot Lake Basin and approach the surface, the Banana Lake zone is located near the axis of the basin and therefore occurs at a much greater depth.

Historical drill hole locations are shown in Figure 5. Outlines of the major ore zones are shown as blue (Nordic) and green (Quirke) dashed lines. The poorly defined southern boundary of the Quirke Zone has recently been reassessed by Alan MacEachern based on the assumed position of the pinch-out of the uranium-bearing Ryan Member of the Matinenda Formation. This interpretation is also shown on Figure 5 based on the assessment report prepared by MacEachern for Appia (MacEachern, 2009).

5.3.2 Teasdale Lake Zone

The area near Teasdale Lake (Figure 4) has been drilled during many periods, but the major historical drilling programs were completed during the mid-1950s as follows:

1954-1955	Conecho Mines Ltd.	19 diamond drill holes – 9 holes not filed with Mining Recorder.
1954-1956	San Antonio Gold Mines Ltd.	6 diamond drill holes totalling 4,496.5 m (14,753 feet).
1954-1957	Roche Long Lac Gold Mines Ltd	5 diamond drill holes totalling 3,246.9 m (10,653 feet).

The foregoing holes were targeted on the south-easterly extension of the main uranium-bearing zone on the north limb of the Quirke Lake Syncline. The area of drilling was centred only three kilometres ESE of the Can-Met Mine and four kilometres east of the Stanrock Mine.

The Conecho Mines Ltd. ("**Conecho**") drilling program was evidently designed to test along strike of the Panel Mine in an area where the uranium-bearing Matinenda Fm. occurs at a relatively shallow depth. All of the holes were drilled vertically. Four of the holes reviewed by WGM produced interesting intersections:

C-4	48.8 – 52.1 m (160.0-171.0 ft)	3.3 m (11 feet)	0.4 lbs U_3O_8/ton	(0.020% U_3O_8)
C-6	59.0 – 59.4 (193.6-194.9)	0.4 m (1.3)	0.68 lbs U_3O_8/ton	(0.034% U_3O_8)
C-10	241.5- 244.4 (792.3-801.7)	2.9 m (9.4)	0.2 lbs U_3O_8/ton	(0.010% U_3O_8)
C-13	312.7-322.6 (1026.0-1058.4)	9.9 m (32.4)	0.54 lbs U_3O_8/ton	(0.027% U_3O_8)

WGM was not able to obtain logs for all of the Conecho drill holes because records for many holes (C9, C10, C12, C14 through C19) do not appear in the MNDM assessment files. Nevertheless, the records for the other holes show that the overlying sequence above the top of the Matinenda ranges in thickness from zero to 234 m (768 feet), with only three holes having more than 37 m (122 feet) of overlying material.

APE REP / APE_03_DH_Loc.dwg (Layout: 75k)
Last revision date: Thursday 25 July 2013

Watts, Griffis and McOuat

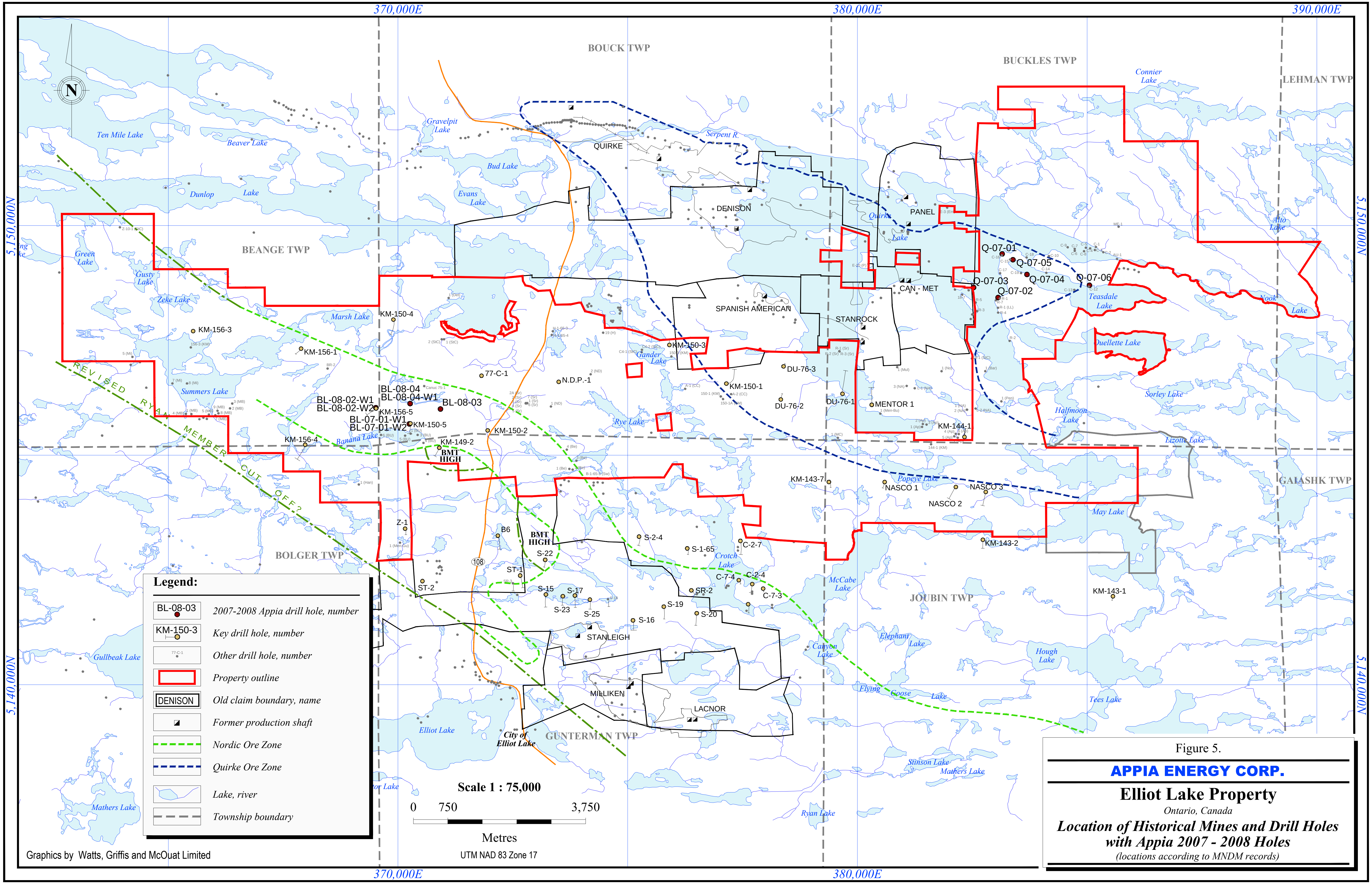


Figure 5.

APPIA ENERGY CORP.

Elliot Lake Property

Ontario, Canada

Location of Historical Mines and Drill Holes with Appia 2007 - 2008 Holes

(locations according to MNDM records)

An undated independent report written by the late Mr. Doug Sprague, P.Eng., formerly Chief Geologist of Rio Algom Ltd., for Artisan Gold Inc. from which Appia acquired the claims, reports that the first 11 holes failed to intersect commercially interesting uranium mineralization. This seems to reflect the fact that the intersections in holes C-4, C-6 and C-10 (reported above) are generally thin and/or low grade. In addition to what is in the assessment files, Sprague reports the following Conecho intersections:

C-12	*interval not available*	1.5 m (5 feet)	1.23 lbs U_3O_8/ton	(0.062% U_3O_8)
C-14	*as above*	1.5 m (5 feet)	1.12 lbs U_3O_8/ton	(0.056% U_3O_8)
C-15	*as above*	1.5 m (5 feet)	1.38 lbs U_3O_8/ton	(0.069% U_3O_8)
C-16	*as above*	1.5 m (5 feet)	1.00 lbs U_3O_8/ton	(0.050% U_3O_8)
C-17	*as above*	1.5 m (5 feet)	1.07 lbs U_3O_8/ton	(0.054% U_3O_8)
C-18	*as above*	1.5 m (5 feet)	0.98 lbs U_3O_8/ton	(0.049% U_3O_8)
C-19	*as above*	1.5 m (5 feet)	1.42 lbs U_3O_8/ton	(0.071% U_3O_8)

The foregoing Conecho drill holes C-12 through C-19 were evidently completed sometime in late 1955 or in 1956. As the host rocks are not steeply dipping in this area, the intersection length in all of the Conecho holes is very close to the true thickness of the mineralized zone, and it very closely matches the actual mining height for room and pillar mining. A compilation map produced by independent mining engineer Robert MacGregor of Sault Ste Marie, and supplied to WGM shows that C-14 and C-15 intersected, respectively, 1.2 lbs U_3O_8/ton over 4.0 feet (1.2 m) and 1.8 lbs U_3O_8/ton over 3.9 feet (1.1 m), effectively confirming the numbers reported by Sprague.

The San Antonio Gold Mines Ltd. ("**SAGM**") drilling program consisted of a single fence of six vertical holes along a north-south section located south of Teasdale Lake, and immediately east of the Appia property. In moving towards the south, the holes progressively encountered an ever thickening assemblage of strata overlying the basal Matinenda conglomerates. Holes SA-1 and SA-6 are sufficiently close to the Appia property to be of interest. Unfortunately, no assays were filed with the San Antonio drill logs. Sprague reported that none of the holes intersected values of interest. It is clear that holes SA-4 and SA-5 were not drilled deep enough to reach the Matinenda Fm. The third hole was drilled into what may be a basement high which stands above the elevation of the Matinenda Fm. The geological information from hole SA-2 is not present in the MNDM file below 3,322 feet (1,012.5 m), and with a total length of 4,215 feet (1,285 m) it is clear that the hole crossed the prospective Matinenda horizons to basement. Strong radioactivity was reported from a pitchblende vein in hole SA-1 at 2022.5 ft (616.5 m), but no assay is reported. The drill core from hole SA-6 between 2,945 and 3,010 feet (897.6-917.4 m), located immediately above the greenstone basement, was removed before the core was logged and no description is available in the public records. This is very unusual and leads immediately to the speculation that the core was well mineralized, despite Mr. Sprague's belief[4], because the hole is clearly on the trend of mineralization from the Panel Mine.

The Roche Long Lac Gold Mines ("**Roche**") holes were completed on the islands and near the main shoreline of Quirke Lake, approximately 4 km from the Panel Mine and as little as 1.5 km from the Can-Met shaft. Of the seven holes drilled, the MNDM records contain the logs and assays for five. Of these, three holes reported intersections ranging between 2 m and 9.5 m grading between 1.1 and 1.8 lbs U_3O_8 per ton as follows:

R-1	556.4 – 557.0 m (1825.3 – 1827.3 ft)	0.6 m (2.0 feet)	1.1 lbs U_3O_8/ton	(0.055% U_3O_8)
	560.3 – 561.7 m (1838.4 – 1842.9 ft)	1.4 m (4.5 feet)	1.14 lbs U_3O_8/ton	(0.057% U_3O_8)
	652.4 – 563.3 m (1845.0 – 1848.2 ft)	0.9 m (3.2 feet)	0.94 lbs U_3O_8/ton	(0.047% U_3O_8)
R-3	626.9 – 628.4 (2056.8 – 2061.8 ft)	1.5 m (5.0)	1.8 lbs U_3O_8/ton	(0.90% U_3O_8)
R-5	576.7 – 579.6 (1892.0 – 1901.5 ft)	2.9 m (9.5)	1.5 lbs U_3O_8/ton	(0.075% U_3O_8)

Hole number R-4 showed anomalous radioactivity in the interval 611.1-614.8 m (2,005-2,017 ft) but only very low uranium values of 0.01-0.02% U_3O_8 (0.2-0.4 lbs/ton) were reported. Similarly, Roche drill hole R-2 showed anomalous radioactivity at 733.0-742.5 m (2,405-2,436 ft) in the hole, but the samples did now show significant uranium assays.

Mr. Sprague completed a resource estimate which is of an uncertain date, but which WGM believes must be treated as historical and non-compliant with current CIM standards and guidelines. It is based solely on the drilling carried out during the 1950s, and is based in part on Mr. Sprague's experience gained when he was Chief Geologist, Rio Algom Ltd. during the period 1960-1990. Mr. Sprague notes "the calculations were done in the same manner that was used when the mines were in production, in fact, some of the uranium resources were calculated by the Panel Mine staff at the mine's closure as a mine-indicated resource".

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[4] Mr. Sprague notes the 65 ft section was in the Lower Mississagi Fm., however he does not describe the rock type encountered, nor does he say whether quartz-pebble conglomerates were present.

Several of the holes, notably C-20 and C-16, were drilled 250-500 m (800-1,600 feet) from Panel Mine underground workings. A polygonal approach was used whereby each drill hole intersection was applied to the grade and thickness (tonnage) of each resource block. The "mine-indicated" resource blocks are all adjacent to existing Panel Mine workings. Lower confidence "drill-indicated" blocks, are square blocks measuring 800 feet by 800 feet (244 m square) centred on drill hole intersections. Possible blocks having the lowest confidence are those areas that occur between the other blocks. The historical, non-compliant resources reported by Mr. Sprague were quantified as follows in Table 8.

Table 8
Historical Non-Compliant Resources of the Teasdale Zone

Resource Class [1]	Quantity (tons)	Grade (lbs U_3O_8/ton)	Contained U_3O_8 (lbs)
Mine Indicated	1,274,600	1.316	1,676,800
Drill Indicated	5,302,000	1.274	6,756,700
Subtotal	6,576,600	1.295	8,433,500
Possible	10,881,600	1.135	12,353,700
Total [2]	17,458,200	1.206	20,787,200

Note: 1) The foregoing resources are of a historical nature – they should not be relied upon for investment decisions as the estimates are not compliant CIM Standards and Guidelines for the estimation of Mineral Resources and Mineral Reserves and are therefore not compliant with current National Instrument 43-101 requirements. Neither Appia nor its Qualified Persons have done sufficient work to classify the historical resources as current mineral resources under current mineral resource terminology and are not treating the historical resources as current mineral resources.

2) Under current rules, the addition of higher confidence Measured and Indicated resources and lower confidence Inferred Resources ("Possible" under historical terminology) would not be allowed.

5.3.3 Gemico Block #3

Gemico Block #3 was defined by Rio Algom Ltd. within the boundaries of a group of claims that it acquired from Gemico during the late 1970s. The block is illustrated on a map drafted by Rio Algom Ltd. (1979) and provided to WGM by the MNDM in Sault Ste Marie. The map bears the title "Gemico Properties, Elliot Lake Area" and is referenced as drawing #791.

The down-dip location of the uranium-bearing conglomerates is shown on the map. WGM believes that this outline is based on mine geology and evidence from diamond drill holes. A stippled area represents that portion of the uranium-bearing zone which is located under the Gemico claims. Within this area, Rio Algom has estimated that a "potential resource" of some 42.8 million tons of mineralization exists having an average tenor of 0.38 lbs U_3O_8 per ton over an average thickness of 28.5 feet. This estimate is apparently based on a single drill hole, KM-144-1, put down by Kerr McGee near the north-western boundary of the claims. The hole intersected a zone having this grade and thickness. According to the original drill log that WGM obtained from the MNDM assessment files, the mineralized zone contains a higher grading interval at 1,118.0-1,121.4 m (3,668-3679 ft) averaging 0.46 lbs U_3O_8 per ton over a thickness of 3.4 m (11 ft). The volume of the mineralized zone is confined to the Gemico claims and is truncated by the inferred margin of the mineralized zone. It is clear that uranium mineralization extends to the east, north and west of the Gemico claims. The truncation of the mineralization to the south is not justified as two holes, Nasco #2 and Nasco #3 intersected mineralization of interest approximately 500-800 m south of the Gemico claims. Nasco #2 intersected 0.8 lbs U_3O_8 per ton over a thickness of 1.5 m (5 ft), the grade being an average of the initial intersection (0.76 lbs U_3O_8/ton over 1.5 m) and a second wedged cut (0.84 lbs U_3O_8/ton over 1.5 m). Nasco #3 intersected 0.5 lbs U_3O_8 per ton over a thickness of 4.5 m (14.9 ft).

Given Rio Algom's experience as one of the two main uranium producers, and based on the foregoing evidence, WGM accepts the above-mentioned historical estimate as a reasonable estimate of the exploration target within the Gemico #3 block which shows that a higher grading core zone is present, likely grading 0.5-0.8 lbs U_3O_8 per ton, that could positively influence the viability of mining this zone. The potential quantity and grade is conceptual in nature, there has been insufficient exploration to define a mineral resource and it is uncertain if further exploration will result in the target being delineated as a mineral resource. As the area held by Appia also covers the area surrounding the Gemico block, we conclude that the mineral potential on the Appia claims is probably greater than that estimated for the block alone, and that additional drilling is justified to increase the resource base.

5.3.4 Gemico Block #10

Rio Algom estimated that the uranium-bearing conglomerates underlying the Gemico #10 block contained a "potential resource" of 20.7 M tons with an average grade of 0.75 lbs U_3O_8 per ton with an average thickness of 3 m (10 ft). This historical estimate, which does not comply with current NI 43-101 requirements, was based on Kerr McGee drill hole KM-150-1 (1.6 lbs U_3O_8 per ton over 1.5 m [5 ft]), drilled in the north-western area of the zone, as well as two drill hole intersections on the Denison block completed by Denison Mines Ltd and Uranez Mitsui:

DU-76-2	0.62 lbs U_3O_8 per ton over 2.1 m (6.9 ft).
DU-76-3	0.65 lbs U_3O_8 per ton over 3.8 m (12.4 ft).

Like the Gemico #3, the mineral potential of the #10 block is constrained by the geographical boundaries of the claims available to Rio Algom as shown on the above-mentioned Rio Algom map. It is significant that a large block of ground to the north, previously owned by Denison Mines Ltd., is located immediately down-dip of the Stanrock and Spanish American Mines. This block is now part of the claim group held by Appia. The historical resource estimated for the #10 block was further constrained by the limits of the zone thought to be of ore grade at the time of the estimate. According to Sprague (date?), the western margin of this mineralized zone is delimited by the Ramsey Lake Scour, within which the middle Mississagi boulder conglomerate was deposited in a channel eroded downwards through the uranium-bearing Matinenda quartz-pebble conglomerates. This feature is well illustrated on Figure 5 which is from Rupert (1980).

Sprague confirms the intersection in Kerr McGee drill hole 150-1, but refers to an intersection in hole DU-76-2 of 0.40 lbs U_3O_8 per ton over 46.1 feet (14.1 m). This clearly exceeds the intersection reported from other sources, although the two are not mutually exclusive. The sample data were not available to WGM however MacEachern rightly asserts that this represents the entire Denison main zone reefs of the Quirke Ore Zone. The narrower intersection of 0.62 lbs U_3O_8 per ton over 2.1 m (6.9 ft) is the lower reef only.

WGM successfully located the intact casing for hole KM-150-1 in the field and surveyed its location by GPS. WGM's review of the Kerr McGee hole from the original log taken from MNDM assessment files shows that the zone in hole 150-1 can be widened somewhat to take in the lower grading shoulders and thereby give a mineralized width of 2.6 m (8.5 ft) grading 1.1 lbs U_3O_8 per ton.

Hole DU-76-1, collared near Quirke Lake, immediately down dip of the Stanrock Mine, and less than one kilometre east of the Gemico block also produced an interesting intersection of 0.72 lbs U_3O_8 per ton over 4.7 m (15.4 ft).

For the same reasons as cited in respect to the Gemico #3 block, WGM accepts the foregoing historical estimate as a reasonable expression of the magnitude of the exploration target in the Gemico #10 block. The potential quantity and grade is conceptual in nature, there has been insufficient exploration to define a mineral resource and it is uncertain if further exploration will result in the target being delineated as a mineral resource. As the area held by Appia also covers the area surrounding the Gemico block, we conclude that the exploration potential on the Appia claims is probably greater than that estimated for the block alone, and that additional drilling is justified to increase the resource base.

5.3.5 Banana Lake Zone

The area west, north and east of Banana Lake was included in the Gemico #2 claim block. This area has been well tested by deep diamond drill holes, most of which was completed by Kerr McGee Corp. The area north of Banana Lake was also drilled much earlier during 1955-56 by Buffalo Uranium, however the four holes completed totalled only 1,227.1 m (4,026 ft), none being greater than 343.5 m (1,127 ft) in length, and none was sufficiently deep to reach the uranium-bearing Matinenda Fm.

Based on the drilling completed, Rio Algom estimated a historical, non-compliant resource for that part of the uranium-bearing Matinenda located below the Gemico #2 claim block. As with above-mentioned estimates for the Gemico #3 and #10 blocks, the estimate for this area is constrained by the geological limits of the mineralized trend which may extend from the Stanleigh Mine to the southeast. It is also constrained by the physical limits of the claim blocks available to Rio Algom. For example, the uranium-bearing conglomerates clearly extend to the east onto a large claim block formerly controlled by Denison, however this resource area was not included in the Rio Algom estimate. According to MacEachern, Denison did not complete its own forward-looking estimate of the uranium resources on its own claims. The Rio Algom historical resource estimate is also constrained by drill holes that returned trace values for uranium or failed to intersect the Matinenda conglomerates at the anticipated depths, for example in drill holes KM-149-2, KM-156-4 and KM-150-4.

Rio Algom estimated that the Gemico #2 block claims contained a potential uranium resource of 175.8 M tons of U3O8 with an average grade of 0.76 lbs U3O8 per ton, and with an average thickness of approximately 5.4 m (17.6 ft). These historical estimates of grade and tonnage are viewed as reliable and relevant based on the information and methods used at the time. However they are not compliant with resource definitions under NI 43-101 and must be considered only as historical resources. Neither Appia nor its Qualified Persons have done sufficient work to classify the historical resource as a current mineral resource under current mineral resource terminology and are not treating the historical resource as a current mineral resource. The historical resource should not be relied upon. This historical resource estimate was based on a collection of the company's widely spaced drill holes which are summarized as follows:

KM-156-5	0.65 lbs U_3O_8 per ton	(0.33 kg/t)	over 34 ft (10.4 m)
KM-150-5	0.88 lbs U_3O_8 per ton	(0.44 kg/t)	over 44 ft (13.4 m)
KM-150-2	0.68 lbs U_3O_8 per ton	(0.34 kg/t)	over 11 ft (3.4 m)

WGM successfully located the collar and casing for the KM-150-2 drill hole, and surveyed its position by GPS to within an estimated position error of less than 3 m. As this was purely a test of WGM's ability to locate a hole using the historical records as a guide, WGM did not attempt to locate the other holes *(subsequently located by Appia)*.

WGM's review of the assessment record filed for Kerr McGee hole 156-5 showed that the record contains only 61 m (200 feet) of geology ending in the Gowganda Fm. Given the placement of the hole, it is not reasonable to conclude that the hole was intended to be a 61 m hole. No mention of hole abandonment is contained in the public record. WGM is of the opinion that the depth reported on the Rio Algom map (1,554 m or 5,099 ft) is correct and that Kerr McGee filed only that amount of the hole that was needed to maintain the claim(s) in good standing, a common practice at the time. WGM has also of the opinion that the uranium intersection reported on the Rio Algom map was also correct *(subsequently confirmed by Appia's drilling in 2007-08)*.

WGM's review of Kerr McGee hole 150-5 showed that the hole was drilled to 1,497.5 m (4,913 ft) a depth sufficient to ensure that the hole intersected the Matinenda Fm. at approximately 1,433 m (4,700 ft). However, the geology and assay results for the section below 1,346 m (4,416 ft) were selectively removed from the drill hole record filed with the Ministry. What is now available through the MNDM ERMES database (MDI Reference #41J07NE0052) lacks the geological record below 1,346 m (4,416 ft) and lacks sample data. The graphic log for the hole below 1,347.2 m (4,420 ft) has been hidden by a piece of blank paper put in place at the time the log was photocopied. Given the timeframe, Kerr McGee's failure to file complete drill hole records for assessment purposes is not surprising. WGM believes this critical information was withheld because of the higher grade and thicker nature of the uranium intersection. Hole NDM #2, drilled in the southwest corner of the Canuc Property, and filed by New Delhi Mines Ltd. in 1957 had a key section at 1,457-1,581 feet (444.1-481.9 m) in the lower Mississagi Quartzite (Matinenda Fm), only 21 feet (6.4 m) above the bottom of the hole, blacked out by felt pen to protect information WGM assumes the company considered to be proprietary (AFRI Reference # 41J07NE0061). In hindsight, it is regretful that the Mining Recorder accepted such submissions at the time [5].

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[5] At the time the drilling was completed, the Ontario Mining Recorder awarded annual assessment credits for work completed. Each claims was required to have a specific number of days of work completed per year. Diamond drilling produced credits of one day per foot drilled, but no additional credits were awarded if the drill log contained assay results. For this reason, companies commonly filed the hanging wall geology which accounted for most of the hole length, and omitted the mineralized zone, or if the zone was included, withheld the assay data.

The foregoing historical Banana Lake resources represent an approximate exploration target that is confined not only by Gemico claim boundaries, but also by drill holes completed by Kerr McGee to the south (149-2 and 156-4) and to the north (150-4) which failed to intersect the Matinenda conglomerates. The potential quantity and grade is conceptual in nature. There has been insufficient exploration to define a mineral resource and it is uncertain if further exploration will result in the target being delineated as a mineral resource. In Beange Twp., a few kilometres to the northwest, hole 156-1 intersected 1.76 lbs U_3O_8 per ton over 0.6 m (2 ft) in an area which is excluded from the foregoing historical resource. This hole suggests additional potential to the west although the economically interesting uranium grades are present in a thin horizon that would not be minable unless greater thicknesses were discovered nearby.

A Canuc Mines Ltd. annual report for 1976 reports the results of three Kerr McGee drill holes completed "immediately west of the present claim holdings of Canuc, which have located a possible extension of the Nordic Zone. These three holes suggest a block of ground which may hold 180,000,000 tons containing a potential reserve of 126,000,000 pounds of uranium oxide". Although the delineation of the "Nordic Zone" requires substantial deep drilling, this reference gives sufficient credence to the Rio Algom historical estimate for WGM to believe that a significant exploration target remains in the area. Based on this estimate, the target might conceptually be in the range of 100-200 million tons grading 0.7-1.4 lbs U_3O_8/ton and containing 70-140 M pounds of uranium oxide. Although Appia's current drilling results in this area are positive, until a comprehensive exploration drilling program is completed it is uncertain whether a NI 43-101 compliant mineral resource of this size will be identified.

Given Rio Algom's experience developing mines at Elliot Lake, WGM is prepared to accept the broad brushed historical approach to defining resources in the Elliot Lake area, especially given the stratiform character of the deposits. This approach was successfully used by both Rio Algom and Denison to plan new mines and in developing new ore zones. However, the foregoing historical resource block underlies an area totalling approximately 6.2 km^2 (2.4 mi^2) and is based on only four holes, with an additional three holes constraining the zone. Although the geological basis for the estimate appears to be reasonable, too little hard evidence is available in this area to associate the resources identified by Rio Algom with any current NI 43-101 compliant resource classification. WGM accepts the abovementioned estimate as a reasonable estimate of the magnitude of the potential exploration target in this area. WGM believes that the thicknesses reported offer underground bulk mining possibilities that could greatly reduce mining costs. Clearly, additional drilling is required to bring these historical resources into NI 43-101 compliance. It must also be stressed that the area held by Appia covers prospective areas that were not on the original claims held by Gemico, and therefore we consider the foregoing statement of potential to be a minimum reflection of the potential of the claims now held by Appia.

5.3.6 Other Zones

In its attempts to corroborate resource information, references were found to drilling carried out by Canuc Mines Limited during 1968 and subsequently on claim blocks 4, 5 and 6. In the company's 1976 annual report, the company states its plans to drill a minimum of three holes to outline mineralization which had been discovered in the northeast corner of Block 5 of its claims. The company had inferred a resource of some 7 million tons grading 1.86 lbs U_3O_8 per ton to exist in this area situated southwest of the Spanish American Mine. The area may have previously been thought to be affected by the Ramsey Lake scour that removed the lower uranium-bearing horizons of the Matinenda Formation, and which marks the NW-trending discontinuity in uranium mineralization shown in Figure 7 crossing the middle of the Appia claim block. Apparently follow-up holes intersected lower, but still interesting grades of 0.63 lbs U_3O_8 per ton over a thickness of 1.4 m (4.6 feet) in drill hole 77-C-1 located on Block 4 (Figure 4 location uncertain). Mr. Len Cunningham, P.Eng., an independent consulting engineer and resident of Kirkland Lake reviewed the results and recommended additional drilling (Canuc Annual Report, December 1977).

Although WGM was unable to determine how the aforementioned historical resources were estimated, and it was unable to locate a log for hole 77-C-1, it does appears certain that economically interesting uranium grades and some uncertain tonnage exists on the former Canuc claims. WGM believes that the foregoing historical resource, which does not comply with current estimation guidelines and standards as contained in NI 43-101, must be treated as an approximation of the size of the exploration target in this area.

5.3.7 Summary of Historical Uranium Resources

The historical resources in the foregoing zones is summarized as follows in Table 9.

Table 9
Historical Non-Compliant Resources on the Appia Elliot Lake Property

Zone	Quantity (tons)	Grade (lbs U_3O_8/ton)	Contained U_3O_8 (lbs)
Teasdale Lake	17,458,200	1.206	20,787,200
Gemico Block #3	42,800,000	0.38	16,264,000
Gemico Block #10	20,700,000	0.75	15,525,000
Banana Lake Zone	175,800,000	0.76	133,608,000
Canuc Zone	7,000,000	1.86	13,020,000
Total	263,758,200	0.76	199,204,200

The historical estimates of grade and tonnage set out above are viewed as reliable and relevant based on the information and methods used at the time. However they are not compliant with resource definitions under NI 43-101 and must be considered only as historical resources. Neither Appia nor its Qualified Persons have done sufficient work to classify the historical resources as current mineral resources under current mineral resource terminology and are not treating the historical resources as current mineral resources. The historical resources should not be relied upon.

The foregoing historical resources summarized in Table 9 were estimated by mine operators using practices that were common at the time, but which do not comply with current regulatory standards and guidelines. It is doubtful that quality control standards used at the time would meet more rigorous requirements in practice today. Specifically, the operators relied on experience and assumptions of continuity rather than factual drill hole geology and assay data. As such, these estimates do not comply with NI 43-101 and should not be used for investment decisions.

It is worth noting that WGM could find no mention of historically estimated rare earth metal resources, a reflection of the fact that such metals were affected by weak markets during the peak uranium production period, and that yttrium-REE production was incidental to uranium. As a result, drill core was not routinely assayed for such metals.

APE REP / APE_05_Rgnl_Geol.dwg (Layout: 250k L_P)
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Watts, Griffis and McOuat

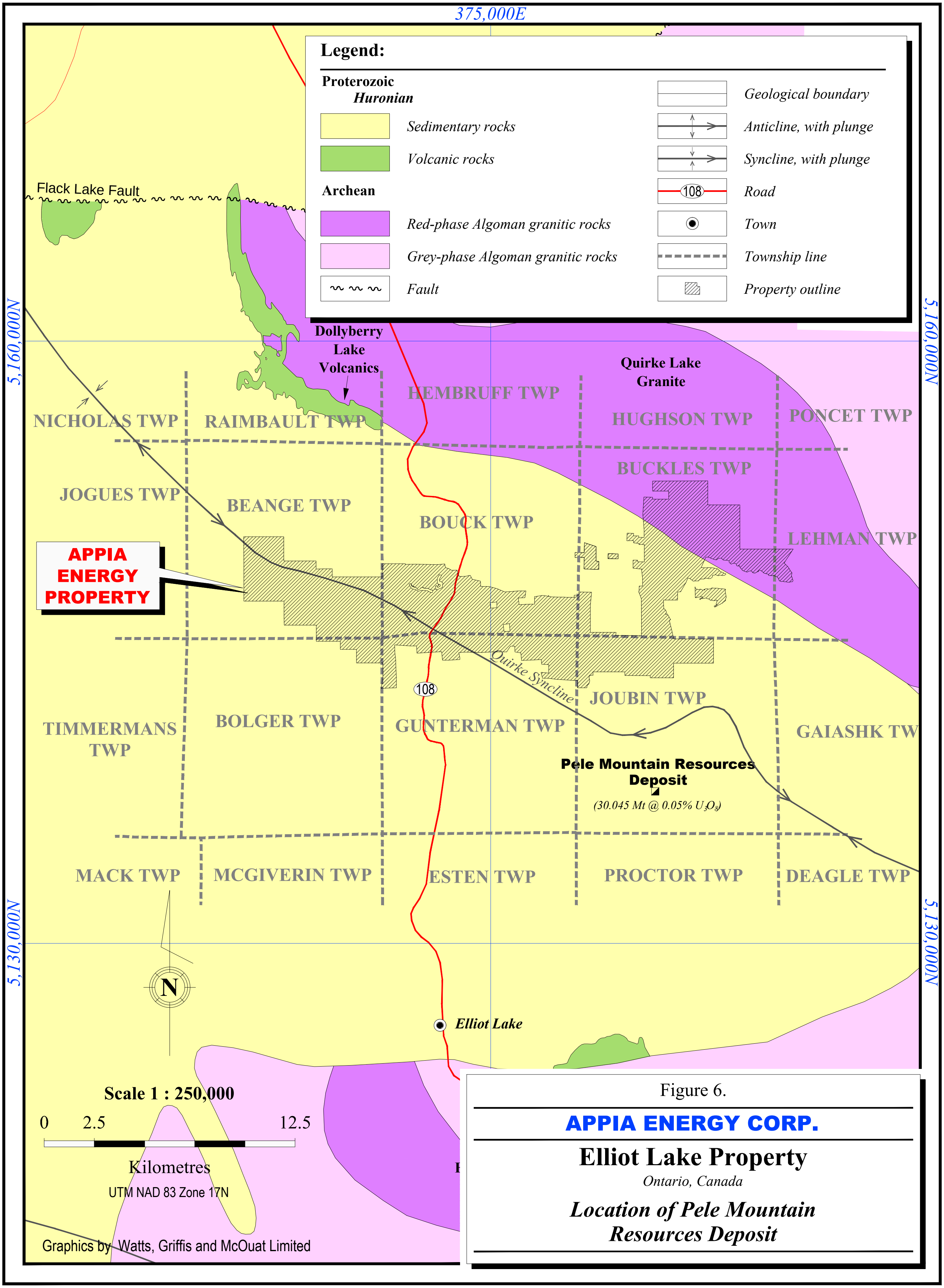

Figure 6.

APPIA ENERGY CORP.

Elliot Lake Property

Ontario, Canada

Location of Pele Mountain Resources Deposit

6. GEOLOGICAL SETTING

6.1 REGIONAL GEOLOGY

The regional geology of the Elliot Lake area was previously described in detail in Workman and Breede (2009, 2012) and will not be repeated as such here.

The Elliot Lake area is located on the southern margin of the Archean component of the Superior Province of the Canadian Shield (Figure 6). As is typical across North America, the margin is marked by a series of structural basins and troughs which contain late Archean to early Proterozoic sedimentary rocks. The structural basins or troughs that contain uranium-bearing conglomerate formed within or on the Archean continental crust. The southern limit of the Archean has not been precisely located because Paleozoic, and younger sedimentary rocks cover most of the area south of the early Proterozoic basins.

Significant iron formation deposits in NW Ontario and the American border states as well as most of the known occurrences of uraniferous quartz-pebble conglomerate are found in the marginal basins. Although the uranium deposits are diverse, and differ in age by as much as several hundred million years, they share many sedimentary and structural characteristics. The sedimentary sequences laid down on the shield margins record several transgressive cycles each resulting in deposition of fluvial-to-marine or glacial-to-marine conglomerates and sandstones, followed by shallow-marine clastic or carbonate rocks (International Atomic Energy Agency, 1987). Generally the final cycle of sedimentation ends with deep-water-marine dark shales, greywacke and volcanic rocks. Episodes of extension, compression, intrusive magmatism and metamorphism occurred during the same approximate period of time

The Lake Huron region, within which Elliot Lake is located, contains the early Proterozoic Huronian Supergroup, of which the basal deposits in the Elliot Lake district contain the world's most important deposits of uranium in Precambrian conglomerate. The ore-bearing conglomerate beds are found in the Matinenda Formation, the basal unit of the Elliot Lake Group within the Huronian Supergroup. The uranium-bearing conglomerate is a clean, well sorted, coarse-pebble conglomerate which was apparently deposited in a mixed littoral and fluvial-deltaic fan environment, possibly as the early Proterozoic sea transgressed up onto the Archean craton. The conglomerate is overlain by and interfingers in a time-transgressive relationship with the shallow-marine McKim Formation.

Pyrite is the main iron mineral found in the Matinenda Formation, whereas superseding formations contain predominantly hematite. The Th-U ratio in radioactive placer deposits first increases to greater than ten in the Lorrain Formation. This is thought to present strong evidence that during the early Proterozoic deposition of the Huronian Supergroup, a profound change in the Earth's atmosphere resulted in a transition from non-oxidizing to oxidizing conditions. Neither the uranium in the quartz-pebble conglomerates nor the iron formation deposits found elsewhere on the edge of the Archean craton would have been stable had the earth’s atmosphere not been anoxic at the time of deposition.

6.2 GEOLOGY OF THE ELLIOT LAKE AREA

The Elliot Lake area is underlain by an approximately east-west trending basin within which the Huronian sedimentary strata on-lap the Archean basement to the north, and presumably also to the south. Uranium mineralization occurs in the predominantly quartzose and arkosic basal conglomerates of the Matinenda Formation, located near the base of the Huronian sequence and unconformably overlying the Archean basement. The stratigraphic nomenclature for the Elliot Lake area is shown in Table 10.

The Huronian succession is folded into an east- trending syncline, the Quirke Lake Syncline, which is located immediately north of the city of Elliot Lake. Uranium-bearing Matinenda Formation strata are exposed on the limbs of the fold, but occur at vertical depths of +/- 1,500 m (5,000 ft) near the centre axis of the basin. Uranium mines are located on both limbs and the Quirke Lake structure has been well tested an explored by underground mine developments as well as deep exploration drilling. The Can-Met, Denison, Panel, Quirke, New Quirke, Stanrock and Spanish American mines are located on the north limb whereas the Buckles, Milliken, Lacnor, Nordic and Stanleigh mines are situated on the south limb.

The Matinenda Formation is the coarse-grained sandstone unit at the base of the stratigraphically lowest megacycle. To the north, it on-laps over an irregular Archean basement surface, filling paleo-valleys and draping over intervening hills. Uranium-bearing quartz-pebble conglomerates (Plate 2) occur within the sandstones in the lower part of the Matinenda Formation, forming laterally extensive deposits with NW-trending long axes. In a general sense, the NW end of the conglomerates either abuts against basement or is cut of by an erosive scour at the base of the overlying Ramsay Lake Formation. The conglomerates die out to the southeast by an increase in the proportion of interbedded sandstone wedges and a general reduction in grain size.

The uranium-bearing portion of the Matinenda is divided into three members. From uppermost downwards, these are the Manfred Member, the Stinson Member and the Ryan Member. The presence and thickness of these members and their uranium-bearing zones is dependent on the relative elevation of the Archean unconformity and the topography of its surface.

Two principal ore zones are present: the Quirke Ore Zone on the north limb of the basin (the Quirke Lake Syncline), and the Nordic Ore Zone on the south limb. The Quirke Ore Zone occurs in the Manfred Member of the Matinenda Formation. The Nordic Ore Zone occurs in the Ryan Member. It is important to note that there is no Ryan Member on the north limb and the Manfred Member is absent on the south limb.

Plate 2: Typical Elliot Lake ore from the Matinenda Formation – compact, well indurated quartz-pebble conglomerate with detrital pyrite and interstitial uranium minerals – 3 cm hammer scale.

Table 10

Nomenclature for Huronian Stratigraphy in the Blind River Area

Age	Group	Formation	Lithology	Thickness	Depositional Environment	Source	Mineralization
Proterozoic	Cobalt	Bar River	quartzite	>300 m at Flack Lake; >1,212 m at Willisville	shallow water	variable currents from north	
		Gordon Lake	siltstone, sandstone	300 m at Flack Lake; 1,212 m at Willisville	shallow water		
		Lorrain	quartzite, conglomerate, arkose	606-1,820 m	shallow water	north-northwest	thorium-uranium in north
		Gowganda	conglomerate, greywacke, quartzite, siltstone	152 – 1,280 m	glacial in north; glacial-marine in south	north	
							unconformity
	Quirke Lake	Serpent	quartzite	0 – 335 m	shallow water	northwest	
		Espanola	limestone, dolostones, siltstone	0 – 457 m	shallow water	northwest	traces of uranium in Victoria Twp.
		Bruce	conglomerate	0 – 61 m	glacial - shallow water	north?	
	Hough Lake	Mississagi	quartzite	0 – 914+ m	shallow water	west-northwest in west; north in southeast	uranium near basement highs
		Pecors	argillite	12 – 305+ m	shallow water	north-northwest	traces of uranium near basement highs
		Ramsay Lake	conglomerate	1.5 – 61 m	glacial - shallow water	northwest?	traces of U where unconformable on Matinenda Fm.
	Elliot Lake	McKim	argillite-greywacke	0 – 762 m	shallow water (turbidite)	northwest	traces of uranium near basement highs
		Matinenda	quartzite, arkose, conglomerate	0 – 213+ m	shallow water	northwest	uranium-thorium-rare earths in conglomerates in basement lows
				unconformity			
Archean			andesite, basalt, felsic volcanic rocks		subaerial	Flack Lake, Murray Lake	Uranium-thorium in conglomerate interbeds

Nomenclature after Robertson et al, 1969

The Stinson Member of the Matinenda Formation lacks uranium in economically interesting concentrations. The base of the Stinson in some areas of the Nordic Ore Zone is marked by angular, grey granite-clast conglomerate (as compared to quartz pebble clasts in the ore reefs), usually with a matrix of mostly smaller grey granitic material and some, mostly minor, pyrite. This horizon, is usually 2.0-5.5 m thick and is called the Stinson basal conglomerate - it can be very useful as a marker or reference horizon to indicate the top of the Nordic Ore Zone reef hosting Ryan member.

On balance of evidence, a fluvial placer mode of origin is accepted as the most reasonable genetic model for the uranium deposits hosted in the Matinenda Formation. At Elliot Lake the occurrence of large-scale flood events has been proposed as a means of widely depositing detrital uranium. Some idea of the extensive nature of these deposits is provided by the scale of Figure 7. The documented presence of glacially derived mixtites associated with Matinenda sediments leads to speculation that catastrophic ice-margin lake drainage flowing down an outwash fan deposited the uraniferous conglomeratic units present in the lower Matinenda Formation.

The Quirke Ore Zone is a classic sedimentary delta-type deposit. Quartzose and conglomeratic sediments bearing detrital uranium were introduced through a narrow 1,800 m (6,000 ft.) wide valley in the basement and spread out to the east and southeast to cover an area of approximately 80 square kilometres (30 sq. miles). There is very little Stinson member and no Ryan member between the Manfred member and the basement in the Quirke Ore Zone. Where the Manfred member is thickest, there are two pairs of reefs separated by 36 m (120 feet) of quartzite. The past producing mines of the Quirke Ore Zone were: Denison, Stanrock, CanMet, Quirke (1), New Quirke (2), Panel and Spanish American.

Outside of the mined areas at it's southeast end, much of the Nordic Ore Zone is not well defined by surface diamond drilling. It has been thought to begin approximately 6.5 kilometres (4 miles) northwest of Banana Lake as a 1.5 - 2.5 km (1 - 1.5 mile) wide basement depression channel with relatively steep basement sides (MacEachern, 2009). It extends for approximately 11 km (7 miles) south and southeast of Banana Lake, widening to approximately 13 km (8 miles). There may be some Stinson Member but no Manfred Member overlying the Ryan Member in the Nordic Ore Zone. Where the Ryan Member is thickest there are three reefs in the Nordic Ore Zone. In descending order these are the Pardee, the Nordic and the Lacnor Reefs. The past producing mines of the Nordic Ore Zone were: Stanleigh, Milliken, Lacnor, Nordic and Buckles. Most of the uranium produced was from mining in the Nordic and Lacnor Reefs. Where there is sufficient thickness of the Ryan

Member above the Pardee Reef, thin conglomerate or pebble beds called “Floater Reefs” may be present, but to date these occurrences are very thin and do not appear to be economic.

Below the Lacnor Reef, Appia holes BL-07-01, BL-08-02 and BL-08-03 have intersected reefs composed of rounded 8-15cm (3-6 inch) white quartz cobbles (Cobble Reef or Cobble Quartzite), with pale olive green irregular-shaped siltstone clasts and a few black chert clasts. Uranium grades in these rocks appear to be related to the amount of pyrite in the individual beds.

Another zone called the Pardee Zone is located approximately 4.5 km (3 miles) east of the Nordic Mine, east of the southeast corner of the Nordic Ore Zone. The Pardee Zone is approximately 2.5 square kilometres (1 square mile) in size and is separated from the Nordic Ore Zone by a basement high (ridge). Pele has been working on the Pardee Zone since early 2007 and has completed major drilling programs on its Eco Ridge Deposit.

The uranium-bearing conglomerates are massively bedded, but do show localized evidence of horizontal stratification. Trough cross-stratification due to meandering deltaic channel development is present in the pebble conglomerates in areas where numerous sandstone lenses occur. Occasionally the cross-sets can be traced from the conglomerate into sandstone lenses. Sandstones interlayered with the conglomerate and forming units separating conglomerate packages are generally trough cross-stratified with cross-set amplitude averaging approximately 12 cm.

Detrital uraninite and brannerite is concentrated in the more massive portions of the longitudinal bars as well as in lags along horizontal reactivation surfaces in stacked bars. The bars themselves represent rare, discrete high energy events in a succession that is dominated by braid-channel deposits (trough cross-stratified sandstones). The gravel bars are localized in the lower portion of the formation, usually being confined to paleovalleys (Roscoe, 1969).

The water-borne transport of uranium detritus was from north to south during deposition of the lower portions of the Matinenda. As time passed the regional paleoflow direction gradually changed to NW to SE and eventually to WNW to ESE. The counter-clockwise rotation in paleocurrent direction is thought to reflect crustal subsidence to the east of the area in which the Matinenda Formation was studied.

Pyrobitumen occurs in the Matinenda Formation in and near ore-bearing horizons. Stevenson et al (1990) report the occurrence of stratiform and dispersed kerogens, and concluded that the kerogens formed from mats of cyanobacteria that were affected by diagenetic and low-grade metamorphic processes including partial remobilization. During burial and metamorphism, rising temperatures cracked the kerogens to form petroleum, which migrated into fractures and subsequently became pyrobitumen through a combination of water-washing and thermal cracking which converted the oil into a more tarry form. As this tarry material detached from the wall, it formed spheroids that floated upward and were trapped in vuggy openings in the fractures. It is clear to WGM that the presence of kerogens might have contributed to the stabilization of uranium minerals under strongly reducing conditions in the mineralized beds.

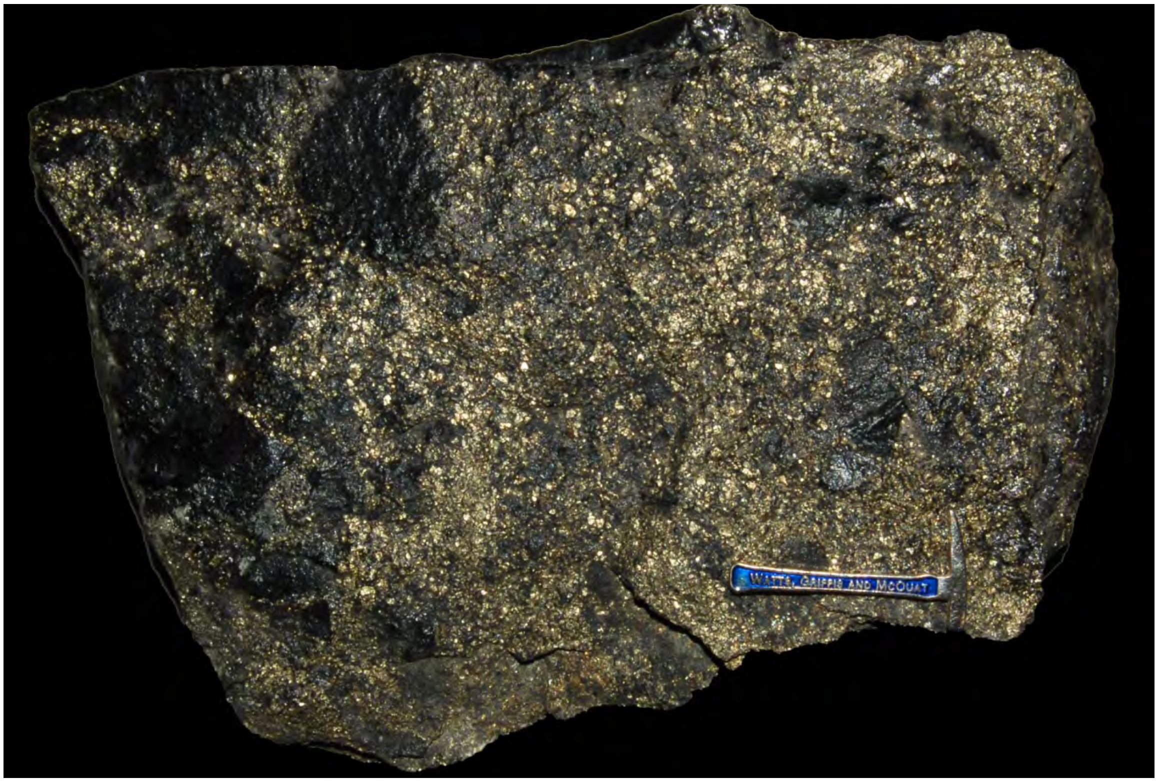

Plate 3: Black quartz-pebble conglomerate commonly referred to as “chlorite ore” - generally thought of as high-grade ore due to significant higher pyrite-brannerite contents – 3 cm hammer scale.

Economically interesting uranium mineralization is not pervasive throughout the basin. Figure 7 shows the distribution of the main uranium-bearing zones of quartz-pebble conglomerate. The favourable horizon is affected by the topography on which the conglomerates were deposited, as well as scours (river channels) which eroded down through the conglomerates following their initial deposition. As is also clear, large areas in the deep basin such as that near Banana Lake, have been shown to contain uranium values exceeding 0.5 lbs per ton. Yttrium-REE minerals have long been known to co-exist with uranium.

APE REP / APE_07_U_Distribution.dwg (Layout: 75k)
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Watts, Griffis and McOuat

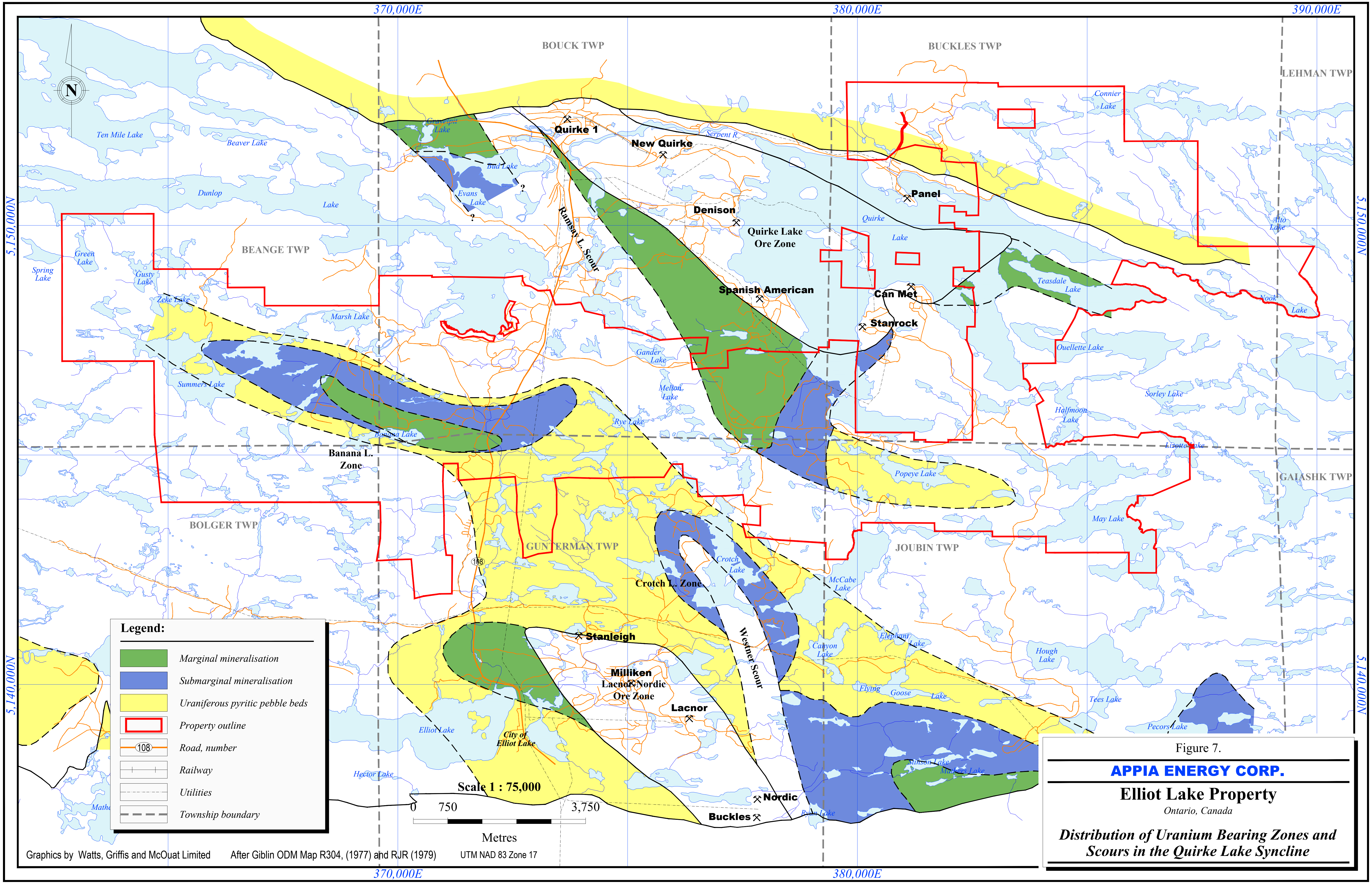


Figure 7.

APPIA ENERGY CORP.

Elliot Lake Property

Ontario, Canada

Distribution of Uranium Bearing Zones and Scours in the Quirke Lake Syncline

7. DEPOSIT TYPES

The Elliot Lake (and Agnew Lake) deposits are known as paleoplacers and classified by the Geological Survey of Canada as sub-type 1.1.1 (Eckstrand et al, 1995). Uraniferous conglomerates occur in many parts of the world, and are similar to those of other metal commodities, notably gold, platinum group metals, tin, tungsten, rare earth minerals, titanium, zirconium and chromium. The economic minerals are typically deposited in conglomerates at the base of a sedimentary cycle which may, over time, see a gradual transition to lower energy deposition. Although similarities exist between these deposits through the geological timescale, the younger deposits tend to be hematite-rich (subtype 1.1.2) whereas late Archean and early Proterozoic deposits tend to be associated with pyrite. This difference is one factor of many that indicate that the early Earth's atmosphere was anoxic and transitioned to an oxygenated atmosphere somewhat later.

The paleo-placer deposits are stratabound, commonly occurring in stacked sheet-like bodies of conglomerate. Mineralization is entire disseminated and the highest grades are associated with quartz-pebble conglomerates. The pebbles are generally well rounded (Plate 4), and some association between pebble size and uranium grade is noted. Placer deposits are created wherever rapidly flowing water allows heavy mineral particles to settle out while less dense mineral particles and rock fragments are transported through the depositional site. The term paleoplacer is generally reserved for only such mineral concentrations as constitute economically interesting deposits in lithified strata. The erosion of the parent rock and transport of detrital material results in degradation of all but the hardest minerals.

During the 1940s and 1950s, debate centred on whether these deposits were truly syngenetic (placers) or whether they were epigenetic (grown in place) or purely hydrothermal. Davidson (1958) favoured a hydrothermal model which has fallen out of favour, although the potential for recrystallization of uraninite and accretion of additional uranium onto existing mineral grains is still recognized as a possibility. Friedman (1958) points out that Th-enrichment is regional at Elliot Lake, extending well beyond those zones of U-enrichment outlined by exploration drilling. He concluded that the weight of evidence suggests a sedimentary origin for the mineralization because no known hydrothermal process could explain the widespread thorium anomaly. This is supported by Roscoe (1959) of the Geological Survey of Canada who very concisely states "the ore deposits near Blind River represent exceptional, uranium-rich, deposits within an extensive province of thorium-rich clastic sedimentary rocks". The presence of resistate minerals, such as uranium bearing silicates (zircon), is also difficult to explain in an epigenetic model. In the Blind River District, the presence of brannerite (UTi_2O_6) and U-bearing phosphates such as monazite ($[Ce,La,Nd,U,Th]PO_4$) and xenotime

(Y-UPO_4) relates quite well to the weathering of a U-Th and Ti enriched (granitic) source. Brannerite it is believed to have developed as a result of uranium ions adsorbing onto decomposing Ti minerals such as ilmenite.

Plate 4: Elliot Lake ore - rounded quartz pebbles in U-bearing, sulphide-rich matrix – 3 cm hammer scale.

Most recently, Robinson and Spooner (1984) have underscored the strong evidence for a paleoplacer origin for the U-Th mineralization in the Blind River District. New evidence shows that the regional metamorphic grade is negligible and that the quartz-pebble conglomerates are affected by syn-depositional faulting consistent with a rift margin setting. The authors add that primary uraninite grains were deposited with coarse smoky quartz, perthitic microcline, magnetite with ilmenite lamellae, monazite and zircon, however, the bulk of the pyrite which constitutes 5-10 vol-% of the ore is post-depositional in origin. Pyrite occurs as overgrowths on detrital pyrite grains and on uraninite grains altering to coffinite. The simple mineralogy of the Elliot Lake ores has been well documented.

8. MINERALIZATION

The ore mineralogy was previously described in some detail in Workman and Breede, the contents of which are summarized as follows:

- mineralization consists primarily of detrital grains of brannerite and uraninite, together with minor uranothorite, monazite and secondary coffinite associated with pyrite, pyrrhotite, zircon, rutile and Ti-magnetite as interstitial fill in a quartz pebble conglomerate;
- main ore mineral is brannerite, containing small inclusions of pyrrhotite and radiogenic galena, and occurring as ovoid, reddish-brown grains associated with bladed rutile surrounded by uranium oxides and rare earth oxides
- second most important ore mineral is uraninite which occurs as black subhedral grains up to 0.1 mm in size containing approximately 6% ThO_2 by substitution;
- monazite is a lesser ore mineral which contains an unusually high uranium content as inclusions (also uranothorite or thorite) - occurs as rounded to subangular grains typically less than 0.3 mm in diameter;
- pyrite content is typically 10-15% of the rock (Robertson, 1981) which only rarely occurs as fracture fillings; also occurs as inclusions in monazite;
- uranothorite and coffinite have been identified as minor mineral phases;
- yttrium has been an important by-product of uranium mining but the other REEs have not seen production to date;
- the average REE balance in the Elliot Lake camp is: 0.8% La_2O_3, 3.7% CeO_2, 1.0% Pr_6O_{11}, 4.1% Nd_2O_3, 4.5% Sm_2O_3, 0.2% Eu_2O_3, 8.5% Gd_2O_3, 1.2% Tb_4O_7, 11.2% Dy_2O_3, 2.6% Ho_2O_3, 5.5% Er_2O_3, 0.9% Tm_2O_3, 4.0% Yb_2O_3, 0.4% Lu_2O_3, and 51.4% Y_2O_3 ;
- non-metallic gangue minerals in the matrix of the conglomerate are represented by quartz, feldspar and sericite. In some mines a dark grey to black hued ore is reported to contain fine grained chlorite (Plate 5);
- thucholite, an organo-uranly compound (U-bearing radioactive bitumen), occurs locally as thin laminae and as a void-filling mineral phase within ore zones at Elliot Lake (Plate 6); and,
- uranium ores are hard and very well indurated resulting in favourable rock conditions for underground mining.

New and specific information concerning the mineralogy is discussed in the Metallurgy section of this report.

Plate 5: Close-up of sulphide surrounding black quartz pebbles in Matinenda conglomerate (the hammer scale is 3 cm in length).

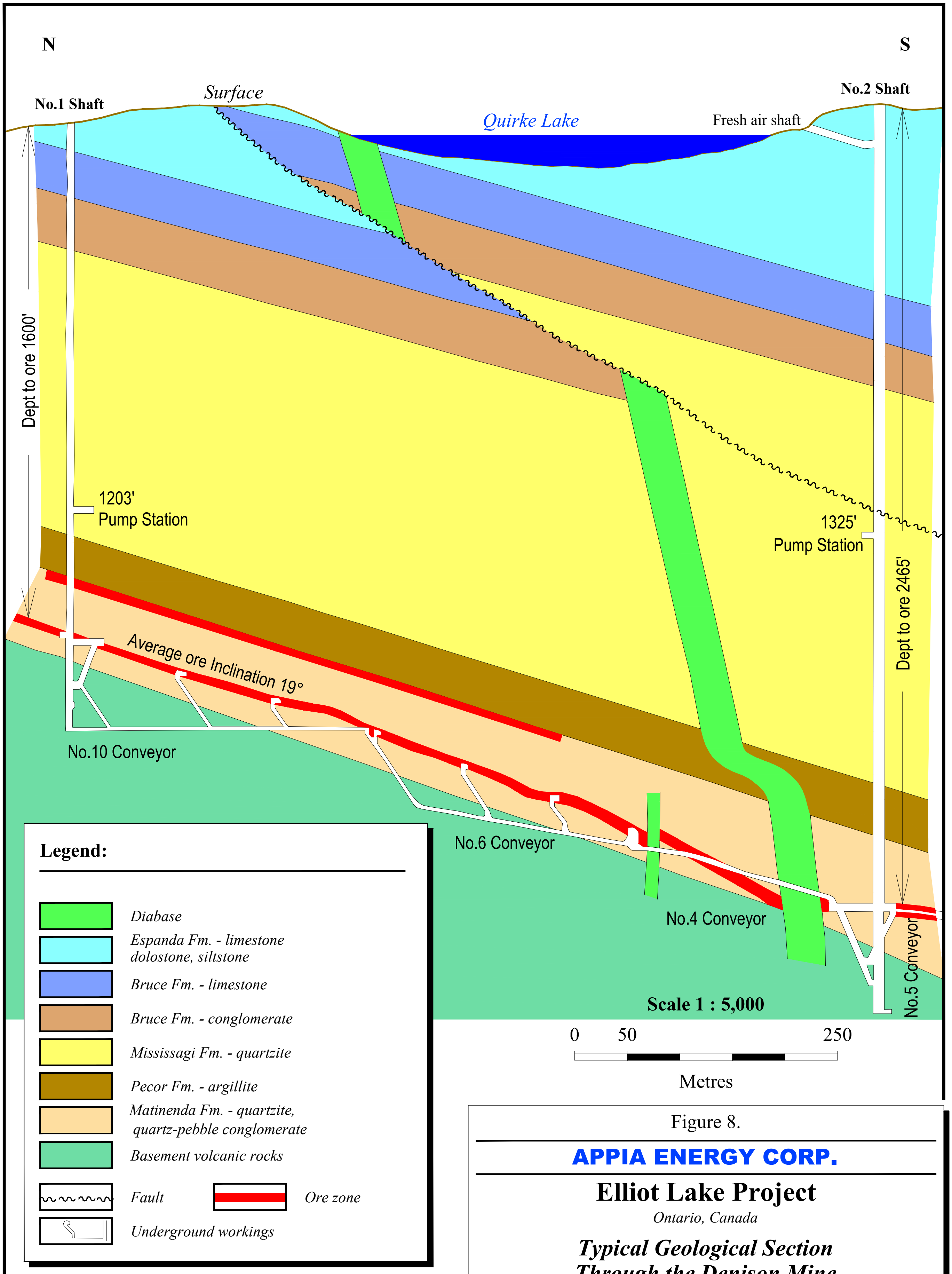
Plate 6: Black thucholite crystals in void in Matinenda Formation quartzite – 3 cm hammer.

The deposits at Elliot Lake have been historically referred to as uranium deposits because of the far greater economic importance derived from past uranium production than that from REEs and thorium. However, in the Elliot Lake deposits, REEs occur in much greater abundance than uranium. In the Teasdale Deposit Mineral Resources, as estimated in this report, the average REE content is approximately 6 times greater than the average uranium grad. Even in that part of the deposit with the highest uranium grade, the REE grade averages approximately 3 times greater.

There are few references as to the physical dimensions of the Elliot Lake deposits, in part because the mines often exploited individual portions of the same large sheet-like deposit. The mineralization is commonly referred to as stratabound and 3-5 metres in thickness and having "good lateral continuity.

Kerr Addison reported on the Agnew Lake Mine giving appreciable insights into the size of the deposits from the resource estimation parameters in use at that mine. Only deposits of considerable uniformity and size would permit the use of drill hole spacings of 400 feet (122 m) for the outlining of probable reserves as defined by Kerr Addison's mine engineering department. Given the need for accountability in production planning, one can well appreciate the uniformity of grade that supported the use of such a wide spacing as the standard convention.

Robertson (1981) describes the physical dimensions of the deposits. The largest of the deposits, the Denison Mine, measured 19,500 m long by 1,400 m to 8,000 m wide. The deposit carried an average grade of 2.5 lbs of U_3O_8 per ton of ore. The next largest at Rio Algom's Quirke Mine measured 13,000 m by 1,800-5,500 m wide. The Quirke A Reef at the #1 mine was 3.5 m thick. The Quirke #2 mine's C Reef was 1.8-3.6 m thick and other uranium-bearing horizons were present. A typical geological section through the Denison Mine is shown in Figure 8.



Graphics by Watts, Griffis and McOuat Limited Afrer Denison (1967)

Figure 8.

APPIA ENERGY CORP.

Elliot Lake Project

Ontario, Canada

Typical Geological Section Through the Denison Mine

9. CURRENT EXPLORATION

9.1 EXPLORATION PROGRAMS COMPLETED BY APPIA

The exploration completed in the recent past by Appia was covered previously in Workman and Breede. It is summarized as follows, however the reader is directed the original source for more detailed information:

2006 Geophysical Surveying

- Airborne MegaTem electromagnetic and magnetic survey by Fugro Airborne Surveys of Toronto (Mississauga);
- 3-line dipole-dipole IP test survey by Gradient Geophysics Inc. of Missoula, Montana using a 152 m (500 ft) electrode spacing and collecting data for n=1 to n=6. Some electrical sounding also performed.

2007-2008 Phase 1 diamond drilling

- diamond drilling (6 holes; 2,650.2 m/8,695 ft) of the Teasdale Zone to corroborate some of the previous drill holes and thereby support a NI 43-101 compliant Mineral Resource estimate;
- diamond drilling of 4 wedged holes (1,235 m/4,052 ft) on the Banana Lake Zone from two previous deep historical holes drilled by Kerr McGee (1969 and 1974) to corroborate the previous deep intersections

2008 Phase 2 diamond drilling

- of two new cored holes from surface on the Banana Lake Zone as well as a short wedge cut from the second hole (total of 3,109 m/10,200 ft).

Appia's early 2007-08 and 2008 drilling programs are summarized in the following table.

Table 11
Appia Diamond Drill Hole Locations and Set-Up Information, 2007-2008 Drilling Programs

Drill Hole	Wedge Depth	Geographic and UTM Co-Ordinates					Bearing	Dip	Length (m)
		Latitude	Longitude	Zone	Easting	Northing			
Q-07-01	----	46° 29' 16.35" N	82° 31' 20.97" W	17T	383151	5149382	0	-90	327.0
Q-07-02	----	46° 28' 45.52" N	82° 31' 24.33" W	17T	383061	5148432	0	-90	609.0
Q-07-03	----	46° 28' 52.28" N	82° 31' 49.52" W	17T	382528	5148651	0	-90	546.0
Q-08-04	----	46° 29' 02.17" N	82° 30' 55.30" W	17T	383690	5148934	0	-90	410.0
Q-08-05	----	46° 29' 12.41" N	82° 31' 09.89" W	17T	383385	5149256	0	-90	375.0
Q-08-06	----	46° 28' 55.43" N	82° 29' 51.32" W	17T	385050	5148700	0	-90	377.0
BL-07-01-W1	1,179.0 m	46° 27' 03.30" N	82° 41' 24.35" W	17T	370200	5145537	0	-90	345.0
BL-07-01-W2	1,169.6 m	46° 27' 03.30" N	82° 41' 24.35" W	17T	370200	5145537	0	-90	317.6
BL-08-02-W1	1,397.4 m	46° 27' 18.85" N	82° 41' 56.75" W	17T	369519	5146032	0	-90	125.6
BL-08-02-W2	1,067.0 m	46° 27' 18.85" N	82° 41' 56.75" W	17T	369519	5146032	0	-90	453.0
BL-08-03	----	46° 27' 18.94" N	82° 40' 50.89" W	17T	370924	5146005	0	-90	1,538.0
BL-08-04	----	46° 27'18.72" N	82° 41' 21.98" W	17T	370263	5146123	0	-90	1,510.0
BL-08-04-W1	1,439.0 m	46° 27' 18.86" N	82° 41' 56.76" W	17T	369519	5146032	0	-77.5	61.0

9.2 RECENT EXPLORATION

Since the completion of the drilling programs in 2008, additional diamond drilling was completed by Appia on the Teasdale Zone during 2012 which has confirmed and materially increased the size of the Teasdale Deposit. The drilling totalled 16 holes completed from surface, one wedged hole and one abandoned hole totalling 8,130.2 m of drilling (Table 12).

Table 12
Diamond Drill Hole Locations and Set-Up Information, 2012 Teasdale Drilling Program

Drill Hole	Geographic and UTM Co-Ordinates					Elevation (m)	Bearing	Dip	Length (m)
	Latitude	Longitude	Zone	Easting	Northing				
AEC 12-01	46° 28’ 57.13” N	82° 31’ 6.88” W	17T	383440	5148783	338	0	-90	522
AEC 12-01a [1]	46° 28’ 57.13” N	82° 31’ 6.88” W	17T	383440	5148783	338	0	-90	75.2
AEC 12-02	46° 29’ 20.51” N	82° 31’ 15.04” W	17T	383280	5149508	337	0	-90	244
AEC 12-03	46° 29’ 20.25” N	82° 31’ 15.03” W	17T	383280	5149500	338	20	-80	216
AEC 12-04	46° 28’ 55.92” N	82° 31’ 53.61” W	17T	382443	5148765	342	133	-70	637
AEC 12-05	46° 29’ 20.35” N	82° 31’ 14.89” W	17T	383283	5149503	340	180	-75	284
AEC 12-05b [2]	46° 28’ 36.41” N	82° 31’ 46.87” W	17T	382575	5148160	365	60	-70	633
AEC 12-06	46° 29’ 13.78” N	82° 30’ 57.03” W	17T	383660	5149293	320	0	-90	294
AEC 12-07	46° 29’ 11.87” N	82° 30’ 56.97” W	17T	383660	5149234	347	180	-65	381
AEC 12-08	46° 28’ 36.41” N	82° 31’ 46.87” W	17T	382575	5148160	365	0	-55	705
AEC 12-09	46° 29’ 11.87” N	82° 30’ 56.97” W	17T	383660	5149234	347	225	-65	384
AEC 12-10	46° 28’ 50.92” N	82° 31’ 10.88” W	17T	383351	5148593	335	0	-90	552
AEC 12-11	46° 29’ 5.24” N	82° 31’ 22.49” W	17T	383112	5149040	361	0	-90	451
AEC 12-12	46° 28’ 50.92” N	82° 31’ 10.88” W	17T	383351	5148593	335	280	-70	615
AEC 12-13	46° 29’ 5.24” N	82° 31’ 22.49” W	17T	383112	5149040	361	280	-70	460
AEC 12-14	46° 28’ 50.98” N	82° 31’ 11.02” W	17T	383348	5148595	336	0	-70	585
AEC 12-15	46° 28’ 51.24” N	82° 31’ 10.75” W	17T	383354	5148603	362	120	-60	672
AEC 12-16	46° 29’ 5.24” N	82° 31’ 22.49” W	17T	383112	5149040	361	0	-65	420

Notes:

1. *a second hole was wedged from 422.8 m- the length shown is the hole length below the wedge.*
2. *original hole AEC 12-05 was abandoned and redrilled*

Appia’s drilling was carried out during the summer months using a barge to transport the drill on Teasdale Lake and drill sites were positioned on the shore of the lake. This approach precluded the need and expense of a helicopter for moving the drill and equipment between sites. Special care an attention was used to prevent petroleum products and other drill run-off from entering the lake. The locations of the holes are shown in Figure 9. All drilling was carried out to produce NQ sized core, however some lower sections were drilled with BQ core due to problems higher in the hole and a need to reduce.

Plate 7: Off-loading barge components from the CanMet landing on Teasdale Lake.

Plate 8: Appia's drill on an island in CanMet Bay, Teasdale Lake with barge in right foreground. Note the lack of disturbance to the existing forest cover.

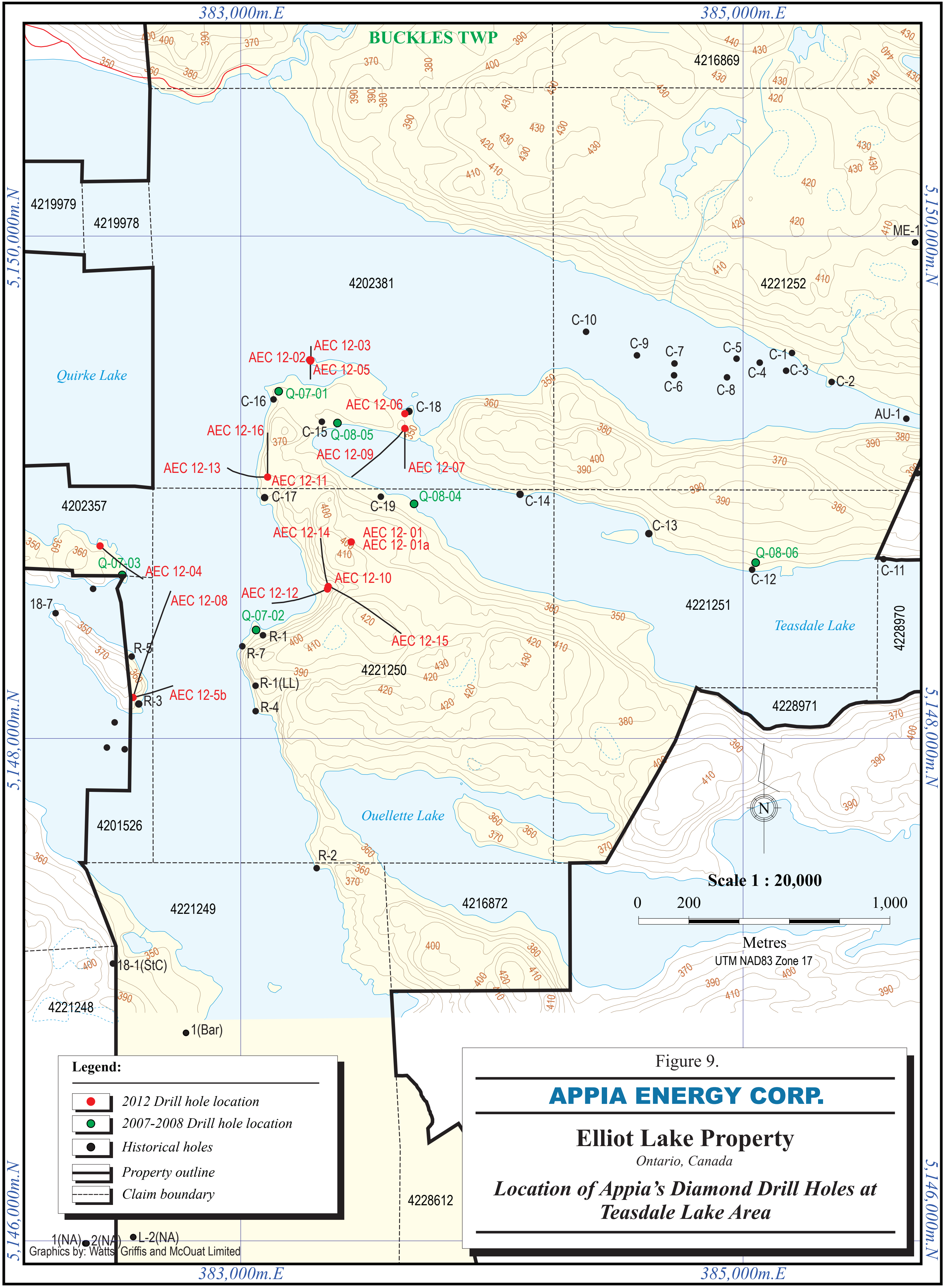


Figure 9.

APPIA ENERGY CORP.

Elliot Lake Property

Ontario, Canada

Location of Appia's Diamond Drill Holes at Teasdale Lake Area

The previous WGM drilling plan proposed to Appia called for drill sites to be established on ice platforms in Teasdale Lake during the winter in order to assure an optimum drill hole spacing in the target zone. This was deemed important as the purpose of the recommended drilling campaign was to support a new (up-dated) Mineral Resource estimate for the Teasdale Deposit. With Appia's approach to summer drilling, multiple holes were drilled from many of the drill sites on different angles and bearings to ensure that the uranium-bearing target horizon was intersected in much the same manner as originally planned. All drill holes were geologically logged by Appia's QP, Mr. Alan MacEachern, P.Geo., formerly the Chief Mine Geologist for Denison Mines Ltd., who also selected and submitted the samples for analysis. In WGM's view, the 2012 drilling accomplished much of what was previously proposed, although only 17 of the proposed 39 holes were drilled leaving many areas with significant resource potential yet to be tested.

9.3 SAMPLE PREPARATION, ANALYSIS AND SECURITY

The sampling procedure utilized by Appia's personnel during the drill program is summarized as follows:

1) the core was geologically logged and sections were selected for analysis based on geology and radiometric activity using a hand-held RS-125 Super-SPEC portable gamma ray spectrometer manufactured by Radiation Solutions of Mississauga, Ontario, Canada [6];
2) the mineralized core intervals were split in the core shack in Elliot Lake using a diamond core saw - one half of the drill core was bagged, a pre-numbered sample tag was placed in the bag and the samples was sealed before being sent to Activation Laboratories (ActLabs) in Ancaster, Ontario for analysis;
3) the remaining half of the core was retained in the core tray as a permanent record;
4) at the lab, the samples were dried, crushed and pulverized in preparation for the analysis for uranium, thorium, REEs, silver and 28 trace elements as well as the major oxides; and,
5) the trays of split drill core are stored in core racks that are inside a locked building in Elliott Lake.

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[6] The instrument uses a 6 cubic inch NaI crystal sensor to measure a full spectrum of gamma particles in either 256 or 1025 channels. In assay mode it produces count data for the conventional total count, potassium, uranium and thorium energy windows up to a maximum of 65,000 cps. Pre-programmed stripping ratios are used by the instrument to convert the measured count rates to equivalent concentrations for K (%), U (ppm) and Th (ppm). Sample time is user-selectable to improve precision through the use of longer count times. The instrument has an internal memory allowing for up to 1,000 samples and has both USB and Bluetooth output capabilities.

The un-split cores from overlying formations are being stored outside of the core logging building, cross-stacked and covered within a fenced area.

Most elements including the REEs, uranium, thorium and silver, were determined in accordance with Actlabs fusion-mass spectrometer technique; some trace elements and the major oxides used the fusion-ICP technique (both Code 8 analyses).

9.4 DATA VERIFICATION

WGM's verification procedures for historical data, including its success in verifying some of the previously reported drill hole locations and uranium intersections, are discussed in previous reports (Workman and Breede) and will not be repeated here.

WGM has no information on the REE grades of the historical drill core discussed in this report. The logs that report uranium grades lack REE data. We believe that REE assaying was seldom practised as a result of the weak commodity prices that prevailed during much of the uranium mining that occurred in the Elliot Lake camp. We believe that the assured by-product nature of the yttrium-REE production coupled with the relative uniformity of yttrium and REE grades did not provide sufficient incentive for mines and explorers to incur the additional analytical costs.

WGM also confirmed the historical Teasdale resources using a simplified model that duplicated the polygonal approach used previously by Sprague, applying variables to the inputs that might represent a reasonable range of best-estimate scenarios. Details of this are provided in Workman and Breede.

Despite being very active in the design and execution of the initial drilling programs, WGM visited the Appia drill sites after each of the 2007-08 drilling programs, confirmed drill sites and selected check samples for its independent reanalysis. WGM found no significant irregularities in the approaches used by Appia or the information reported. WGM was also actively involved in the communications and decision-making process pertaining to the 2012 drilling. WGM visited the project site on 15 July, 2013 to examine the drill core, to discuss the project status with Mr. MacEachern and to confirm the locations of selected drill sites.

10. 2012 DIAMOND DRILLING

10.1 SUMMARY

As summarized in the foregoing section, Appia completed 16 diamond drill holes from surface and one wedged hole totalling 8,130.2 m of drilling between mid-June and early September, 2012 (Table 12). This included one hole that was abandoned and redrilled due to a failure in the initial hole. The drilling contractor was an independent (private) company under the management of Mr. Richard Lavoie of Val d'Or, Quebec. All holes were drilled through the uranium-brearing hosizons in the lower Matinenda Formation and many were drilled into the Archean basement comprising granite or volcanic rocks.

Plate 9: Appia's smaller drill on the shoreline of Teasdale Lake – sites were chosen to minimize the need for access roads.

Appia's original plan, based on WGM's recommendations made in 2008, was to carry out a winter drilling program using ice platforms to drill-test the Teasdale Deposit on centres approximately 50 m apart. The goal of this program was to provide a basis for classifying the uranium resources as predominantly Indicated Resources. For a variety of reasons, including the desire to avoid the use of an expensive helicopter for project support, Appia elected to carry out the drilling during the summer months. A barge was used to transport the drill on Teasdale Lake, and the drill was off-loaded onto islands and the main shoreline where drill sites were positioned. Special care and attention was used to prevent petroleum products and other drill run-off from entering the lake. Co-ordinate and set-up data for the holes was given in Table 14 under "Recent Exploration", and the hole locations are shown in Figure 9.

The drill holes were completed with NQ equipment at angles ranging from -55 degrees to vertical. In many cases, several drill holes were completed from the same collar location employing different angles or a variety of bearings to provide spaced intersections on the uranium-bearing formation. This approach was made possible by the moderate dip of the host rocks and capitalized on the predictable location and depth of the mineralization. The net benefit of Appia's approach was a reduction of drilling costs from that budgeted. Inclusive of all costs relating to personnel, sampling, assaying…etc, the total cost of the drilling completed was approximately $1.974 million or $243 per metre. This compares very favourably with the original $7 million budget for a 15,405 m program ($454/metre[7]).

The drilling was supported using a Reflex EZ-Shot down-hole surveying tool to capture bearing and inclination data on approximate 50-metre intervals. All drill hole collars were surveyed by GPS using the NAD 83 datum.

Appia analysed a total of 1,213 samples from the 17 diamond drill holes during the course of the 2012 drilling program. In addition, Appia included quality assurance/quality control ("**QA/QC**") samples made up of 27 duplicate samples, 13 blanks and 22 certified standards. Appia's control samples comprised approximately 1 sample in every 20. The samples were analysed at the Actlabs accredited geochemical laboratory located in Ancaster, Ontario as described in Section 9.3 of this report.

10.2 DRILL HOLE GEOLOGY

Typical stratigraphic sequences found in the Teasdale drill holes are shown for drill holes AEC 12-08 and AEC 12-09 in Figure 9. The Appia drilling intersected the uraniferous conglomerates at varying depths due to the Matinenda Formation's moderate southerly dip.

Most of the Matinenda formation is composed of quartzite with occasional interbeds of pebbly quartzite. The quartz-pebble conglomerates that are the important uranium-bearing horizons ("reefs") are located in the lowermost part of the Matinenda. Although thin uranium-bearing conglomerate horizons may occur anywhere in the lower half of the Matinenda, two major mineralized horizons are specifically recognized as being of economic importance: the Upper Reef ("**UR**") and the Lower Reef ("**LR**"), separated by the

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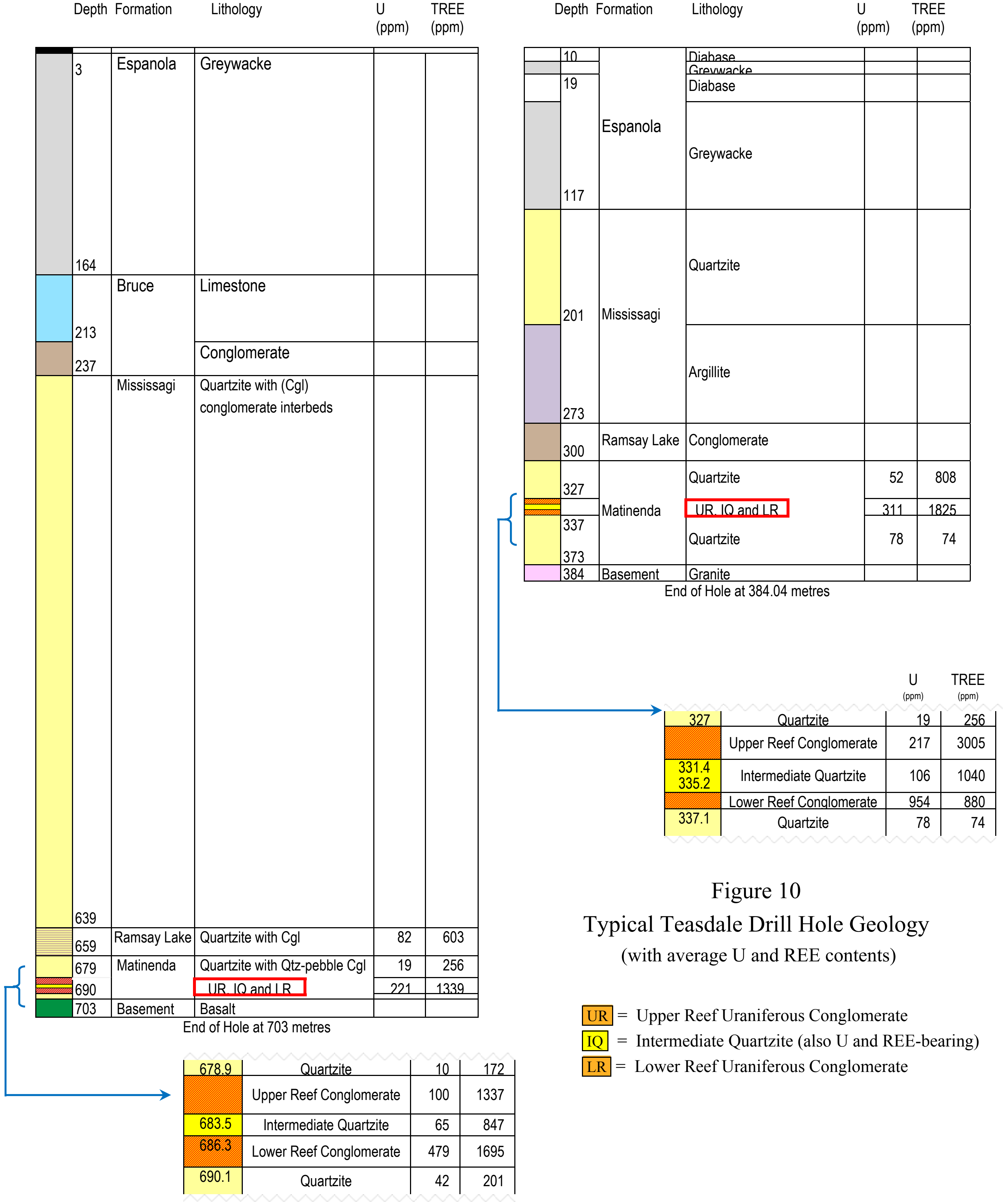
[7] The $7,000,000 budget included a provision for helicopter support for 6,000 m or 40% of the program and included a provision of $120,000 for the construction of ice platforms. The actual amount expended includes the cost of the barge and a tractor for moving the drill.

Graphic Log for Diamond Drill Hole AEC12-08

Graphic Log for Diamond Drill Hole AEC12-09



AEC12-08

Depth	Formation	Lithology	U (ppm)	TREE (ppm)
3 164	Espanola	Greywacke		
213	Bruce	Limestone		
237		Conglomerate		
639	Mississagi	Quartzite with (Cgl) conglomerate interbeds		
659	Ramsay Lake	Quartzite with Cgl	82	603
679	Matinenda	Quartzite with Qtz-pebble Cgl	19	256
690		UR, IQ and LR	221	1339
703	Basement	Basalt		

End of Hole at 703 metres

Depth	Lithology	U	TREE
678.9	Quartzite	10	172
	Upper Reef Conglomerate	100	1337
683.5	Intermediate Quartzite	65	847
686.3	Lower Reef Conglomerate	479	1695
690.1	Quartzite	42	201

AEC12-09

Depth	Formation	Lithology	U (ppm)	TREE (ppm)
10	Espanola	Diabase		
		Greywacke		
19		Diabase		
117		Greywacke		
	Mississagi	Quartzite		
201 273		Argillite		
300	Ramsay Lake	Conglomerate		
327	Matinenda	Quartzite	52	808
		UR, IQ and LR	311	1825
337 373		Quartzite	78	74
384	Basement	Granite		

End of Hole at 384.04 metres

Depth	Lithology	U (ppm)	TREE (ppm)
327	Quartzite	19	256
	Upper Reef Conglomerate	217	3005
331.4 335.2	Intermediate Quartzite	106	1040
	Lower Reef Conglomerate	954	880
337.1	Quartzite	78	74

Figure 10

Typical Teasdale Drill Hole Geology

(with average U and REE contents)

UR = Upper Reef Uraniferous Conglomerate

IQ = Intermediate Quartzite (also U and REE-bearing)

LR = Lower Reef Uraniferous Conglomerate

Intermediate Quartzite ("**IQ**"). A summary of the average intersected grades for each of these horizons in the Appia drill holes is presented in Table 13.

Although quartzite is not the major host to mineralization, it is not unusual for grades in excess of 250 ppm U and 2,000 ppm TREEs to occur in thin, gritty to pebbly quartzite beds up to 25 cm thick in the hanging wall above the main U-bearing horizons. For example, in hole AEC12-12, a 20 cm thick pebbly-quartzite ("**PQ**") bed located 1.35 m above the UR carries 413 ppm U and 6,063 ppm TREEs. Moving upwards and away from the uranium host rocks, these matrix-supported PQ beds become less common and occur farther apart resulting in a gradual lowering of overall (bulk) grades for both U and REEs. These horizons also occur in the footwall section immediately below the main uranium-bearing reefs, and have been accordingly referred to as Hanging Wall Pebbly Quartzite or Footwall Pebbly Quartzite in the company's drill hole logs. Some quartzite sections contain thin horizons that are pyrite-bearing, with the pyrite occurring as fine, scattered grains or as continuous mm-scale seams along bedding laminations.

In most of the Appia DDH's, the UR is poor to faint, and is a mixture of interbedded pebbly quartzite, conglomerate and minor quartzite. According to MacEachern, it was unrecognized and not assayed in most of the historical Conecho logs, and it is not shown on mine plans covering the eastern areas of the Panel Mine adjacent to the Teasdale Deposit. Doug Sprague did not include the UR on his plan of reserve blocks for Conecho. The assaying of drill core for REE mineralization resulted in Appia's discovery of REE mineralization of potential economic significance in most beds of PQ and in quartzite adjacent to the thin conglomerate beds that comprise the UR. The UR is enriched in REEs relative to U (Table 13) and the average ratio of REEs to U is 10.4 : 1. It's average thickness is 4.4 m.

The IQ comprises the section immediately overlying the LR, and it is composed of between two and four metres of quartzite with minor PQ and little or no quartz-pebble conglomerate. It averages approximately 2.7 m in thickness and on rare occasions it exceeds 4 m in thickness, however in hole AEC12-13 the IQ is only 52 cm in thickness. This hole is unusual in that it is also characterized by a strongly mineralized siltstone unit underlying the lower reef which is discussed further in the following text. These variations may be imposed by bedding-parallel slips (faults) that are difficult to recognize in the core. Because of U and REE enrichment in the PQ horizons, the IQ typically contains between 50 and 200 ppm U (average = 99 ppm) and between 500 and 1,500 ppm TREEs (average = 938 ppm) as can be seen in Table 15. In the IQ unit, the REE:U ratio averages 9.1 : 1 in the Appia drill holes. In general, the average grade of the IQ increases with higher PQ contents.

Table 13
Summary of Analytical Data by Drill Hole and by Uranium-Bearing Horizon

Drill Hole Number	Drill Hole Interval			UNIT	Uranium	Rare Earth Concentrations																	Thorium
						Total Rare Earths	Light Rare Earth Elements						Heavy Rare Earth Elements										
	From	To	Length				La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Hf	Y	
	(m)	(m)	(m)		(ppm)	(ppm)	(ppm)	(ppm)	(ppm)	(ppm)	(ppm)	(ppm)	(ppm)	(ppm)	(ppm)	(ppm)	(ppm)	(ppm)	(ppm)	(ppm)	(ppm)	(ppm)	(ppm)
AEC12-01	461.53	465.78	4.25	UR	136	1,773	434	826	77	268	45	1.7	26	3.1	14	2.2	5.6	0.8	4	0.6	6	59	326
AEC12-01	465.78	468.71	2.93	IQ	111	1,202	301	548	54	187	31	1.2	18	2.1	9	1.4	3.7	0.5	3	0.4	4	38	237
AEC12-01	468.71	471.66	2.95	LR	448	2,283	557	1023	101	355	63	2.6	38	4.9	22	3.5	8.7	1.2	7	0.9	4	91	443
AEC12-01a	454.43	456.33	1.90	UR	105	1,925	498	873	89	291	49	1.8	27	3.1	14	2.1	5.2	0.7	4	0.6	6	60	342
AEC12-01a	456.33	459.09	2.76	IQ	128	1,095	283	488	49	160	28	1.1	16	2.1	10	1.5	3.9	0.5	3	0.5	5	43	211
AEC12-01a	459.09	464.42	5.33	LR	366	2,005	502	906	89	303	53	2.2	32	4.0	19	2.9	7.3	1.0	6	0.8	4	74	333
AEC12-02	182.25	187.39	5.14	UR	308	2,817	718	1,277	123	420	73	2.4	47	5.3	24	3.6	9.3	1.3	8	1.1	8	98	512
AEC12-02	187.39	190.09	2.70	IQ	67	732	196	332	33	109	19	0.7	12	1.2	5	0.8	1.9	0.3	2	0.3	2	17	112
AEC12-02	190.09	191.48	1.39	LR	453	855	205	372	38	126	25	2.1	17	2.7	12	2.5	5.5	1.2	5	1.2	3	44	178
AEC12-03	175.17	178.75	3.58	UR	300	2,684	668	1,210	126	414	67	2.5	42	5.3	23	3.7	9.6	1.3	7	1.1	7	93	564
AEC12-03	178.75	180.70	1.95	IQ	95	738	180	340	34	116	19	0.1	11	1.3	6	0.9	2.3	0.3	2	0.3	3	22	154
AEC12-03 *1	180.70	181.31	0.61	LR	50	458	113	207	21	72	11	0.1	7	0.8	4	0.6	1.5	0.2	1	0.2	2	15	117
AEC12-04	556.44	561.46	5.02	UR	223	2,555	656	1,150	116	389	65	2.4	42	4.7	20	3.2	8.0	1.1	6	0.9	8	82	514
AEC12-04	561.46	564.17	2.71	IQ	141	1,171	295	526	54	184	32	1.6	20	2.2	9	1.4	3.5	0.5	3	0.4	3	36	237
AEC12-04	564.17	565.83	1.66	LR	456	1,917	473	845	87	297	53	3.1	35	4.3	20	3.2	7.5	1.0	5	0.7	4	80	405
AEC12-05	215.53	220.37	4.84	UR	304	2,266	576	1,021	98	336	57	2.0	36	4.5	21	3.3	8.7	1.2	7	1.0	7	87	429
AEC12-05	220.37	223.64	3.27	IQ	191	1,159	298	527	51	174	29	1.1	17	2.1	10	1.5	3.8	0.5	3	0.5	3	36	232
AEC12-05	223.64	225.39	1.75	LR	444	794	188	335	34	119	23	1.5	16	2.3	12	2.0	5.1	0.7	4	0.6	2	48	176
AEC12-05b	615.64	619.73	4.09	UR	148	1,815	458	822	79	273	46	1.6	29	3.5	16	2.4	6.2	0.9	5	0.8	8	63	346
AEC12-05b	619.73	622.41	2.68	IQ	58	949	249	430	41	138	22	0.9	14	1.7	8	1.2	3.4	0.5	3	0.5	6	30	168
AEC12-05b	622.41	625.02	2.61	LR	361	2,544	667	1152	111	371	58	2.5	36	4.6	21	3.3	9.1	1.4	9	1.4	16	81	450
AEC12-06	210.35	215.08	4.73	UR	964	10,241	2,647	4,619	460	1,533	256	9.4	151	18.9	84	13.3	33.7	4.6	28	4.1	36	344	1,943
AEC12-06	215.08	216.59	1.51	IQ	50	561	143	254	26	86	14	0.6	8	1.0	4	0.7	1.8	0.3	2	0.2	3	18	114
AEC12-06	216.59	219.39	2.80	LR	142	822	208	369	37	125	21	1.1	12	1.6	7	1.2	3.0	0.4	2	0.4	3	30	163
AEC12-07	354.62	359.16	4.54	UR	168	2,002	518	900	91	302	51	1.9	31	3.4	15	2.4	6.3	0.9	5	0.8	7	68	370
AEC12-07	359.16	362.12	2.96	IQ	86	1,116	296	511	52	169	27	1.2	15	1.6	7	1.0	2.7	0.4	2	0.3	4	28	205
AEC12-07	362.12	364.75	2.63	LR	996	791	172	314	34	116	25	2.0	19	3.1	17	2.9	7.5	1.0	6	0.8	3	69	234
AEC12-08	679.17	683.52	4.35	UR	106	1,386	362	619	63	206	33	1.4	22	2.6	11	1.8	4.6	0.7	4	0.6	6	49	263
AEC12-08	683.52	686.26	2.74	IQ	65	847	221	383	39	128	20	0.9	13	1.5	6	1.0	2.6	0.4	2	0.3	4	24	178
AEC12-08	686.26	690.08	3.82	LR	479	1,695	427	741	77	254	45	2.1	30	4.1	19	3.0	7.6	1.0	6	0.8	3	75	358

Drill Hole Number	Drill Hole Interval			UNIT	Uranium	Rare Earth Concentrations																	Thorium
						Total Rare Earths	Light Rare Earth Elements						Heavy Rare Earth Elements										
	From	To	Length				La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Hf	Y	
	(m)	(m)	(m)		(ppm)	(ppm)	(ppm)	(ppm)	(ppm)	(ppm)	(ppm)	(ppm)	(ppm)	(ppm)	(ppm)	(ppm)	(ppm)	(ppm)	(ppm)	(ppm)	(ppm)	(ppm)	(ppm)
Summary of Analytical Data by Drill Hole and by Uranium-Bearing Horizon - Continued																							
AEC12-09	327.33	331.37	4.04	*UR*	*217*	3,005	778	1,353	134	451	75	2.4	47	5.2	24	3.7	9.1	1.2	7	1.1	10	102	552
AEC12-09	331.37	335.24	3.87	IQ	96	946	245	428	43	142	23	0.9	15	1.6	7	1.1	2.9	0.4	2	0.4	4	29	187
AEC12-09	335.24	337.07	1.83	LR	954	880	197	358	38	127	27	2.1	21	3.3	17	3.0	7.6	1.1	6	0.8	2	70	239
AEC12-10	514.52	519.30	4.78	UR	187	1,959	515	887	84	283	48	1.6	28	3.3	16	2.5	6.7	0.9	5	0.8	7	70.1	379
AEC12-10	519.30	522.85	3.55	IQ	77	659	173	294	30	99	16	0.7	11	1.2	6	0.9	2.3	0.3	2	0.3	3	21.9	132
AEC12-10 *2	522.85	523.13	0.28	LR	394	1,101	279	461	49	162	29	1.7	22	3.0	15	2.5	6.2	0.9	5	0.7	3	60	209
AEC12-11	395.18	399.31	4.13	UR	256	2,391	600	1,088	105	360	62	2.1	40	4.5	21	3.2	8.2	1.1	7	1.0	6	83.8	425
AEC12-11	399.31	401.94	2.63	IQ	181	1,386	348	637	60	211	36	1.4	21	2.4	11	1.7	4.3	0.6	4	0.5	3	44.8	262
AEC12-11	401.94	405.07	3.13	LR	355	1,311	317	590	56	200	36	1.6	22	2.8	13	2.1	5.6	0.8	5	0.6	3	54.4	260
AEC12-12	563.82	566.47	2.65	UR	316	2,399	592	1,075	109	343	63	3.0	37	4.5	20	3.0	7.9	1.1	6	10.2	5	126	547
AEC12-12	566.47	569.67	3.20	IQ	71	656	168	296	29	97	16	0.9	10	1.2	6	0.9	2.4	0.3	2	0.3	3	23.2	122
AEC12-12	569.67	575.11	5.44	LR	431	1,497	374	649	64	215	40	2.8	29	3.9	19	3.0	7.7	1.0	6	0.8	5	76.6	380
AEC12-13	444.72	448.02	3.30	UR	206	2,240	562	1,036	99	336	54	1.9	33	3.9	17	2.7	6.7	0.9	5	0.8	8	72.5	449
AEC12-13	448.02	448.54	0.52	IQ	50	521	134	230	22	74	13	0.5	9	1.2	5	0.8	2.1	0.3	2	0.3	4	24	111
AEC12-13	448.54	452.43	3.89	LR	474	4,099	1,022	1,893	183	623	101	3.4	62	7.4	32	4.9	12.5	1.6	9	1.4	12	131	831
AEC12-13	452.43	453.56	1.13	LS	1144	3,062	705	1,344	133	463	84	5.9	60	8.6	40	6.6	16.8	2.2	12	1.8	15	165	772
AEC12-14 *3	*not sampled*																						
AEC12-15	617.20	621.89	4.69	UR	164	1,780	462	808	79	261	44	2.1	28	3.8	15	2.8	6.6	1.3	6	1.2	5	59.3	342
AEC12-15	621.89	625.19	3.30	IQ	56	650	166	297	29	98	16	0.6	10	1.1	5	0.8	2.0	0.3	2	0.3	3	19	131
AEC12-15	625.19	628.60	3.41	LR	335	672	167	287	30	98	18	1.1	13	1.8	9	1.5	4.0	0.5	3	0.4	2	35.6	151
AEC12-16	392.18	397.33	5.15	UR	292	2,687	702	1,209	120	398	68	2.4	42	4.9	22	3.3	8.3	1.1	7	1.0	7	89.9	491
AEC12-16	397.33	399.49	2.16	IQ	160	1,565	424	694	72	235	39	1.5	24	2.6	12	1.7	4.4	0.6	4	0.5	4	45.9	286
AEC12-16	399.49	402.77	3.28	LR	309	1,984	521	874	92	301	52	2.2	32	3.9	18	2.7	6.7	0.9	5	0.8	4	68.2	365

Notes:

1) In drill hole AEC12-03, the Lower Reef was poorly developed.

2) In drill hole AEC12-10, the Lower Reef was displaced by a swarm of lamprophyre dikes and only the uppermost 28 cm of the lower uranium-bearing horizon was present in this hole – some fault dislocation is possible.

3) In drill hole AEC12-14, the Upper and Lower Reefs and the Intermediate Quartzite were displaced by a dike and so were not present in this hole.

The LR is the main carrier of uranium, and it is this reef that was exploited in the Elliot Lake uranium mines. Most of the Appia holes intersected a LR ranging in thickness from 1.4 m to 5.4 m, and the average length of the LR in the Teasdale drill holes was 2.95 m. The true thickness of the LR is approximately 2.7 m and it averages 438 ppm U and 1,512 ppm TREEs. In holes where the geology shows the reef to be poorly developed or compromised by intrusive rocks, such as in holes AEC12-03 and AEC12-10, grades may be substantially lower. In well developed LR, the REE:U ratio is 3.4: 1.

10.3 DISCUSSION OF DRILLING RESULTS

Appia's continuing drilling of the Teasdale area has extended the area of known mineralization beyond the limits imposed by the company's first phase of drilling. A typical composited vertical section that is approximately normal to the strike is presented in Figure 11. The earlier drilling confirmed the results reported from the original Conecho drilling carried out during 1954-55 and provided some new geological and assay data between the older holes. Appia's initial drilling program also confirmed as "reasonable" an earlier resource estimate by Rio Algom's Chief Geologist Doug Sprague, P.Eng., and provided a basis whereby Appia disclosed a modest NI 43-101 compliant resource estimate for the Teasdale deposit as reported by WGM (Workman and Breede). The drilling carried out by Appia in 2012 substantially extended the area of known mineralization by completing a significant part of the diamond drilling previously recommended by WGM. The average grade and intersected thickness of the UR-IQ-LR unit is shown for each hole. As is shown, the mineralized zone is open down-dip.

All of the Appia holes completed during the 2012 drilling programs intersected the uranium-REE bearing horizons in the lower part of the Matinenda Formation, namely the UR, IQ and LR horizons. The total length of the mineralized sections in the drill holes varied from 8.6 m to 11.4 m in the recent drill holes. The grades-thicknesses encountered by Appia were more or less the same as those intersections taken from historical records, although complete comparisons are impossible since the upper reef was not recognized as such at the time, and the original holes were assayed only for uranium. The Appia drilling does show that the mineralized zone extends to the boundary of the company's claims and is probably contiguous with the mineralization that remains in the old Panel Mine workings.

WGM carried out a new mineral resources estimate to provide an up-date to its earlier estimate reported by Workman and Breede. This estimate, and the parameters on which it is based are contained in Section 17 of this report.

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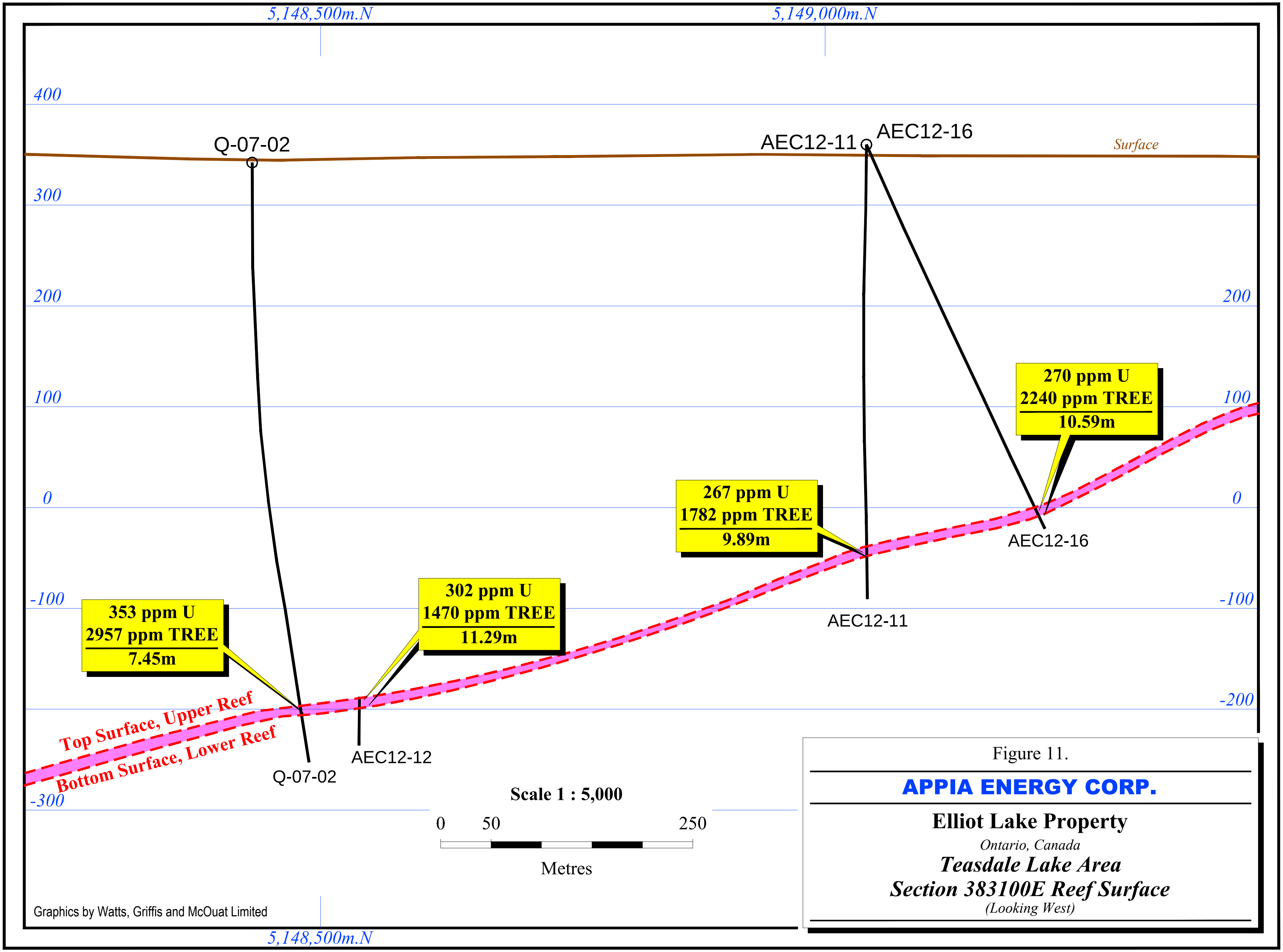

Figure 11.

APPIA ENERGY CORP.

Elliot Lake Property

Ontario, Canada

Teasdale Lake Area

Section 383100E Reef Surface

(Looking West)

Watts, Griffis and McOuat

11. SAMPLING METHOD AND APPROACH

During the drilling program, uranium-bearing intervals were delineated on the basis of diagnostic radiometric signatures as measured with a hand-held RS-125 Super-SPEC portable gamma ray spectrometer manufactured by Radiation Solutions of Mississauga, Ontario, Canada. The specifications and capabilities of this instrument are described in Section 9.2 of this report. It is important to understand that the equivalent potassium, uranium and thorium data provided by portable spectrometers allow insight into the elemental make-up of a radioactive source, but they do not provide analytical data. Such data can only be accurately provided through conventional geochemical analysis under controlled conditions in a laboratory. Equivalent metal data is calculated based on statistical algorithms integral to the instrument's software, and the accuracy of such data is influenced by the manner in which the instrument is used, its performance, ambient conditions and operator experience. Radiometric data was used only as a guide in selecting intervals to be sampled and assayed.

After geologically logging the core, the mineralized sections were delimited for sampling by permanent marks across the core and a cutting line was added to ensure consistent sampling. Individual samples were selected based on geology such that conglomerate horizons, pebbly quartzite beds and intervals of quartzite were sampled separately. No sample exceeded one metre in length, and samples as short as 15 cm were taken. The core was cut in half using a diamond saw, with one half of the core being placed in the sample bag with a pre-numbered tag and sealed, and the other half returned to the core tray with a matching sample tag as a permanent archival record. Plate 10 shows a core tray with pre-arranged sample tags prior to cutting.

Appia's 2012 drilling program generated 1,211 routine drill core samples. In addition, 69 duplicate samples, 47 certified standards and 27 blank samples were inserted into the sample stream and blindly submitted to the laboratory for quality assurance/quality control ("**QA/QC**") purposes. All samples were analysed as batches and the lab was not aware of the QA/QC samples. Three different certified standards from the CANMET[8] and CDN Labs[9] of Burnaby, B.C. were used (DL 1A, UTS-4 and BL-3).

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[8] Canada Centre for Mineral and Energy Technology – Energy Mines and Resources Canada; Uranium Tailings Reference Materials.

[9] CDN Laboratories Ltd., 10945-B River Road, Delta, B.C., Canada, V4C 2R8; Phone No.: 604-540-2233

The field blank was developed using essentially barren country rock. Control samples constituted approximately 10.6% of the samples submitted by Appia. This QA/QC program was in addition to the internal control program carried out by the laboratory.

Plate 10: Two core trays with sample tags inserted at selected intervals prior to cutting.

Plate 11: Close-up of core tray with sample tags inserted at selected intervals and cutting line marked prior to cutting.

12. SAMPLE PREPARATION AND SECURITY

For reasons cited herein, no surface sampling has been carried out by Appia on the Property, and all samples submitted for analysis have been derived from diamond drill core.

The Appia drill core samples were submitted to Activation Laboratories ("**Actlabs**") of Ancaster, Ontario. Actlabs is a fully accredited geochemical laboratory that meets both ISO/IEC 17025 with CAN-P-1579 standards as recommended by the Toronto Stock Exchange-Ontario Securities Commission mineral standards taskforce.

On receiving the samples, Actlabs dried and crushed the entire core sample to a nominal 85% passing a #10 mesh screen, before repeated riffle splitting of the crusher product to generate an aliquot of approximately 250 g. The subsample was then pulverized to a nominal 95% passing a #200 mesh screen using a ring and puck pulverizer to ensure that REE minerals were sufficiently reduced to ensure complete fusion during the assaying technique. Cleaner (wash) sand was used between each sample to prevent carry-over.

Most elements including the REEs, uranium, thorium and silver, were determined in accordance with Actlabs' fusion-mass spectrometer technique; some trace elements and the major oxides used the fusion-ICP technique (both Code 8 analyses).

The Code 8 Nb-Zr-Y-Ta-U-Th-Phosphate assay package was used to analyze for these elements by fusion with lithium metaborate/tetraborate in platinum crucibles with the molten glass cast into a glass disc. These glass disks were then analyzed on a Panalytical Axios Advanced wavelength dispersive XRF instrument. REEs and major oxides were determined separately using an ICP and ICP/MS instrument after a lithium metaborate/tetraborate fusion. Mass balance is required as an additional quality control technique and elemental totals of the oxides should be between 98 to 101%. Although an acid digestion can be used for REEs, it is not advised since REEs can occur as resistate minerals meaning that any such digestion will produce a partial analysis reflecting only acid-soluble REEs.

Previously, the analysis of Appia's samples for uranium was primarily by Actlabs' Code 5D which uses neutron activation and delayed neutron counting (DNC). Approximately one gram of sample was weighed into a polyethylene capsule which in turn was sealed into a carrier vial for neutron irradiation within a slowpoke nuclear reactor. The sequentially irradiated samples are transferred automatically to the BF3 counting array detector using a computer automated system. Calibration is achieved with certified reference materials. All elements in the sample absorb neutrons which produce a subsequent emission that can be used to measure the composition of the sample using an array of BF3 neutron detectors. This technique, more generally referred to as neutron activation analysis, is ideal for measuring uranium and many other trace elements from sub-ppm to percentage levels. The method does have limitations as certain interferences can occur. It measures total metal content which may not be relevant in the sense of mineral economics, for example, it measures total uranium rather than soluble uranium. While the difference may be trivial in most geological environments, DNC analysis may include non-recoverable uranium that is contained in the crystal lattice of resistate minerals such as zircon. Fortunately, the Elliot Lake mineralization contains no significant amount of refractory minerals and thus the uranium concentration database should be unaffected by mineralogically induced assay variability.

A few samples were analysed for gold using an Actlabs Code 1A2 procedure which is a conventional 1050°C fire assay on a 30 g charge with an atomic absorption instrumental finish giving a 5 ppb lower detection limit (the upper limit is 3,000 ppb). Normally, any sample exceeding the upper limit of 3,000 ppb is reanalyzed using a gravimetric finish in which the prill is weighed.

The Appia geologist retained possession of samples until they were delivered to the courier for shipping to the lab.

All the split cores are currently being stored in core racks that are inside a locked building in the town of Elliot Lake. The un-split cores, which are not generally mineralized or of importance from a resource standpoint, are being stored outside the building, cross-stacked, in a fenced area. Sample intervals from the drill program are permanently recorded in drill logs combined with the assay results.

13. DATA CORROBORATION

13.1 OVERVIEW

Appia's exploration work has been under the management of Alan MacEachern who provided independent QP oversight for the Appia exploration program as well as core logging and sampling. WGM visited the project site during 15 July, 2013 and confirmed drill hole locations and briefly examined the drill core which had been extensively sampled both for routine analysis and for metallurgical testing.

Although the WGS-84 datum normally be used for the Elliot Lake project, Appia uses the NAD-27 (Canada) datum all location measurements to maintain consistency with the Ministry of Northern Development and Mines ("**MNDM**") which uses this datum for its records of drill hole locations stored in its ERMES database. During the previous Appia drilling program, WGM used the NAD-27 datum for checking drill hole locations. In comparing the use of WGS-84 and NAD-27, WGM has found that the physical difference in co-ordinates between the two datums was 3-5 metres as measured on the ground. The GPS Utility software licensed to WGM converted between the two datums and between geographic co-ordinates and UTM co-ordinates with no significant variance after the data is discounted for the estimated position error. In Workman and Breede (2009), WGM reported on inaccuracies in the MNDM co-ordinates for previously drilled holes and Appia's success in resurveying the locations of the on-shore historical holes drilled by Conecho in 1954-55.

As part of its data corroboration efforts, WGM secured original assay certificates for the Appia drill core samples from Actlabs. WGM carried out a detailed audit of the project's assay database and found no errors that would have compromised the resource estimate. WGM questioned the accuracy of one down-hole survey value which was in error in the drill hole database and this was corrected after referring to the original survey data.

In confirming the locations of drill sites, WGM measured the locations of the holes shown in Table 14 which are compared with the original measurements. In general the WGM co-ordinates compared favourably with the original co-ordinates, and in no cases where the variances sufficient to significantly influence the Teasdale resource estimate.

Table 14
WGM Confirmation of Selected Diamond Drill Hole Locations, 2012 Teasdale Drilling

Drill Hole	Geographic and UTM Co-Ordinates						
	Original Location			WGM Measurement			Difference (metres)
	Latitude	Longitude	Elevation	Latitude	Longitude	Elevation	
AEC 12-02	46° 29’ 20.51” N	82° 31’ 15.04” W	337	46° 29’ 20.6” N	82° 31’ 14.8” W	346	5.8
AEC 12-04	46° 28’ 55.92” N	82° 31’ 53.61” W	342	46° 28’ 56.0” N	82° 31’ 53.6” W	352	5.0
AEC 12-05	46° 29’ 20.35” N	82° 31’ 14.89” W	340	46° 29’ 20.4” N	82° 31’ 14.8” W	350	2.6
AEC 12-06	46° 29’ 13.78” N	82° 30’ 57.03” W	320	disturbed drill-site – exact hole location uncertain			+/- 5
AEC 12-07	46° 29’ 11.87” N	82° 30’ 56.97” W	347	46° 29’ 12.1” N	82° 30’ 57.0” W	352	7.1
AEC 12-09	46° 29’ 11.87” N	82° 30’ 56.97” W	347	46° 29’ 12.2” N	82° 30’ 56.9” W	353	10.3

13.2 BEDROCK SAMPLING

As mentioned in the foregoing text, the mineralized conglomerates in the Matinenda Formation rarely outcrop and do not outcrop on the Appia property. No amount of surface sampling can provide Appia with any answers concerning the uranium-bearing conglomerates that have been intersected on the Appia property.

Appia previously followed WGM’s recommendation for a Phase 1 drilling program intended to corroborate some of the historical drill holes in the Teasdale Lake Zone. This work was successful in confirming the original intersections, and while some variations were encountered, no discrepancies were found such that uranium mineralization was absent where formerly reported. In addition, the thicknesses of the mineralized zones were comparable (Workman and Breede). On this basis, Appia was confident in the outcomes of the follow-up 2012 drilling program which is the subject of this report.

13.3 EVALUATION OF LABORATORY PERFORMANCE

WGM previously discussed Appia's QA/QC program and laboratory performance relating to the earlier drilling programs carried out by Appia at Teasdale Lake and Banana Lake (Workman and Breede). In that review, WGM found that four standards failed, falling outside of what it would accept as a normal range of values. Actlab's internal checks that were inserted at the time of analysis performed well, so it is possible that the failures represent anomalies within the standard (as unlikely as that may seem), or some form of contamination that occurred during sample preparation. These findings were discussed with the initial Appia project geologist, Sonny Bernales, and it was agreed that additional tests would be made in the future on any samples associated with unusual assays of such standards. This resulted in a doubling of the frequency that Appia inserted standards and other quality control samples in the sample stream submitted to Actlabs.

Workman and Breede also reported on a set of WGM check samples collected previously which showed reasonably good overall correlation to the assay results reported originally. WGM also cautioned on need for careful sample preparation because uranium mineralization tends to occur as brittle, heavy mineral grains (some microscopic) that may be difficult to properly homogenize within granular or pulped samples. Care is required in sample preparation and any process involving the subdivision of samples as the slightest vibration can result in the settlement of heavy mineral particles.

The results of Appia's most recent QA/QC program are illustrated in the following graphs. Figure 12 illustrates the comparison between duplicate samples. Assays are shown for U, Th and TREEs. The first set, field duplicates, is derived from re-cutting the normal half-core sample of drill core into two equally sized quarter-core samples which are submitted blindly to the lab under different sample numbers. The second set, lab duplicates, is produced internally by the Actlabs sample preparation crew by creating a second sample from one sample in each group of +/-25 samples submitted by the client. These samples serve as an internal quality control measure along with certified standards. The results show some outliers which triggered re-analysis at the laboratory and the results were inconclusive, pointing to an irregular distribution of U or REE minerals in the sample, and not to laboratory error.

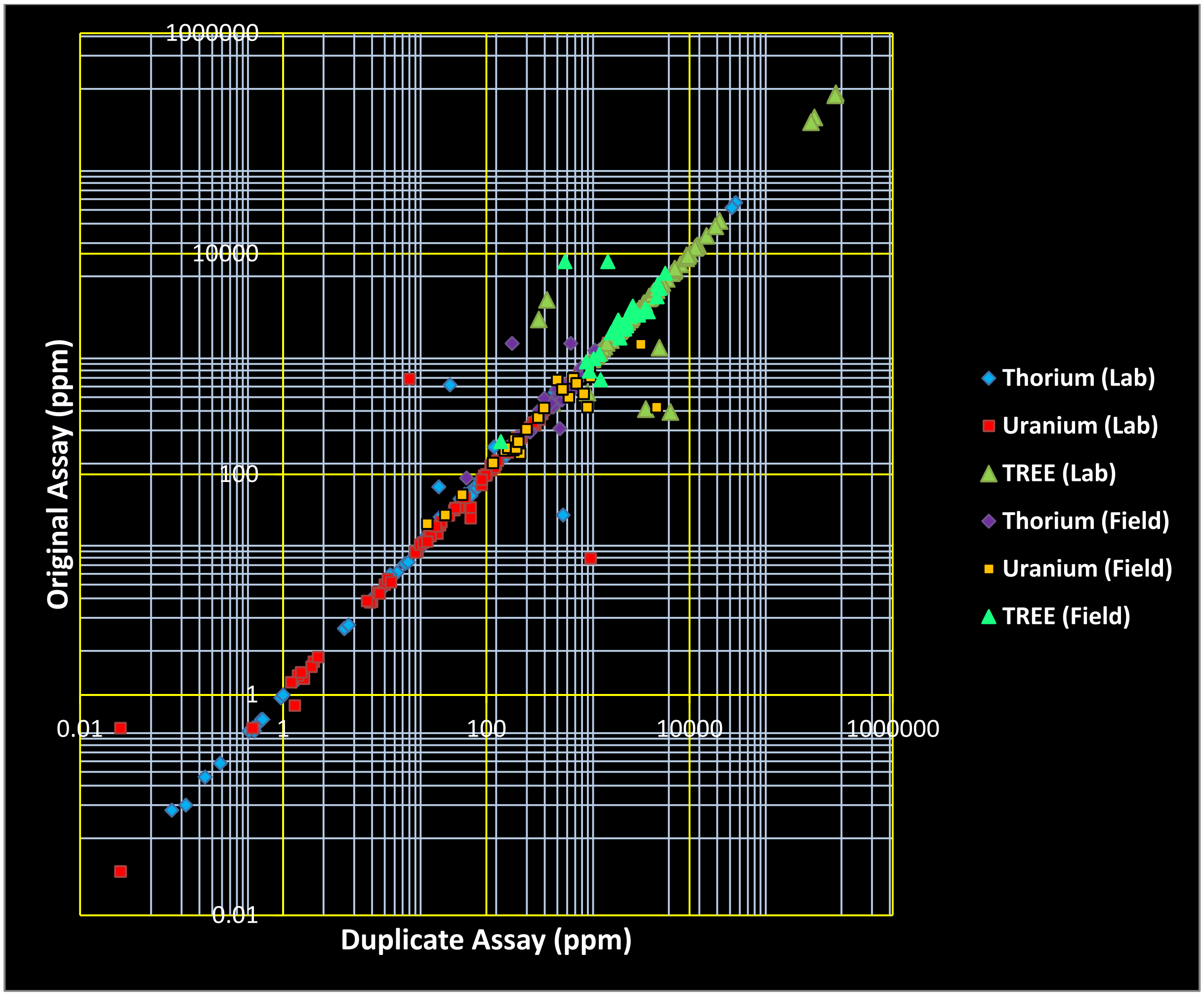

Figure 12: The graph shows the correlation between the assays on duplicate samples, both laboratory duplicates and field duplicates, illustrating little or no systemic differences in the two populations. Outliers exist in both populations which WGM attributes to variations in the amount of mineralization within the samples rather than lab error.

Appia's use and assaying of blank samples resulted in the maximum, minimum and average values shown in Table 15. While the uranium, thorium and rare earth content of the blanks is not zero, the values have assayed within a narrow range that is sufficiently low to provide a meaningful guide as an expression of laboratory performance. The high REE value detected in one sample (#W1381775) skewed the average grade of the entire blank sample population as it contained 74 ppm Ce, 177 ppm La, 22 ppm Dy, 91.5 ppm Nd and 17.7 ppm Sm, all values that significantly higher than normal (4-7 x average). This sample was analysed after several samples that were weakly mineralized in REEs, however the level of enrichment was relatively low and thus carry-over contamination cannot explain the relatively high values in the blank. The standards used by both the laboratory and those inserted by Appia for this group of samples also analysed within very close tolerances of the certified values. Appia's two inserted standards that accompanied this group of samples assayed within 2-3% for the

REEs and were within less than 1% for uranium and thorium. These results suggest that Appia's blanks are imperfect, with some carrying a range of REE values that reduce their usefulness as control samples.

Table 15

Comparison of Appia Blanks and Average Teasdale Grade

Sample Averaged	Uranium	Thorium	TREE
	(all values in ppm)		
Appia Blanks – Minimum Grade	1	3	20
Appia Blanks – Maximum Grade	33	43	440
Appia Blanks – Average Grade	11	20	112
Average Teasdale Grade [1]	237 / 277	280 / 307	1570 / 1650

NOTE: Grades shown are for Indicated Resources and Inferred Resources.

WGM concluded that Appia's control samples generally supported the high-quality performance of the laboratory and that better blank source material should be acquired for QA/QC purposes. In terms of WGM's experience from many other exploration projects, including uranium, base metals, REEs and gold, Appia's field duplicates performed no better and no worse than other duplicate sample populations. WGM's experience has shown that field duplicates are generally a poor way to test laboratory performance because it is rare that a true duplicate sample can be produced from the opposite sides of the drill core[10].

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[10] WGM believes that the only good way to achieve a field duplicate is to process (finely crush and/or pulverize) the core in the field, and then to use a laboratory quality sample divider to split the sample into two equal halves. This demands care and expertise since improper use of a sample divider can introduce heterogeneity into the samples.

14. ADJACENT PROPERTIES

Under NI 43-101 and in the context of this report, an "Adjacent Property" is a mineral property with a boundary reasonably close to the boundary of the Appia Elliot Lake area property.

An adjacent exploration property is held by Pele Mountain Resources Inc. ("**Pele**") of Toronto, Canada. The Pele property comprises a 100-percent interest in 313 mining claims covering more than 12,500 acres near Elliot Lake and covering portions of Rio Algom's past-producing Lacnor Mine property where the Pardee reef was identified. Pele has referred to this as its Eco Ridge Mine Uranium Project ("**Eco Ridge Project**"). Pele's ownership is held via its wholly owned subsidiary, First Canadian Uranium Inc. ("**FCU**"), and is subject to a 1.75% net smelter royalty that is owned by CanAlaska Uranium Ltd. on the Pardee Claim Group, of which Pele may buy back 1-percent for $1-million. The location of the Eco Ridge Project is shown in Figure 13.

The Pele property has been extensively drilled with more than 100 historic drill holes completed since 1953 by a number of companies including McIntyre Porcupine Mines Ltd. which was the initial discoverer of uranium mineralization in this area on claims it had optioned from Aquarius Porcupine Mines Ltd. Aquarius subsequently constructed a 30-metre adit for sampling purposes. New Jersey Zinc Exploration Co. also drilled a few holes in the area as did St. Mary's Uranium Mines Limited, Stancan Uranium Corp., Algom Uranium Mines and many others. The data was compiled by Rio Algom in 1977 after it acquired the exploration rights to this area. Pele completed a single 224 m hole during 2006.

Based essentially on the historical work, Scott Wilson Roscoe Postle Associates Inc. ("**SW-RPA**") released a NI 43-101 compliant report dated 15 January, 2007 (available on SEDAR) for the property which contained a mineral resource estimate that totalled 30.05 Mt grading 0.05% U_3O_8, in the Inferred Mineral Resource category for a total inferred uranium content of 33.05 Mlbs of U_3O_8.

SW-RPA completed a Preliminary Assessment during late 2007 and its report entitled "Preliminary Assessment on the Elliot Lake Project, Ontario, Canada Prepared for Pele Mountain Resources Inc." by Cochrane et al (2007) and dated 3 October, 2007 was filed on SEDAR. Under NI 43-101, feasibility studies are not allowed on Inferred Resources, however a company may complete a Preliminary Assessment which is defined as an economic analysis carried out to investigate the potential viability of mineral resources at an early stage in a project. Also known as "Scoping Studies", reviews such as these can answer questions concerning the need for a project to discover additional resources at the same grade, or alternatively, whether the company requires higher grading resources to meet viability criteria. The SW-RPA study was based on the previously estimated mineral resources (0.030% U_3O_8 cut-off grade, 2.70 specific gravity), 10% dilution, a US $55.00 per pound U_3O_8 commodity price and an exchange rate of C$1.00 to US$0.90 (Cochrane et al, 2007). The mining method selected was based on a combination of panel drifting and horizontal long-hole slashing with approximately 60% of the ore treated in place by underground bioleaching and 40% of the ore hauled to surface by ramp for conventional milling and treatment in an acid-leach plant. A 3,214 tonne per day production rate was used in the study with ore averaging 0.045% U_3O_8 over an 18 year mine life. Uranium recovery was assumed to be 90% by conventional milling and 70% by bioleaching. The study concluded:

- capital costs related to project development would be C$195 million;
- on-going capital costs would add another C$63 million over the 20-year life of the project, including the rehabilitation period);
- operating costs per pound of U_3O_8 produced over the life of the project would be US$55.51;
- costs associated with decommissioning would total C$31 million;
- based on a commodity price of US $95.00 per pound of U_3O_8, the project generated gross revenue of C1.5 billion and had a net present value ("**NPV**") of C$363.5 million using a zero discount rate and C$41 million using a 10% discount rate, both before taxes;
- at the 10% discount rate, the project had a pre-tax internal rate of return ("**IRR**") of 13%; and,
- a US$5 increase in the commodity price would increase the IRR to 15%.

SW-RPA recommended that Pele increase the density of drilling using a maximum 200 m by 200 m hole spacing which was chosen as sufficient to increase the confidence level of the mineral resources and allow Inferred Resources to be converted to Indicated Resources, and thus able to support a feasibility study.

Based on recommendations from SW-RPA, Pele completed follow-up drilling during late 2007 and early 2008 which expanded a "higher grading" zone within the deposit. Pele's press release on the subject dated 25 January 2008 reported uranium grades ranging from 0.034% U_3O_8 (0.68 lbs U_3O_8/ton) over an estimated true width of 2.20 m to a high of 0.080% U_3O_8 (1.60 lbs U_3O_8/ton) over an estimated true width of 2.66 m. The two widest intersections were both estimated at 2.92 m true thickness and these had uranium oxide contents of 0.060% and 0.070%, respectively (1.20 and 1.40 lbs U_3O_8/ton). In the same press release, Pele updated its Elliot Lake uranium resources to 6.3 Mt of Indicated Resources averaging 0.051% U_3O_8 and 41.0 Mt of Inferred Resources averaging 0.044% U_3O_8 for a total contained resource of "42 million pounds of NI 43-101 compliant U_3O_8 resources". WGM cautions that the adding of inferred and indicated resources is not allowed under NI 43-101 rules and CIM Standards and Guidelines, nevertheless the numbers do show the magnitude of the potential resources on the Pele's Elliot Lake claims. An additional press release on 6 March, 2008 showed additional intersections having the same tenor of mineralization as those holes released previously.

On 1 May 2008, Pele announced that it would initiate a preliminary feasibility study based on the positive results of its scoping study. A subsequent MD&A document completed on 27 May 2008 and filed on SEDAR, provides a summary of the SW-RPA scoping study and reiterates that the sufficient resources were found to support an 18-year mine life producing 826,000 lbs of U_3O_8 per year at a cash operating cost of US $55.51 per pound.

Pele initiated site characterization studies base line environmental studies in preparation for undertaking a Environmental Impact Assessment. Discussions centering on advancing the project towards development are on-going with the Canadian Nuclear Safety Commission, the town of Elliot Lake and Serpent River First Nation representatives.

On 9 September, 2008 Pele announced that it had submitted the project description for its now named Eco Ridge Mine at Elliot Lake to the federal government's major projects management office and the Canadian Nuclear Safety Commission (CNSC). This move initiated the official permitting process for Pele's planned uranium mine and processing facility to be located near Elliot Lake. Pele plans for mining, processing and waste management to make innovative use of proven technologies to build a new facility that will be significantly more advanced and environmentally-friendly than historic operations in the Elliot Lake region. Mining will be accomplished using ramps from surface, trackless development and long-hole slashing. Underground leach cells and surface heap leach cells will be designed to fully contain the leach solutions and to allow for progressive decommissioning. The news release states that no tailings pond will be required at the Eco Ridge Mine. The project description provided target dates for completion of the licensing and permitting activities by year-end 2010, the beginning of construction in early-2011 and the commencement of uranium production in late-2012.

APE REP / APE_05_Rgnl_Geol.dwg (Layout: 250k L_P)
Last revision date: Thursday 25 July 2013

Watts, Griffis and McOuat

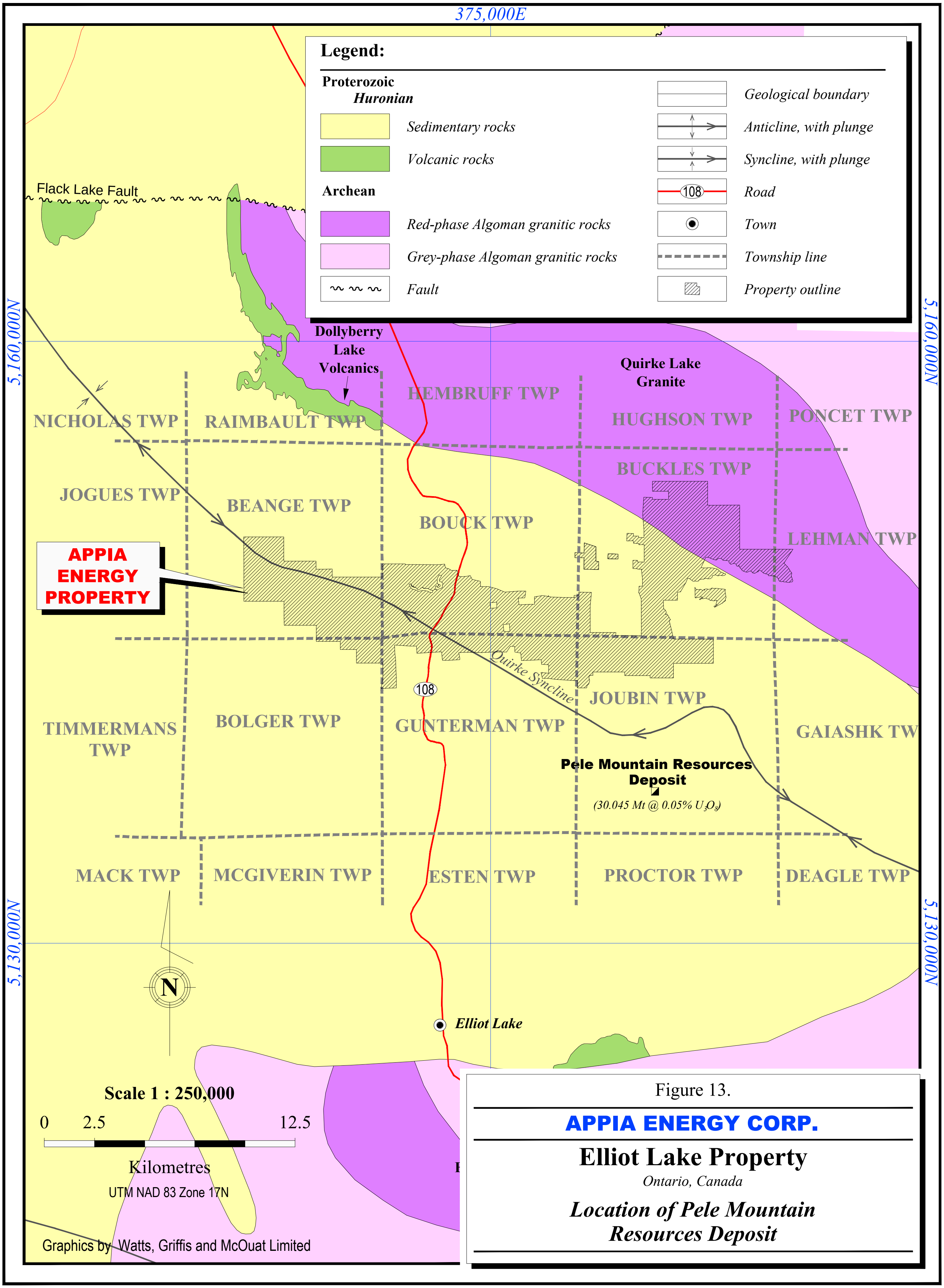

Figure 13.

APPIA ENERGY CORP.

Elliot Lake Property

Ontario, Canada

Location of Pele Mountain Resources Deposit

A follow-up press release on October 28, 2008, announced measures in response to market conditions, including postponement of the pre-feasibility study and certain components of the permitting process. Pele proposed underground uranium mining with uranium processing through a combination of underground bioleaching and surface heap leach extraction at the Eco Ridge Project. Under Pele's plan, approximately 35% of mined ore would be trucked to surface for heap leaching.

A Pele new release dated 27 July, 2009 refers to the project containing "a NI 43-101 compliant resource of 6.4 million pounds of "indicated" U_3O_8 (5.68 million tonnes grading 0.051-percent U_3O_8) and 36.1 million pounds of "inferred" U_3O_8 (37.26 tonnes grading 0.044-percent U_3O_8) with the potential for significant expansion." These grades are equivalent to 1.02 lbs U_3O_8 per ton in the Indicated Resources class and 0.88 lbs U_3O_8 per ton in the Inferred Resources class.

During July, 2009, FCU signed an agreement with the City of Elliot Lake in respect to the purchase of the surface rights to a key group of 48 patented mining claims (796 ha) that are part of the Eco Ridge Project. The surface rights covered by the Lease include areas planned for mine portals and other surface plant, equipment and related infrastructure.

On 7 October, 2009 Pele announced new Rare Earth Element ("**REE**") analytical data from selected drill intersections confirming widespread REE mineralization with the uranium at its Eco Ridge Mine uranium project. As is known from historical records, the Elliot Lake mines were historical producers of significant amounts of yttrium as a by-product of uranium production. The mineralization was prevalent within the uranium-bearing conglomerates. The average leach extraction of heavy REEs plus yttrium averaged 64%, and Pele concluded that these are sufficiently recoverable to add to the economic value of the uranium resources.

This Eco Ridge resource estimate was up-dated in a report dated 5 April, 2011 by RPA, formerly known as SW-RPA, that the deposit contained Indicated Resources of 14.31 Mt grading 0.048% U_3O_8 (0.96 lbs U_3O_8 per ton) and 0.164% total rare earth elements ("**REE**"s) or 3.28 lbs/ton with additional Inferred Resources of 33.12 Mt grading 0.043% U_3O_8 (0.86 lbs U_3O_8 per ton) and 0.132% total REEs or 2.64 lbs/ton (Ciuculescu, 2011). The total contained metal was 15.2 million pounds of U_3O_8 and 51.9 Mlbs of REEs in the Indicated category and 31.4 Mlbs of U_3O_8 and 96.4 Mlbs of REEs in the Inferred category. The resources were based on a cut-off grade of 0.028% U_3O_8 and a long term uranium price of $60 per pound of uranium oxide (the current price is stable at $68).

In July, 2011, Pele announced the results of a new Preliminary Assessment for the Eco Ridge Project, including these key findings based on a 9,400-tonne/day operation with 14-year life-of-mine production of 10.7 Mlbs of total rare earth oxides (REOs) and 24.9 Mlbs of U_3O_8:

- cumulative operating cash flow of US$1.72-billion
- cumulative pre-tax cash flow of US$1.31-billion
- operating cash cost of US $16 per pound U_3O_8, net of REO credits
- start-up capital costs of US $212 M and sustaining capital costs of US $195 M.
- positive NPV of $533 million (at a 10% discount rate)
- internal rate of return (IRR) of 47 percent (47%)

On the basis of the foregoing analysis, there is no question that with the right market fundamentals, the remaining uranium-REE deposits in the Elliot Lake basin represent a viable long-term resource that can be extracted with a very robust rate of return on investment.

Since the PEA, Pele's continued drilling of its Eco Ridge project has yielded results that the company views as positive. On 10 June, 2013, Pele announced an 11% increase in the Indicated Resources at Eco Ridge to 22.7 Mt at an average grade of 0.045% U_3O_8 (0.90 lbs U_3O_8/ton) and 1,606 ppm total REOs (Table 16). The amount of contained metal increased 10% to 22.65 Mlbs of U_3O_8 and 80.5 Mlbs of REOs. Most of Pele's drilling was done to enlarge the area of known mineralization and was insufficient to classify this mineralization as Indicated. Nevertheless, the mineralization classified as Inferred Resources increased 116% increase to a total of 36.6 Mt at an average grade of 0.047% U_3O_8 (0.94 lbs U_3O_8/ton) and 1,554 ppm total REOs. The amount of contained metal in the Inferred Resources increased to 37.6 Mlbs of U_3O_8 and 125.3 Mlbs of REOs. Pele announced that expanded higher-grading zones had been identified and that its previous PEA had shown that these would positively impact project economics in the early production life of the project. Pele also identified scandium as a potential high-value by-product of uranium and REE mining.

Table 16
Pele Mountain resources Eco Ridge Mineral Resources Estimate – June, 2013

Resource Classification	Tonnes	U_3O_8		Total REO			U_3O_8 Equivalent
	('000s)	(%)	('000 lbs)	(ppm)	('000 lbs)	(%)	('000 lbs)
Indicated	22,743	0.045	22,554	1,606	80,510	0.099	49,827
Inferred	36,560	0.047	37,623	1,554	125,248	0.102	81,842

Source: Pele Mountain Minerals news release dated 10 June, 2013 and available on its website at http://pelemountain.com/news.php

15. MINING

15.1 OVERVIEW OF ELLIOT LAKE REGIONAL OPERATIONS

No uranium mines are presently active in the Elliot Lake area.

Further to the east, Prophecy Platinum Corp. (“**Prophecy**”) owns the Shakespeare Nickel Mine, formerly discovered and operated by Ursa Major Minerals with which Prophecy amalgamated in 2012. Open pit mining of the West Zone was suspended during February, 2012 when the company failed to secure an agreement with Xstrata Nickel for processing of Shakespeare ore at its Strathcona Mill located at Sudbury. Prophecy announced a new resource estimate on 12 September, 2012 whereby the underground-minable East Zone deposit contained an Indicated Resource of 3.57 Mt averaging 0.32% Ni, 0.39% Cu, 0.02% Co, 0.34 g Pt/t, 0.37g Pd/t and 0.20 g Au/t. The deposit also contained an Inferred Resource of 1.87 Mt averaging 0.32% Ni, 0.36% Cu, 0.02% Co, 0.34 g Pt/t, 0.36g Pd/t and 0.21 g Au/t. The resources were estimated using a $50/tonne NSR cut-off. The mine is located very near the former Agnew Lake uranium mine, and about 70 km west of Sudbury, Ontario.

15.2 FUTURE OPERATIONS ON THE APPIA PROPERTY

Although it is premature to speculate concerning future mine development on the Appia claims, it is certain that any potential mine development would be as an underground operation. The history of mine development in the Elliot Lake camp strongly suggests that the mining method would be room and pillar, or some modified version of this method. With the recent recognition of significant rare earth mineralization in the Upper Reef, Appia is considering several possible mining scenarios including:

- mining and processing the uranium-rich Lower Reef alone;
- mining the REE-enriched Upper Reef, extracting the Intermediate Quartzite to be used as back-fill and then mining the Lower Reef for its uranium; and,
- mining and processing the Upper and Lower Reefs simultaneously with the Intermediate Quartzite as a single unit to permit the use of larger equipment and thus achieve the maximum cost-reductions that the greater back height and scale offers.

No certain decisions have been made in this regard as of the date of this report.

16. MINERAL PROCESSING AND METALLURGICAL TESTING

16.1 BACKGROUND

Uranium production in the Elliot Lake area commenced in 1955 and by 1959 there were 11 mines with associated processing plants. The process used at all mills comprised crushing; grinding; high-temperature, high acid, high-density leaching; liquid-solid separation; and ion exchange for uranium recovery. Thorium recovery was started in one plant in 1959 and rare earth recovery was started in 1965.

Until recently, no substantive metallurgical research on mineral processing to determine uranium recovery parameters has been carried out since the last of the Elliot Lake mines closed in 1996. Notwithstanding this fact, the efficient recovery of uranium was well established at the time of closure. Over the many years that the mines operated, uranium recoveries averaged approximately 95% even while head grades declined from 2-3 lbs U_3O_8/ton (1-1.5 kg U_3O_8/tonne) to 1.6 lbs U_3O_8/ton (0.8 kg U_3O_8/tonne). Recoveries were certainly assisted by the occurrence of the economic minerals as discrete well-liberated subhedral-to-euhedral grains, although some uraninite grains were intergrown with U-Th silicates that leached well under the conditions employed.

Denison Mines was the first to apply bio-leaching, a relatively new technology in 1987 when it was introduced. That year the mine recovered 840,000 lbs (380,952 kg) of U_3O_8 by in-stope bacterial leaching (tonnage under leach not reported). Recoveries were facilitated by the simple mineralogy of the ores: brannerite ($U_xTh_{1-x}Ti_2O_6$), uraninite (UO_2) and minor low-uranium monazite ($[U,REE]PO_4$). Coffinite ($[U,Th]SiO_4$) and uranothorite ($[Th,U]SiO_4$), though present, are less important ore minerals. Both brannerite and uraninite are high-uranium minerals in their pure uranium end-members, the former containing 62.8% UO_2 and the latter being pure uranium oxide less any daughter products. Data presented by Robertson (1981) shows that the U:Th ratio in Blind River uraninites varied from 12.6:1 in some Panel Mine ores to as low as 5.4:1 in some Denison Mine ores with a regional average being 10:1.

Although uranium recovery is not metallurgically complex compared to many other mineral commodities, it does require robust leaching conditions for the Elliot Lake ores. Ifill et al (1989) demonstrates that a uranothorite grain subjected to harsh leaching conditions dissolved within 0.5 hr, and that elevated temperatures and greater acid content lead to rapid dissolution of uranothorite.

16.2 PREVIOUS APPIA STUDIES

During early 2011, Appia selected 3 core samples of uranium-bearing quartz-pebble conglomerate (57729, 57741 and 57757) for QEMSCAN® analysis at the SGS-Lakefield metallurgical testing facility located in Lakefield, Ontario, Canada. The results of the research are summarized in a SGS report dated 22 June, 2011.

SGS describes QEMSCAN as "an acronym for Quantitative Evaluation of Materials by Scanning Electron Microscopy, a system which differs from image analysis systems in that it is configured to measure mineralogical variability based on chemistry at the micrometer-scale. QEMSCANTM utilizes both the back-scattered electron (BSE) signal intensity as well as an Energy Dispersive X-ray Signal (EDS) at each measurement point. It thus makes no simplifications or assumptions of homogeneity based on the BSE intensity, as many mineral phases show BSE overlap. EDS signals are used to assign mineral identities to each measurement point by comparing the EDS spectrum against a mineral species identification program (SIP) or database."

The results from whole rock (major element/oxide) analysis by XRD and trace element analysis by ICP is found in the report by SGS dated 22 June, 2011. Selected analytical and mineralogical data from the samples is summarized as follows in Tables 17 and 18.

Table 17
Selected Chemical Analytical Data for Samples
Used for QEMSCAN Study at SGS-Lakefield

Sample Number	U	Th	Fe	La	Ce	Pr	Nd	Y	Zr	K	Al	Ca	Si
57729	0.12	0.13	2.48	0.16	0.27	0.03	0.09	0.03	0.11	3.10	4.68	0.17	36.27
57741	0.11	0.12	4.64	0.17	0.29	0.03	0.09	0.03	0.05	1.30	2.06	2.58	34.68
57757	0.02	0.03	3.62	0.03	0.06	0.01	0.02	0.00	0.02	2.21	3.29	0.62	37.26

* All values are in per cent.

The SGS QEMSCAN®™ of the samples (nominal pixel resolution of 1-2 µm) clearly demonstrated that the non-quartz fraction is almost entirely restricted to the matrix between quartz pebbles. Potassium feldspar was also found to be a major component in the matrix, occurring as sub-rounded grains up to 4,000 µm in size, some containing inclusions of muscovite. Muscovite grains less than 100 µm in size typically dot the rims of quartz pebbles. Calcite, though rare, appears as essentially inclusion-free veinlets up to 1 mm in thickness. Some of the thicker calcite veins carry very fine (<200 µm) inclusions of pyrite and plagioclase.

Six polished thin sections were prepared from the three core samples. The qualitative X-Ray diffraction results shown in Table 18 were derived by SGS-Lakefield using a Bruker D8 Advance Diffractometer, and indicate that for the three samples tested the major mineral was quartz. Minor mineral assemblages included K-feldspar-pyrite-mica, pyrite-calcite-K-feldspar and pyrite-K-feldspar. In all samples, mica was present in very low or trace amounts (Yeung and Zhou, 2011). The X-ray diffraction data did not indicate any unusual compositions to the major and minor minerals, however the data did not include useful information on uranium, REEs, or thorium which are present at the ppm level.

Table 18

Selected Mineralogy Data for Polished Thin Sections from Core Samples

Sample Number	Qtz	Kspar	Musc	Ca	Py	Fe-Ti Oxides	Mon	Syn	Bas	U-Th	Z	Ap	Other REE	Col
57729-1	71.1	14.5	6.7	0.0	4.3	0.0	1.0	0.0	0.0	0.1	0.1	0.1	0.0	0.0
57729-2A	67.4	12.6	17.1	0.3	1.1	0.0	0.4	0.0	0.0	0.0	0.1	0.0	0.0	0.0
57729-2B	57.1	13.4	17.1	0.0	5.4	3.7	2.4	0.0	0.0	0.2	0.4	0.1	0.0	0.0
57741-A	72.4	3.5	7.0	1.7	10.9	2.1	1.7	0.1	0.1	0.2	0.1	0.2	0.0	0.0
57741-B	77.5	3.4	3.8	3.9	7.5	1.9	1.5	0.1	0.0	0.1	0.1	0.0	0.0	0.0
57757	73.5	10.7	8.5	0.9	4.2	0.6	0.2	0.0	0.0	0.1	0.0	0.2	0.0	0.0

All values are in per cent. Qtz = quartz; Kspar = K-Feldspar; Musc = muscovite; Ca = calcite; Py = pyrite; Mon = monazite; Syn = synchysite; Bas = bastnasite; Th = thorite; Z = zircon; Ap = apatite; Col = columbite

Initial probe work indicates that grains have an a mean particle size ranging between 65 and 120 µm. Bastnasite and synchysite are significantly finer at 22-28 µm and 23-31 µm, respectively. Other unnamed REE minerals are also quite fine at 22-24 µm. U-Th minerals showed a mean grain size of about 60 µm.

Metallic mineral grains are not uniformly distributed in the matrix. Pyrite, Fe (+/- Ti) oxides, monazite, pyrochlore, REE minerals, apatite and zircon typically occur as enriched bands crossing the matrix except where disrupted by larger quartz pebbles. This has the appearance of micro-bedding or mm-scale sorting of the heavy mineral fractions (Plate 12). This aspect of heavy mineral distribution is better evidenced in some sample sections than in others, especially those scan images in which grain size is less than 3 mm. In scan images, calcite and mica have the appearance of being a late veining stage as evidenced by it occurring along fractures both in the matrix and within quartz pebbles.

A Scanning Electron Microscope (SEM) equipped with X-ray Energy Dispersive Spectrometer was used to acquire back scattered electron images and semi-quantitative analyses. Initial SEM data indicated that monazite is the main REE carrier followed by bastnasite/synchysite/parisite. REE phases include monazite $(Ce,La,Nd,Th)PO_4$ and fluoro-carbonates including bastnäsite, bastnäsite-(Ce) with a formula of $(Ce,La,Nd)CO_3F$, and calcium fluoro-carbonates $Ca(Ce,La)_2(CO_3)_3F_2$ and synchysite $Ca(Ce,La,Nd)(CO_3)_2F$. Uranium and thorium are likely present as thorite and uraninite, uranothorite, thorite and coffinite although these mineral names are based on semi-quantitative SEM-EDS analyses. Uranium and thorium minerals occur together and are difficult to resolve at the 15 µm resolution of the probe. Taking as an example sample 57729-1, SGS summarized the mineralization as:

- "Pyrite is significant in the sample, generally fine-grained, <50 µm to 0.5 mm, with aggregates up to 1 mm, but generally ~0.3 mm in size. It is well-formed and crystalline and is present as subhedral, angular grains, and locally forms aggregates. It is generally disseminated in the sample and interstitial to the main silicates. It hosts rare chalcopyrite inclusions of <50 µm.
- Monazite mineralization (1%) is characterized by fine-grained particles and is strongly associated with pyrite. Monazite is up to 0.3 mm in size and is subhedral to subrounded in habit. It has a cloudy appearance under the optical microscope that is attributed to the fine-grained thorite inclusions. It carries mainly cerium, and less lanthanum and neodymium.
- Thorite is tentatively identified and may carry significant uranium. It ranges from 5 µm to 30 µm in size, as angular and sub-rounded inclusions in monazite, but also forms distinct grains up to 0.3 mm in size. It is also associated with pyrite mineralization, and occurs interstitial to silicates and as attachments on monazite.
- Uranium minerals occur as either coffinite/uraninite or uranothorite. They occur as 5 µm to 70 µm, sub-rounded and anhedral shaped grains having a heterogeneous textural and chemical nature. They are locally complexly intergrown with silicates and pyrite. They form micrometric rims around pyrite and inclusions close to the edge of pyrite. They are also observed in close association with monazite and thorite."

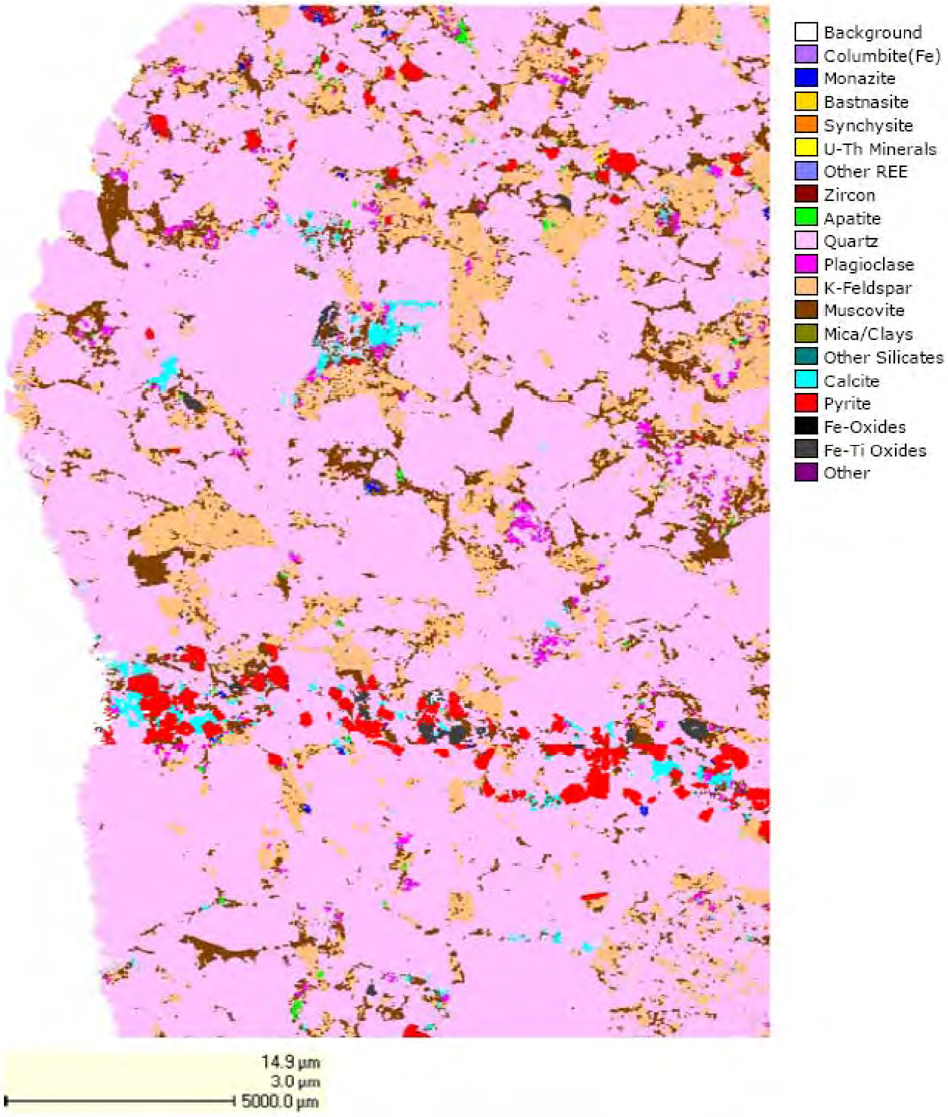

Plate 12: QEMSCAN image of sample 57757 showing pyrite, Fe-oxides, U-Th and REE minerals concentrated along bedding feature) from SGS, June 2011.

SGS concluded that the REE-U-Th mineralization is of a disseminated type, and it is strongly associated with pyrite. The metallic minerals are generally interstitial to the main silicates, and rarely do these minerals occur as inclusions in silicates, e.g., quartz. SGS concluded that the distribution related to the original bedding in the rocks, a view shared by WGM. SGS speculated that these features would allow good liberation of the metallic minerals collectively during mineral processing.

SGS underscores the close association between urano-thorium and REE minerals with pyrite, however some SEM images show more than an association; Plate 13 shows a U-Th phase clearly replacing pyrite.

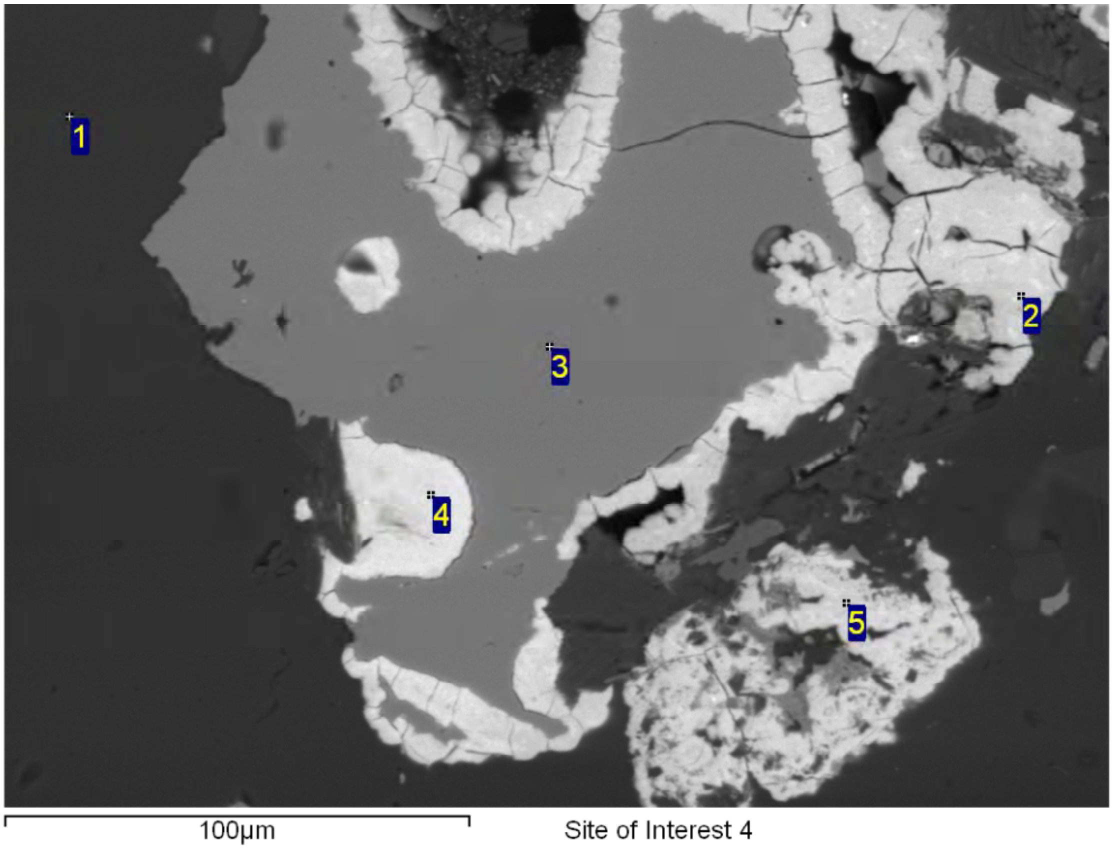

Plate 13: SEM image of U-Th +/- Y mineral (areas 2, 4 and 5) rimming and replacing pyrite (area 3) in quartz groundmass (area 1) – study site 4 in sample # 57741A – from SGS, June 2011.

The QEMSCAN data (Table 19) show that sample #57757 contains significantly less uranium, thorium and REEs than the other two samples which carry in excess of 0.1% U and 0.6% REEs. SGS concluded that cerium, lanthanum, neodymium, uranium and thorium contents were significantly elevated in samples #57729 and #57741 and enrichment was also present in yttrium, samarium and praseodymium. Monazite and fluoro-carbonates were found to account for most of the LREE. Yttrium was frequently identified in uranium and thorium minerals. SGS recommended that electron microprobe analyses be carried out to determine the distribution of the REE within the minerals identified. Monazite is the primary REE phase. Bastnasite, synchysite and parisite are also present in trace amounts and carry a proportion of the LREE. Monazite also hosts a number of micrometric thorite inclusions.

Thorite is the principle Th-mineral. Uranium is predominantly present as coffinite/uraninite and uranothorite.

Table 19
REE and Related Trace Element Geochemistry
of Samples Selected For QEMSCAN Study

Element	#57729-2 A/B	#57741	#57757
La (ppm)	1550	1700	304
Ce (ppm)	2680	2870	570
Nd (ppm)	924	936	166
Pr (ppm)	318	324	52.1
Sm (ppm)	153	155	23.6
Dy (ppm)	74	65	9
Er (ppm)	31	26.3	3.5
Eu (ppm)	6.1	5.8	1.6
Gd (ppm)	126	125	20
Ho (ppm)	12.6	11.1	1.5
Lu (ppm)	3.4	2.8	< 0.6
Sc (ppm)	7	4	3
Tb (ppm)	17.1	15.9	2.1
Yb (ppm)	25.2	20.6	2.6
Tm (ppm)	4.27	3.48	< 0.8
Y (ppm)	289	279	35.2
U (ppm)	1160	1100	158
Th (ppm)	1310	1210	333
Nb (%)	0.01	< 0.01	< 0.01
Zr (%)	0.11	0.05	0.02

The commercial viability of REE mineralization was previously demonstrated by the historical recovery of yttrium as a by-product of uranium production at the Elliot Lake mines. These operations proved that separate facilities were not required to leach the REEs. Yttrium was well leached during the normal uranium dissolution process and that, once in solution, yttrium could be recovered. However the mine operators ignored the other REEs because the market was adequately served by deposits elsewhere and because they were not effectively dissolved under normal uranium leaching conditions. At present, Appia plans to produce a high-value REE-uranium concentrate through beneficiation of the ore. Appia future production plans will be determined by market conditions at that time, but will likely focus on either extracting the metals from solution as uranium oxide and as combined REE-oxides known as mischmetal, or producing a U-REE concentrate for sale. Appia's assay data indicates that the value of the REEs present will largely vest in cerium, lanthanum, neodymium and yttrium which account for 86.3% of the total REEs present.

16.3 CURRENT TESTING

16.3.1 Introduction

Based on the historical production of yttrium as a by-product of uranium mining in the Blind River area, Appia recognized the potential of the Elliot Lake uranium ores to contain significant amounts of rare earth metals. At the time Appia's project commenced, limited data was available documenting the presence of REEs in the Elliot Lake ores even though petrographers had recognized the presence of monazite. Although monazite is commonly enriched in REEs, it was consistently referred to, if at all, as a "uranium ore mineral" and no mention was made of its REE potential even though yttrium was being recovered as a co-product with uranium (Thomson, 1960, Robertson, 1968, Robertson, 1976). It was not until 2007-08 when Appia drilled the Teasdale and Banana Lake uranium zones and analyzed the drill core that quantitative evidence was obtained showing the presence of economically significant REE mineralization in the stratabound, 3.2 m thick Lower Reef ("**LR**") horizon. Appia's exploration also led to the recognition of a second overlying horizon, the 3.95 m thick Upper Reef ("**UR**"), that contained significantly higher REE contents, although a somewhat lower average uranium content. The UR and LR are separated by the more weakly mineralized Intermediate Quartzite averaging 2.65 m in thickness.

Rather than mining only the LR as had been done in the past, Appia recognized the cost-saving potential of bulk mining the entire LR-IQ-UR assemblage and the economic benefit of recovering REEs as well as uranium. At the end of its most recent drilling in the latter half of 2012, Appia concluded that it had sufficient evidence of widespread REE-U mineralization and sufficient sample media to commence a meaningful metallurgical testing program to investigate processing options for the recovery of REEs and uranium.

Following a competition between laboratories, WGM and Appia selected SGS Minerals Services, Lakefield Site ("**SGS**") to undertake beneficiation and hydrometallurgical testwork under WGM supervision and managed by Senior Associate Metallurgist, John Goode, P.Eng. Drill core was quarter-cut and dispatched in secure containers from Elliot Lake to SGS where it was crushed and composited prior to testwork. The testing program was designed by Mr. Goode in consultation with SGS personnel as follows: (1) mineralogical work was conducted by Tomas Hrstka and Tassos Grammatikopoulos; (2) beneficiation work was carried out under the supervision of Feng Gao, Ben Yu, Yashashree Chaugule, Jason Garbutt, and Dan Lang; (3) hydrometallurgical work was supervised by James Brown, Michael Archer and

Krystal Davis; and (4) reports were prepared by Su McKenzie. The final SGS report is appended to this report.

16.3.2 Samples

Appia and WGM selected drill core to represent the UR, IQ and LR material. Remaining half core was sawed into quarter core that was shipped to SGS who prepared metallurgical composites of each material type as well as a Master Composite containing appropriate portions of the three individual composites. Analyses for the four composites are presented in Table 20.

Table 20
Elemental Analysis of Appia Composites

Element	UR Comp	IQ Comp	LR Comp	Master Comp
La	536	259	352	407
Ce	1000	488	681	767
Pr	105	51.7	73.5	81.0
Nd	328	162	235	254
Sm	54.3	26.2	41.9	42.6
Eu	2.9	1.5	2.7	1.9
Gd	34.0	15.7	28.5	26.1
Tb	4.7	1.8	4.2	3.3
Dy	20.3	8.6	19.2	16.5
Ho	3.7	1.4	3.4	2.6
Y	76.0	31.0	65.0	62.0
Er	8.8	3.3	8.2	6.7
Tm	2.0	0.4	1.3	0.8
Yb	7.1	2.5	5.9	5.2
Lu	2.3	< 0.5	1.0	0.7
U	218	120	520	270
Th	445	218	350	354
S (%)	1.46	1.04	3.32	1.92

16.3.3 Mineralogy

SGS completed mineralogical examination of the three composites using X-ray diffraction (XRD), QEMSCAN, Electron Microprobe Analysis (EMPA), Scanning Electron Microscopy (SEM) and chemical analysis. XRD and QEMSCAN showed that all three samples contained about 66% quartz (SiO_2) obviously from the quartz pebbles of the conglomerate. Samples also contained approximately 15% potassium-feldspar ($KAlSi_3O_8$), and pyrite (FeS_2) amounting to 10%, 4.1%, and 2.6% respectively for the LR, UR, and IQ composites respectively. The samples also contained minor graphite and about 10% mica ($K(Mg,Fe)_3Si_3AlO_{10}(OH)_2$).

The distribution of the minerals containing U, Th, and the REE are indicated in Figure 14.

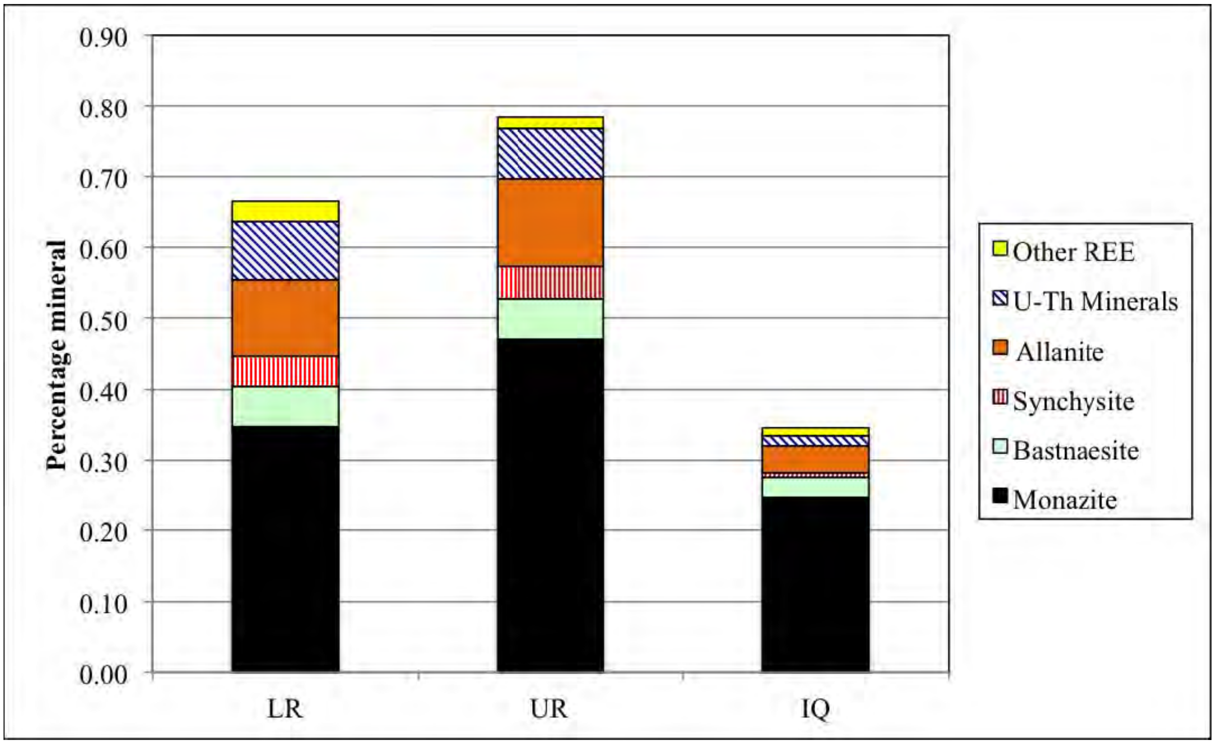

Figure 14: Distribution of U, Th and REE minerals in Appia samples

Brannerite is often mentioned as being an important component of Elliot Lake ore. QEMSCAN did not indicate the presence of brannerite, but it was tentatively identified during the SEM examination of the samples as either micrometric inclusions in various minerals and local alteration of the U-Th phases.

QEMSCAN demonstrated that the mean grain size of the minerals of interest were below 20 µm in all three composites.

EMPA work showed that approximately 70% of the Y was contained in U-Th minerals with the balance in monazite. This distribution agrees well with the historical U and Y leach recovery values for Elliot Lake ore.

16.3.4 Beneficiation

Prior metallurgical data from the Elliot Lake camp, initial hydrometallurgical testwork on Appia samples, and simple economic analyses showed that leaching of whole ore would provide for good recovery of uranium, reasonable recovery of yttrium but poor recovery of the other REEs. In contrast, concentration of the more refractory REE minerals would allow their intensive hydrometallurgical processing and the possibility for high recoveries of all REEs. Leaching of uranium from flotation tailings was shown to be simple and effective. Hence SGS was directed in a program aimed primarily at concentrating the REE minerals followed by the recovery of pay elements from the concentrate and tailings. Preliminary cost estimates showed that high REE recovery to a concentrate of about 10% mass could be effective.

Earlier literature on the beneficiation of Elliot Lake ore showed that ore sorting and dense media separation could be effective but would not be able to produce the degree of up-grading needed (Grimes, 1973). These methods were therefore not examined in the preliminary work program but could be looked at in the future.

Grinding Work Index

A standard Bond ball mill work index measurement was made using a 150 µm closing screen A metric value of 16.2 was determined.

Gravity concentration

SGS undertook gravity separation tests on the Master Composite using a Mozley Mineral Separator at grind sizes of 80% passing 208 µm and 80% passing 108 µm. SGS also completed a Wilfley table test starting with material ground to 100% passing 850 µm with the tails ground to -500 µm, re-tabled, tails re-ground to -106 µm and re-tabled. Summary results are provided in Table 21.

Table 21
Summary Results of Gravity Separation Tests

Test	Conditions	Stream	Distribution (%)						
			Mass	U	Th	S	LREE	HREE	TREE
G1	Mozley, 80% past 208	Conc.	6.6	51.5	61.1	84.3	58.7	56.1	58.6
		Conc. + tails -38	25.7	74.7	81.2	94.7	78.6	77.7	78.6
G2	Mozley, 80% past 108	Conc.	5.8	44.8	59.3	81.0	58.0	53.1	57.7
		Conc. + tails -38	37.8	75.2	83.6	94.5	82.3	79.5	82.1
W1	Wilfley, progressive grind	100% past 850 µm	19.9	53.0	59.1	82.9	54.8	48.7	54.3
		+100% past 500 µm	29.0	68.6	75.5	90.6	70.2	62.7	69.6
		+100% past 106 µm	33.0	73.4	80.7	93.3	75.8	67.8	75.2

Notes.

1. LREE are the “light” rare earths including La, Ce, Pr, Nd, Sm, and Eu. HREE are the remaining rare earths and Y. The TREE include the REE from La to Lu and including Y.
2. The tails were screened and assayed and enhanced grades found in the -38 mesh fraction hence the combinations shown in Table III.

Magnetic Separation

SGS conducted wet high intensity magnetic separation (“WHIMS”) at intensities of approximately 2,000 G (Gauss), 5,000 G, 10,000 G, and 17,000 G on samples of Master Composite ground to 80% passing 208 µm and 108 µm. Results are summarized in Table 22.

Table 22
Summary Results of Magnetic Separation Tests

Test	80% passing (µm)	Fraction	Distribution, %						
			Mass	U	Th	S	LREE	HREE	TREE
M1	208	0 - 2000 G mag.	2.2	9.1	8.5	7.2	8.1	8.9	8.1
		0 - 5000 G mag.	4.5	23.7	26.3	14.5	29.7	27.0	29.6
		0 - 10,000 G mag.	6.6	44.3	55.4	22.0	60.8	53.9	60.4
		0 - 17,000 G conc.	8.6	61.2	74.2	28.8	78.2	72.2	77.8
		0 - 17,000 G conc.&-38 tail	28.9	74.8	83.8	46.3	85.7	82.4	85.5
M2	108	0 - 2000 G mag.	2.5	8.9	8.8	5.5	8.4	9.2	8.4
		0 - 5000 G mag.	4.9	28.2	35.3	13.8	42.2	35.7	41.7
		0 - 10,000 G mag.	4.9	28.2	35.3	13.8	42.2	35.7	41.7
		0 - 17,000 G conc.	7.6	52.6	64.6	20.9	71.6	62.7	71.0
		0 - 17,000 G conc.&-38 tail	38.5	74.9	81.2	44.5	85.3	80.1	84.9

Froth Flotation

SGS completed 20 batch flotation tests on the Master, UR, IQ, and LR Composites. The collector that was generally used was LR19 which is a blended reagent developed by Lakefield Research for the flotation of Elliot Lake ores in the 1960s (Grimes, 1973) and comprising 62% FA2 (a fatty acid), 27% Cytec Aero 855 (a petroleum sulphonate), 9% kerosene, and 2% MIBC. Other collectors that were investigated were cupferron and FS-2 (Muthuswami et al, 1983) and salicylhydroximate provided by the Tieling Flotation Reagents Factory, Liaoning, China.

Tests with LR19 were shown to be substantially superior to all other reagents when applied to Appia Master Composite. Figure 15 summarizes data for flotation test F6 which comprised rougher flotation and cleaning stages and was done after stage grinding to 80% passing 135 µm. Test F7 done after grinding to 80% passing 80 µm gave identical recovery values. Reagent additions were a nominal 1.5 kg/t of LR19 and 1 kg/t of PQ Metso sodium metasilicate.

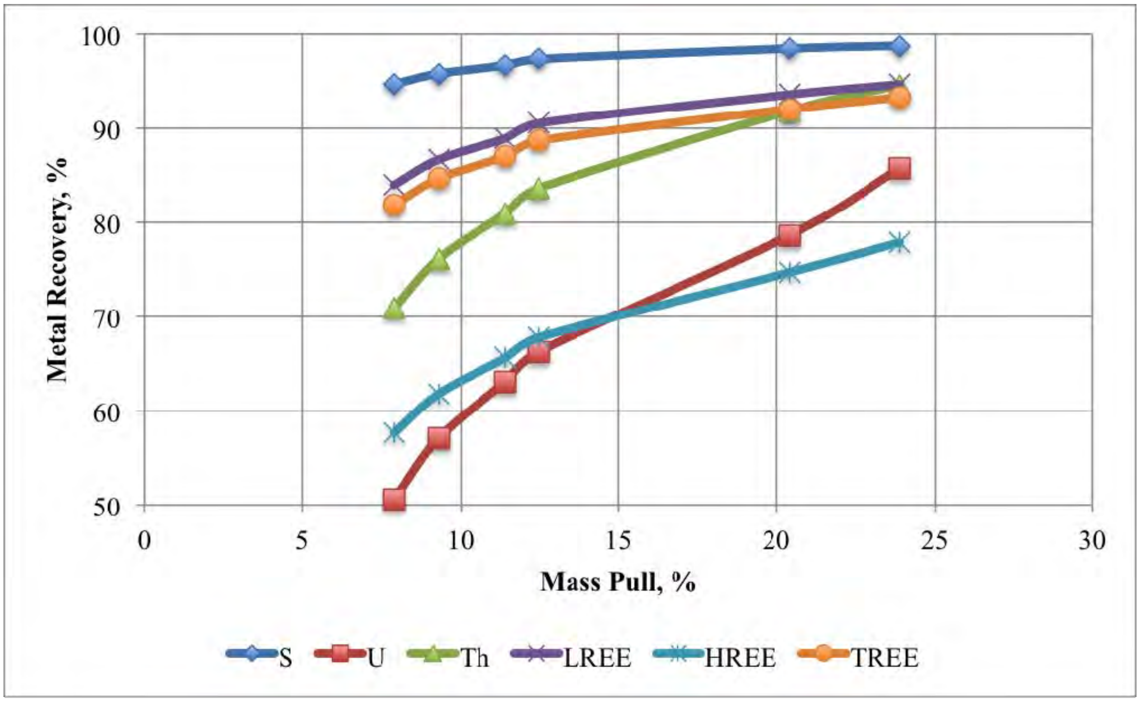

Figure 15: Flotation of Appia ore using LR19 collector

Figure 15 also shows very high recovery of pyrite and reasonably effective recovery of the LREE at low mass pulls with the HREE and U recovery lagging behind. As discussed later, the lower U and HREE recoveries are not a major concern if the flotation tailings are leached since uranium and HREE dissolution in a tailings leach are reasonably high.

The LREE and HREE recovery results for a successful beneficiation test (F6) are plotted together in Figure 16. The figure shows that froth flotation performed significantly better at recovering the LREE-bearing minerals which are those requiring an aggressive cracking process. WHIMS performed somewhat better than flotation in recovering the HREE-bearing minerals however that is not too critical given that a tailings leach is reasonably effective at recovering HREE. There may be a case for a hybrid circuit involving flotation and WHIMS or gravity concentration and such possibilities might be investigated at later stages of the project development.

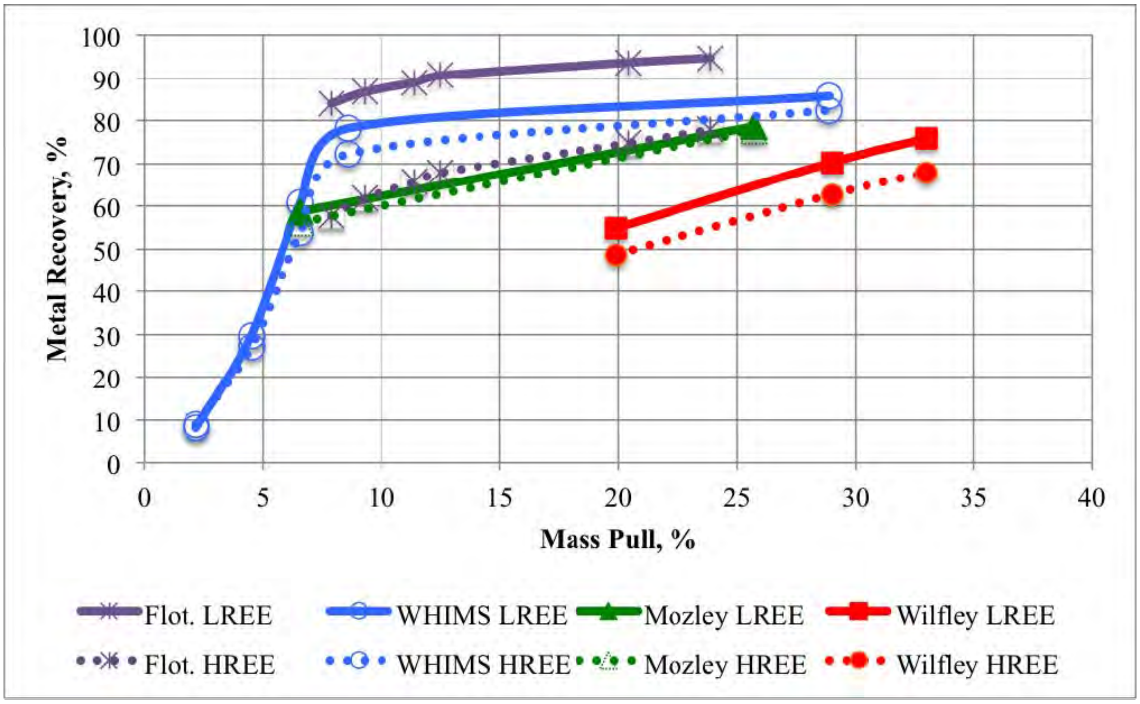

Figure 16: Comparison of beneficiation methods

16.3.5 Hydrometallurgy

Whole Ore Leaching

Two whole ore leach tests were conducted on ore ground to 80% passing 212 µm (test AL-1) and 75 µm (test AL-2). In both cases the leach was performed at 80^0 C with 50 g/L free acid and over 72 h with kinetic sampling. The results for AL-2 are plotted in Figure 17.

The performance of the LREE in test AL-2 (and other tests) indicates LREE precipitation with time. A possible mechanism is the formation of insoluble LREE double sulphates caused by the entry of K and Na into the system by the dissolution of K-feldspar and other minerals. K+Na assays in test AL-2 were 1.3 g/L at the 12 h mark (>90% K, balance Na) and increased to 2.3, 3.3, and 4.1 g/L respectively in the 24, 48, and 72 h samples.

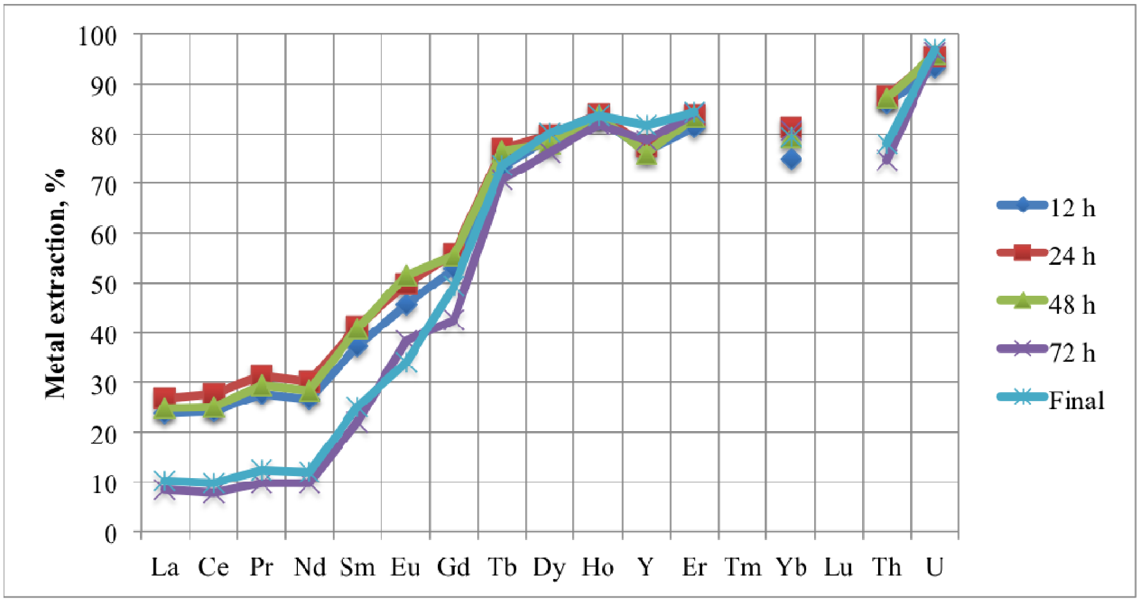

Figure 17: Whole ore leach of Appia ore, 80^0 C, 50 g/L free acid

Since U leaching seemed to be essentially complete after 48 h, such leach times, or shorter, would be appropriate for maximization of LREE recovery. The acid demand for such leach times were 57 kg/t and 47 kg/t for the coarse and fine grinds respectively.

Flotation Concentrate Leaching

Several different methods of processing the flotation concentrate were investigated including simple atmospheric acid leaching, pressure oxidation in which the pyrite was oxidized to generate lixiviant solution and simultaneously promote metal dissolution, and sulphuric acid baking – a standard way of cracking monazite/bastnaesite minerals.

<u>Agitated Atmospheric Leaching</u> Flotation concentrate was subjected to a 48 hour long, 50 g/L sulphuric acid leach procedure at 80^0 C with kinetic sampling. Not too surprisingly, the U and REE extraction

data were very similar to those for the whole ore with about 30% extraction for La and Ce and about 80% for Y and related HREEs.

Pug Leaching

In the 1970s, Rio Algom pilot tested the pug leaching of Elliot Lake ore and patented the process (Mather, 1978). A sample of flotation concentrate was mixed as a paste with sulphuric acid and the resulting pellets held for 24 hours at a nominal 80^0 C with moist air passing through the pellet bed. After 24 hours, the ore was slurried with water, filtered and the products assayed. Data showed less than 30% U extraction, very low LREE extraction and about 30% HREE extraction.

Pressure Oxidation

SGS was instructed to perform a single pressure oxidation (POX) test on a flotation concentrate containing 814 g/t of U, 5,877 g/t of TREE, and 9.2%S. Test conditions included pre-acidification to pH 1.8 (39 kg/t acid added), a temperature of 210^0 C and an oxygen overpressure of 689 kPa (100 psi). The test was extended over four hours although full oxidation of pyrite and uranium extraction was expected in a shorter period of time. It was speculated that some cracking of refractory REE minerals might also occur given adequate time.

In this test, 88% of the S was oxidized, 98% of the U solubilized, along with 28% of the TREEs, and 71% of the HREEs. Y extraction was 80% reflecting the close association between U mineralization and Y. Clearly monazite was not cracked during the POX operation.

Acid Baking

Several acid bake tests were performed on various Appia flotation concentrates. Initial tests indicated that a temperature of 200^0 C and an acid addition of 1 t/t concentrate, and a 3 hour retention time was satisfactory with an acid consumption of 0.3 t/t, U extraction of 98%, HREE extraction of 90%, and LREE extraction increasing from 60% for La to 90% for Sm.

A pre-leach and acid bake process, simulating a counter-current leach system, was executed in tests AL-7 and AB-6 on a concentrate assaying 780 g/t U and 13,636 g/t TREE obtained from a flotation process (tests F19, F20, and F21) which yielded 10.6% concentrate mass containing 92% of the S, 31% of the U, 79% of the LREE and 50% of the HREE – somewhat inferior results to those of test F6.

The pre-leach required 105 kg/t of acid. The acid bake on the pre-leach residue was done with a 600 kg/t acid addition and at a temperature of 250^0 C for 3 hours. The subsequent water leach contained 130 kg/t of free acid which would, in a counter-current system, be more than sufficient for the pre-leach meaning that the fresh acid demand for the pre-leach and bake system is about 600 kg/t of concentrate.

Metal extraction in the AL-7/AB-6 pre-leach and acid bake tests were quite satisfactory as illustrated in Figure 18.

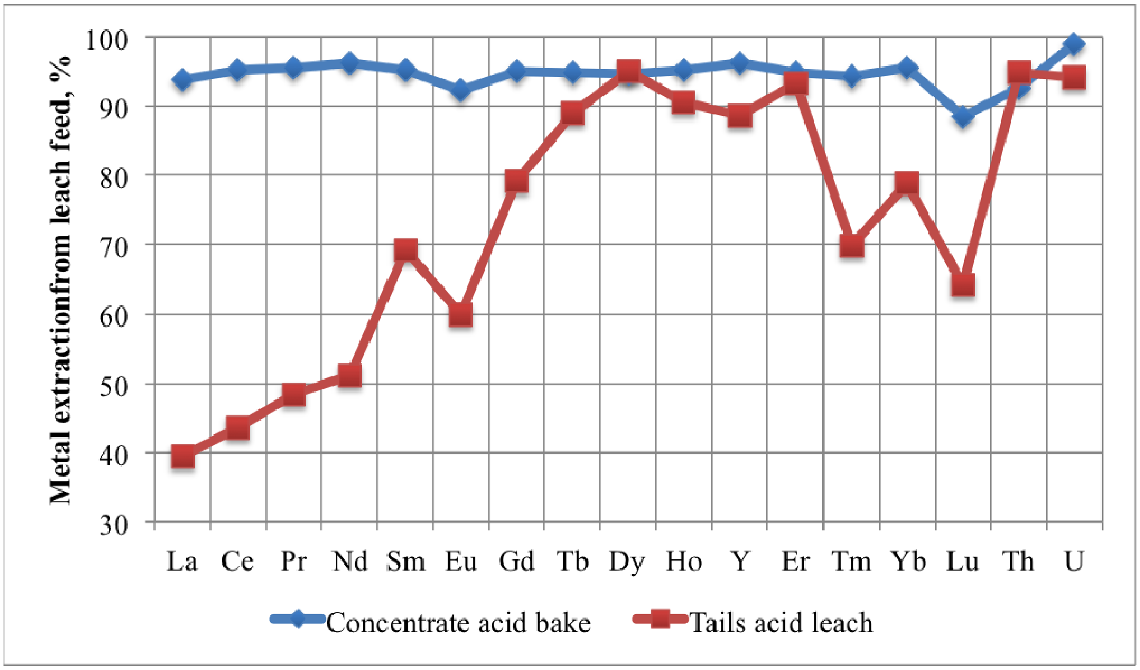

Figure 18: Extraction by acid bake of flotation concentrate and atmospheric leach of tails

Flotation Tailings Leaching

Samples of the flotation tailings corresponding to the concentrate used in the AL-7/AB-6 pre-leach and acid bake test were leached for 24 hours at both room temperature and 50^0 C in dilute sulphuric acid in tests AL-9 and AL-10. The data showed that in both cases metal extraction was substantially complete after just 12 hours. Acid demand was about 27 kg/t. The higher temperature leach offered 5% better U extraction and an average of 9% greater REE extraction and so that system was selected. Data are plotted in Figure 18 alongside the extraction data for the concentrate acid bake.

16.3.6 Solution / Slurry recovery Options and test Results

In the 1950s and on, efficient U recovery from Elliot Lake leach slurry was readily done using partial neutralization, liquid solid separation ("**LSS**"), clarification, and strong base ion exchange ("**IX**") for U extraction. REE were recovered from the IX barren using solvent extraction ("**SX**") with di-2-ethyl hexyl phosphoric acid ("**DEHPA**") either directly or after partial removal of ferric iron and thorium by precipitation.

A modern approach to U and REE recovery might include LSS followed by SX for U using a tertiary amine, raffinate solution adjustment, followed by SX for REE using DEHPA or a modern substitute. An alternative approach, offering substantial capital cost savings, would be to dispense with LSS and simply use resin-in-pulp ("**RIP**") for U as was practiced at many earlier plants, is presently used by Paladin Energy (Hladun, 2010), and analyzed and advocated by others (Goode and Brown, 2010). However, to completely eliminate LSS also requires the use of RIP for REE recovery.

Limited testwork was done on post-leach metal recovery. The work showed that neutralization of a clear solution to a pH of 3.5 using MgO would precipitate ferric iron and more than 70% of the Th without significant loss of U or REE. The same procedure using finely ground $CaCO_3$ was far less effective in that although similar levels of ferric iron and Th elimination were obtained, REE losses were significant at pH 2 and were unacceptable by the time pH 3.5 was reached. Limestone precipitation from a slurry, rather than a clear solution could be more effective and will be tested in future work.

A limited amount of IX testwork was done in which a pregnant solution was treated by strong base IX for U recovery and the barren then treated with Lewatit OC 1026 resin for REE adsorption. This macroporous resin has DEHPA incorporated in the resin and so a RIP system analogous to the earlier SX systems could be possible. HREE extraction was high but LREE extraction was low and further work is indicated.

Another option, which was not tested, is the simultaneous extraction of U and the REE by a suitable solvent or resin followed by selective stripping/elution or other means of separately recovering the two products. Further testwork is needed to examine this option and other aspects of U and REE extraction options – both from clear solution using SX and from slurry using RIP.

16.3.7 Flowsheet Options and Overall Recovery

The main flowsheet options available are:

- a simple grind and leach of the Teasdale ore followed by either LSS and SX or RIP for U and REE recovery from solution – sequentially or simultaneously

- a flotation process to recover a high grade concentrate, acid baking of the concentrate and acid leaching of the flotation tailings. This option could be detailed in several ways as follows:
  - separate LSS operations on the acid bake residue and leached tailings and inter-linked SX circuits for U and REE recovery in which weakly loaded extractant from the tailings circuit is passed on to the higher grade concentrate circuit
  - separate inter-linked RIP circuits for the acid bake residue and tailings leach circuits
  - joining the concentrate leach and tailings leach pregnant slurries and recovering U and REE using either LSS and SX or RIP
  - a common U recovery circuit and separate recoveries of dominantly LREE and HREE products from the acid bake and tails leach circuits respectively

The total recovery levels expected from a 36 h whole ore leach on finely ground ore (AL-2), allowing for 5% soluble loss of U and 10% loss of REE, are plotted in Figure 19.

Experimental data for the flotation, acid bake, tailings leach circuit have been combined and are also presented in Figure 19 along with an estimate of the total overall recovery allowing for 5% soluble loss.

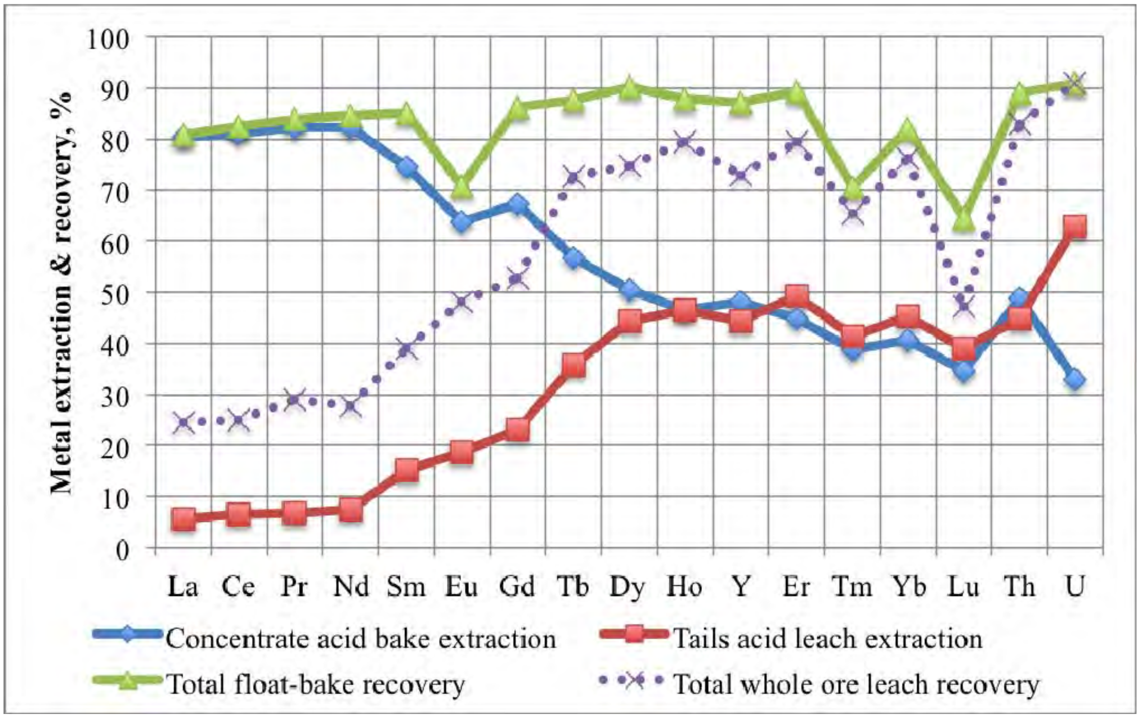

Figure 19: A comparison of recoveries from whole ore leaching only, and a flotation concentrate bake and tailings leach circuit.

16.3.8 Conclusions

Preliminary testwork and analysis indicates that a flotation-concentrate acid bake process route combined with a tailings leach can give high extraction of U and REE from Appia's Teasdale uranium – rare earth deposit. It is believed that planned additional testwork and data analysis will lead to reduced costs and improved recovery.

A very preliminary estimate of the capital and operating cost of three process options are presented in Table 23. Costs are for the process plant only.

Table 23
Preliminary Production, Capital, and Operating costs for 7,000 t/d operation

Parameter	Unit	Leach-Filter U:SX; REE:SX	Float-Bake-Filter U:SX; REE:SX	Leach U:RIP; REE:RIP
		Metal Recovery		
LREE	t/a	893	2,840	893
HREE	t/a	182	229	182
TREE	t/a	1,075	3,069	1,075
U	t/a	551	551	551
		Costs		
Capital cost	M$	$347.00	$418.00	$284.00
Operating cost	M$/a	$75.00	$110.00	$74.00
Operating cost	$/t	$29.21	$43.17	$28.85

17. MINERAL RESOURCE AND MINERAL RESERVE ESTIMATES

17.1 DEFINITIONS

The classification of Mineral Resources used in this report conforms with the definitions provided in the final version of NI 43-101, which came into effect on February 1, 2001, as revised on April 8, 2011. We further confirm that, in arriving at our classification, we have followed the guidelines adopted by the Council of the Canadian Institute of Mining Metallurgy and Petroleum ("**CIM**") Standards. The relevant definitions for the CIM Standards/NI 43-101 are as follows:

> A *Mineral Resource* is a concentration or occurrence of diamonds, natural, solid, inorganic or fossilized organic material including base and precious metals, coal, and industrial minerals in or on the Earth's crust in such form and quantity and of such a grade or quality that it has reasonable prospects for economic extraction. The location, quantity, grade, geological characteristics and continuity of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge.
>
> An *Inferred Mineral Resource* is that part of a Mineral Resource for which quantity and grade or quality can be estimated on the basis of geological evidence and limited sampling and reasonably assumed, but not verified, geological and grade continuity. The estimate is based on limited information and sampling gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes.
>
> An *Indicated Mineral Resource* is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics, can be estimated with a level of confidence sufficient to allow the appropriate application of technical and economic parameters, to support mine planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes that are spaced closely enough for geological and grade continuity to be reasonably assumed.
>
> A *Measured Mineral Resource* is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, physical characteristics are so well established that they can be estimated with confidence sufficient to allow the appropriate application of technical and economic parameters, to support production planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes that are spaced closely enough to confirm both geological and grade continuity.

> A *Mineral Reserve* is the economically mineable part of a Measured or Indicated Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This Study must include adequate information on mining, processing, metallurgical, economic and other relevant factors that demonstrate, at the time of reporting, that economic extraction can be justified. A Mineral Reserve includes diluting materials and allowances for losses that may occur when the material is mined.
>
> A *Probable Mineral Reserve* is the economically mineable part of an Indicated, and in some circumstances a Measured Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This Study must include adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction can be justified.
>
> A *Proven Mineral Reserve* is the economically mineable part of a Measured Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This Study must include adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction is justified.

Mineral Resource classification is based on certainty and continuity of geology and grades. In most deposits, there are areas where the uncertainty is greater than in others. The majority of the time, this is directly related to the drilling density. Areas more densely drilled are usually better known and understood than areas with sparser drilling.

17.2 TEASDALE LAKE ZONE

17.2.1 BACKGROUND

WGM prepared an initial Mineral Resource estimate for the Teasdale Zone in 2008, and a subsequent update in 2011 (Workman and Breede). Both reports also contain a discussion of a historical estimate made by Mr. Doug Sprague, P.Eng., the former Chief Geologist for Rio Algom Ltd. at Elliot Lake, and WGM's reported detailed audit of that estimate.

17.2.2 2008 MINERAL RESOURCE ESTIMATE

A historical resource estimate was prepared by Mr. Doug Sprague, P.Eng., former Chief Geologist for Rio Algom Ltd. at Elliot Lake (Sprague). This estimate, totalling some 17.5 million tons of uranium mineralization having an average grade of 1.21 lbs U_3O_8 per short ton, is herein described in this report under Section 5.3 Historical Reserves and Resources.

During 2008, WGM carried out a detailed audit of the Sprague resource estimate. WGM's review was based on a combined database encompassing the original 16 historical drill holes plus six holes completed by Appia during its winter 2007-08 drilling program. WGM concluded that a search radius of 89 metres (338 feet) and a cut-off grade of 0.65 lbs U_3O_8/ton would produce a resource estimate that approximated that of Mr. Sprague. The result of WGM's audit was that the zone contained approximately 18.5 M tons grading 1.17 lbs U_3O_8/ton for a total of approximately 21.72 M lbs of contained U_3O_8. WGM's search radius was comfortably within the 400 ft extrapolation distance used historically for reserve estimation at the operating mines at Elliot Lake. WGM was essentially satisfied that the Sprague estimate was reasonable based on the information available.

During 2008, WGM also completed an updated uranium resource estimate for the Teasdale Lake Zone that was reported in a technical report by Workman and Vasek (2008). The resources were classified at a range of cut-off grades. The data show that using a cut-off of 0.60 lbs U_3O_8/ton results in an Indicated Mineral Resource of 17.4 million tons (15.8 Mt) with an average grade of 1.10 lbs U_3O_8/ton (0.55 kg U_3O_8/t) and an Inferred Mineral Resource of 48 million tons (43.5 Mt) at the same grade. At this cut-off grade, the uranium oxide contained in Indicated and Inferred resources was 19.0 Mlbs and 52.7 Mlbs, respectively.

17.2.3 2011 MINERAL RESOURCE ESTIMATE

The 2011 WGM Mineral Resource estimate took in both uranium and rare earth element mineralization and was based solely on the six holes previously completed by Appia. This represented a subset of the total 22 holes drilled on the deposit to that point in time. Because only these six Appia holes were assayed for rare earths, the Mineral Resource estimate was restricted to the area of influence of this data and the historical drill holes were necessarily excluded.

No per cent TREE cut-off was used for the reporting of resources, however implicitly there was an internal cut-off grade of about 0.05% TREE (i.e. the lowest grade interval included in the mineralized envelope at the hanging wall and footwall contacts). The resource envelop was geologically constrained by the geological contacts of the zone as follows:

- the upper surface of the stratigraphically highest U-bearing conglomerate (reef); and,
- the under surface of the stratigraphically lowest U-bearing reef.

WGM imposed a 2.44-metre (8 ft) minimum thickness requirement on the Teasdale Zone which reflects historical mining practices in the Elliot Lake district. All of the Appia drill hole intersections exceeded this thickness.

17.2.4 2013 MINERAL RESOURCE ESTIMATE

As with the 2011 estimate, the current Mineral Resource estimate takes in both uranium and rare earth element mineralization and is based on the previous six holes completed by Appia (holes Q-07-01 through Q-08-06 totalling 2,650.2 m), plus the additional 18 holes completed during the 2012 exploration program (holes AEC12-01 through AEC12-16 totalling 8,130.2 m). Only these collective 24 Appia holes were assayed for rare earths, hence the current Mineral Resource estimate has been restricted to the area of influence of this data, and the historical drill holes completed by Conecho and others have been necessarily excluded.

The estimate was prepared from a polygonal model using a C$:US$ exchange rate of 1:1 and at a cut-off value of $100 per tonne, using a uranium price of US$70/lb U_3O_8, and a combined TREE price of $78/kg. The resource envelop was also geologically constrained by the geological contacts of the zone as follows:

- the hanging wall or upper surface of the stratigraphically highest U-bearing conglomerate (Upper Reef or "**UR**"); and,
- the foot wall or lower surface of the stratigraphically lowest U-bearing reef (Lower Reef or "**LR**").

WGM imposed a 2.44-metre (8 ft) minimum thickness requirement on the Teasdale Zone which reflects historical mining practices in the Elliot Lake district. All of the Appia drill hole intersections exceeded this thickness. Indicated and Inferred Mineral Resources are reported in Tables 24 and 25 as follows:

Table 24
Summary of Teasdale Zone Uranium and Rare Earth Mineral Resource Estimate

Zone	Tonnes (‘000)	Tons (‘000)	TREE (lbs/ton)	U_3O_8 (lbs/ton)	Average Thickness (m)	Contained TREE ('000 lbs)	Contained U_3O_8 ('000 lbs)
INDICATED RESOURCES							
UR	6,733	7,422	4.20	0.484	4.61	31,199	3,593
IQ	3,006	3,314	1.98	0.259	2.27	6,578	0.857
LR	3,355	3,699	2.68	0.958	2.60	9,912	3,544
Total	**13,095**	**14,435**	**3.30**	**0.554**	**9.48**	**47,689**	**7,995**
INFERRED RESOURCES							
UR	18,326	20,201	3.87	0.421	4.33	78,080	8,498
IQ	10,209	11,254	1.64	0.184	2.78	18,464	2,070
LR	9,972	10,992	3.33	0.869	2.71	36,631	9,564
Total	**38,507**	**42,447**	**3.14**	**0.474**	**9.82**	**133,175**	**20,115**

Notes: Mineral Resources effective 30 July, 2013

1. Mineral Resources are estimated at a cut-off value of $100 per tonne, using a uranium price of US$70/lb U_3O_8, a TREE price of $78/kg, and a C$:US$ exchange rate of 1:0.9.
2. Mineral Resources which are not Mineral Reserves do not have demonstrated economic viability. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.
3. The quantity and grade of reported Inferred Resources in this estimation are uncertain in nature and there has been insufficient exploration to define these Inferred Resources as an Indicated or Measured Mineral Resource and it is uncertain if further exploration will result in upgrading them to an Indicated or Measured Mineral Resource category.
4. The Mineral Resources were estimated using the Canadian Institute of Mining, Metallurgy and Petroleum standards on Mineral Resources and Reserves, Definitions and Guidelines prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council December 11, 2005. These definitions are provided in Section 17.1 of this report.
5. S.G. of 2.85 tonnes/m^3 (or 3.14 tons/m^3) was used.
6. Indicated amounts may not precisely sum due to rounding.

The individual REEs included in the TREE mineral resources are reported as follows in Table 25. The contents are expressed in ppm for comparative reasons (500 ppm = 1 lb/ton).

Table 25
Individual REE Resource Grade Composition Summary for Teasdale

Zone	Light REE (ppm)						**Heavy REE (ppm)**									
	La	Ce	Pr	Nd	Sm	Eu	**Gd**	**Tb**	**Dy**	**Ho**	**Er**	**Tm**	**Yb**	**Lu**	**Hf**	**Y**
INDICATED RESOURCES																
UR	540	951	93.9	313	51.7	1.9	32.8	3.9	17.2	2.7	7.0	0.9	5.5	0.8	6.8	72.9
IQ	256	452	44.9	148	24.4	1.0	14.7	1.8	7.7	1.2	3.1	0.4	2.5	0.4	3.6	30.6
LR	332	596	59.4	201	35.1	1.7	23.2	3.0	14.2	2.3	5.9	0.8	4.5	0.6	3.3	58.1
Total	**422**	**745**	**73.8**	**247**	**41.1**	**1.7**	**26.2**	**3.2**	**14.3**	**2.3**	**5.8**	**0.8**	**4.6**	**0.7**	**5.2**	**59.4**
INFERRED RESOURCES																
UR	498	876	85.9	285	47.2	1.8	29.3	3.5	15.9	2.5	6.5	0.9	5.3	0.8	6.8	67.9
IQ	213	374	37.0	122	20.0	0.8	12.3	1.4	6.4	1.0	2.6	0.4	2.2	0.3	3.3	26.5
LR	417	747	73.9	249	43.4	1.9	28.5	3.6	16.4	2.6	6.6	0.9	5.2	0.7	4.5	66.4
Total	**401**	**709**	**69.9**	**232**	**39.0**	**1.6**	**24.6**	**3.0**	**13.5**	**2.1**	**5.5**	**0.7**	**4.4**	**0.6**	**5.3**	**56.5**

Qualifying notes for Mineral Resources are contained in Table 24.

Although Conecho's historical drilling of 19 holes during 1954-55 discovered economically interesting mineralization beyond the areas drilled by Appia, WGM's estimate does not make use of this information because of WGM inability to confirm the data, which is thought to be reliable, and data levelling issues principally concerning the lack of analytical data for REEs. For this reason, and the fact that the stratabound mineralization is not constrained by Appia's current drilling, additional future drilling is expected to enlarge the Mineral Resources reported herein which are defined by the drilling completed as of the date of this report. Although the amount and grade of additional resources cannot be stated with any confidence, most of these additional resources are expected to be of a grade similar to that intersected to date and are expected to be added through additional step-out drilling down dip on the Matinenda Formation.

17.2.5 GENERAL MINERAL RESOURCE ESTIMATION PROCEDURES

The polygonal model Mineral Resource estimate procedure included:

- importing/compiling and validation of data from Microsoft Excel to Gemcom GEMS v6.2.4 to create a Project database;
- statistical analysis;
- validation of geological model for use as resource envelope;
- compositing assay intervals within the mineralized boundaries - limited to one composite per hole;
- extruding polygons around each drill collar with a radius of 140 and 280 metres, and assigning thickness' equivalent to individual composite lengths;
- reporting volumes and grade in each of the extruded polygons; and,
- categorizing the Mineral Resources according to NI 43-101 and CIM definitions.

WGM believes that its drill hole database is insufficient in size and number of samples to develop reliable variography data to guide the selection of a search ellipse for the Teasdale Deposit. In selecting its maximum polygon size, however, the authors note that Pele disclosed several resource estimates for the Eco Ridge Deposit in which it most recently reports variogram ranges for U_3O_8, Nd and Dy of 375 m for the Main Conglomerate Bed and 450 m for the Hanging Wall Zone. The Pele resource estimate is classified according to a maximum search ellipse of 100 m for Indicated Resources (maximum drill hole spacing of 200 m). Inferred Resources were estimated with a maximum search ellipse of 300 m – the average hole spacing for Inferred Resources was 320 m for the Main Conglomerate Bed and 510 m for the Hanging Wall Zone (Cox, Ciuculescu, Altman and Hwozdyk, 2012).

Historical mining practices demonstrated that a spacing of several hundred metres could be used to predict grade. In light of the geological nature of the deposit, especially its great lateral continuity, a polygonal radius of 140 was used by WGM for defining the area of influence for Indicated Resources. The Inferred Resources were calculated with a similarly defined maximum polygonal radius of 280 m. We believe this approach to be balanced and conservative given the approach using a considerably larger database for the Eco Ridge estimate.

17.2.6 DATABASE

17.2.6.1 General

The data used to generate the Mineral Resource estimates originated from original logs provided in Microsoft Word and PDF file format, as well as assay results in Excel format supplied by Appia, which WGM then consolidated into a single data source. A GEMS project was established to hold all of the reformatted data to be used in the computations necessary for the Mineral Resource estimate.

The Teasdale drill hole database consisted of the six 2011 Appia drill hole collar locations in the UTM co-ordinate system and geological descriptions (holes Q-07-01 to Q-07-3, and Q-08-04 to Q-08-06), along with the eighteen 2012 drill holes numbered AEC12-01 through 16. The database consisted of key data such as drill hole collar, survey, assay, and lithological information as well as geological codes and 1573 assayed intervals containing values for TREE (%) and lbs U_3O_8/ton (and other elements including: Th (ppm), ThO_2 (%), LREE (%), HREE (%), La (ppm), Ce (ppm), Nd (ppm), Gd (ppm), Y (ppm), Pr (ppm), Sm (ppm), Eu (ppm), Tb (ppm), Dy (ppm), Ho (ppm), Er (ppm), Tm (ppm), Yb (ppm), Lu (ppm), and Hf (ppm). Assay intervals averaged 0.29 m in length, with the smallest interval measuring 0.03 m and the largest measuring 0.62 m. Lithological cross-sections of each of the drill holes were supplied in PDF format, as well as original digital assay certificates as supplied by Actlabs of Ancaster, Ontario.

Like with the Banana Lake sampling previously report in workman and Breede (2011),, the distribution of assay intervals within the various rock type units favoured the thicker Upper Reef (497 samples) versus the somewhat thinner Lower Reef (347 samples) and the Intermediate Quartzite (236 samples).

17.2.6.2 Data Validation

Upon receipt of the data, WGM performed the following validation steps:

- ✓ checking for location and elevation discrepancies by comparing collar coordinates with the copies of the original drill logs received from the site;
- ✓ checking minimum and maximum values for each quality value field and confirming/modifying those outside of expected ranges;
- ✓ checking for inconsistency in lithological unit terminology and/or gaps in the lithological code;
- ✓ spot checking original assay certificates with information entered in the database; and,
- ✓ checking for gaps, overlaps and out of sequence intervals for both assays and lithology tables.

WGM found the database to be in good order and suitable for use in its subsequent Mineral Resource estimate. Some gaps or missing intervals identified were due to unsampled and unassayed intervals outside of the mineralized zones. Suspect data comprised erroneous survey data for holes AEC12-07 and AEC12-16, as well as manual data entry errors in the assay and lithological interval logs which WGM corrected by manually cross-checking the entered data against the original hand-written drill logs. In the case of the survey data, it was revealed that in-field survey QA/QC protocols were not always followed, resulting in some poor-quality survey data[11]. Consequently, the survey results used for the final interpretation were interpreted from the collar orientations, and selected down-hole survey readings. The survey results that were not rational based on WGM's experience were ignored. The degree of uncertainty surrounding the geometry of these two holes, particularly AEC12-16, necessarily resulted in a downgrade of mineral resource categorization, as described later in this section.

17.2.6.1 Database Management

The drill hole data were imported into a GEMS multi-tabled workspace specifically designed to manage collar and interval data. The project database stored cross-section and level plan definitions, such that all data pertaining to the project are contained within the same project database. A copy of the project database is stored in WGM's servers in Toronto.

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[11] Down-hole survey data is affected by the ambient magnetic properties of the surrounding rocks as well as the conditions in the drill-rod string and operator error. Data that is suspect should be ignored rather than being entered into the drill hole record (as it was), and the surveys should be repeated at a slightly different depth.

17.2.7 GEOLOGICAL MODELLING PROCEDURES

A single inclined section was defined for the Teasdale Zone which closely paralleled the dip of the mineralized zone. The inclined plane dips about -17° SSW, with a strike of 281° to the NW. Figure 20 shows the drill hole intercepts in 3D and the relative position of the inclined plane in 3D space.

17.2.8 GEOLOGICAL INTERPRETATION

The mineralized zones used for the resource estimate are defined by the volume between the upper surface of the highest reef and the basal contact of the lowest reef, according to Appia's designations. These intervals are as follows:

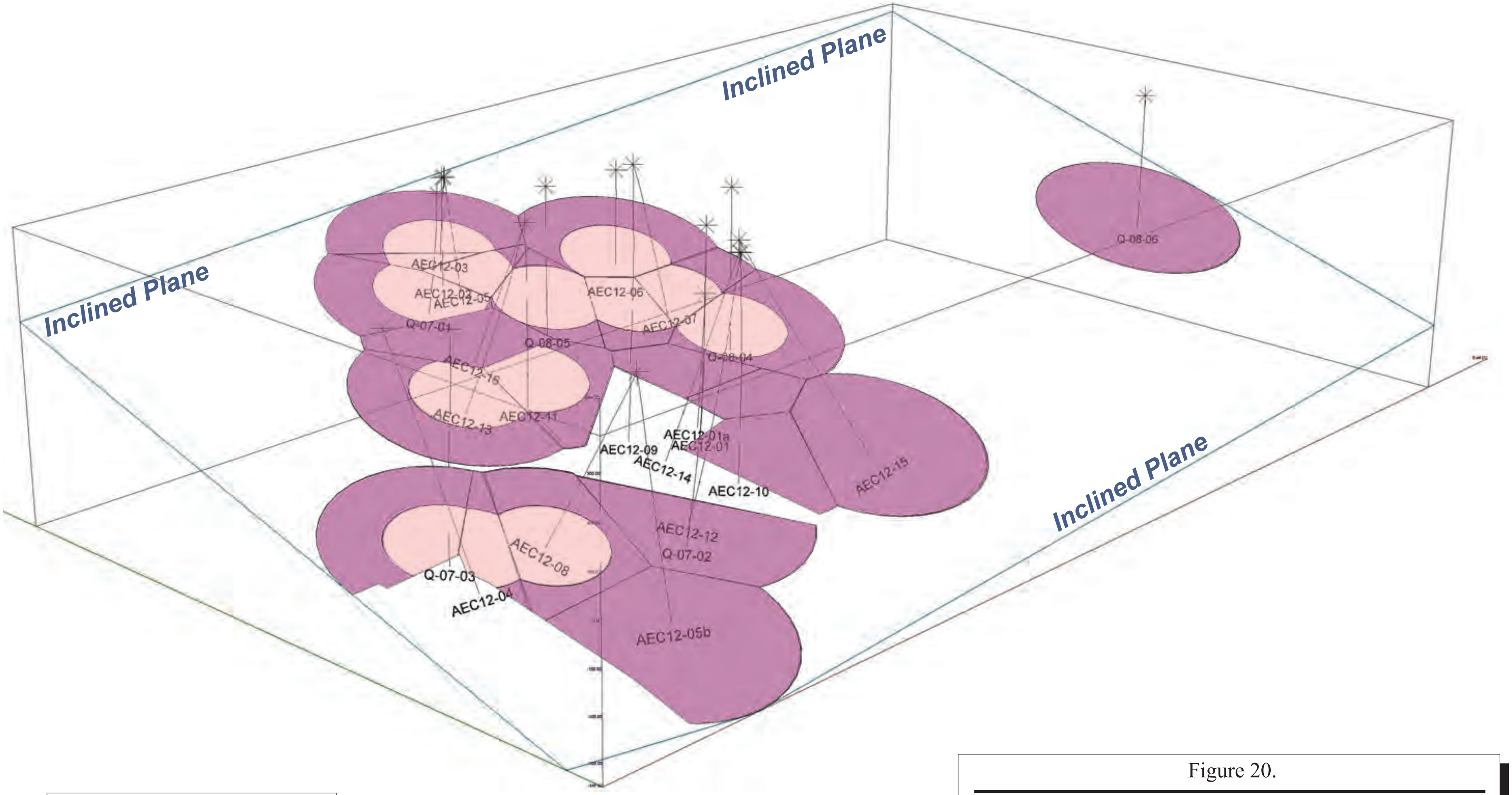
2007-08 Drilling Program

Q-07-01	239.63 m	to	248.70 m
Q-07-02	540.64 m	to	551.80 m
Q-07-03	480.78 m	to	493.80 m
Q-08-04	349.05 m	to	354.00 m
Q-08-05	292.69 m	to	302.90 m
Q-08-06	324.61 m	to	333.31 m

2012 Drilling Program

AEC12-01	461.53 m	to	471.66 m
AEC12-01a	454.43 m	to	464.42 m
AEC12-02	182.25 m	to	191.48 m
AEC12-03	175.17 m	to	180.15 m
AEC12-04	556.44 m	to	565.83 m
AEC12-05	215.53 m	to	225.39 m
AEC12-05b	615.64 m	to	625.02 m
AEC12-06	209.88 m	to	219.39 m
AEC12-07	354.62 m	to	364.75 m
AEC12-08	679.17 m	to	690.08 m
AEC12-09	327.33 m	to	337.07 m
AEC12-10	514.52 m	to	519.30 m
AEC12-11	395.18 m	to	405.07 m
AEC12-12	563.82 m	to	575.11 m
AEC12-13	444.72 m	to	451.87 m
AEC12-14	530.00 m	to	533.00 m
AEC12-15	617.20 m	to	628.24 m
AEC12-16	392.18 m	to	402.77 m

APE REV / APE_15_3D_Model.cdr
Last revision date: Thursday 25 July 2013



Legend:

Mineral Resource Categorization

- ▢ *Indicated*
- ■ *Inferred*

Graphics by Watts, Griffis and McOuat Limited

Figure 20.

APPIA ENERGY CORP.

Elliot Lake Property

Ontario, Canada

3D View Looking NE Showing Drill Holes Through the Teasdale Mineralized Zone

Watts, Griffis and McOuat

17.2.9 TOPOGRAPHIC SURFACE CREATION

A topographic surface or triangulated irregular network ("**TIN**") was generated using collar elevations of the holes drilled from surface for the entire Teasdale Zone. This was not seen as being crucial for this stage of the Mineral Resource estimate, as the zones would only be mined by underground methods.

17.2.10 STATISTICAL ANALYSIS, COMPOSITING, CAPPING AND SPECIFIC GRAVITY

17.2.10.1 Statistical Analysis and Compositing

The original assay intervals varied in length, requiring normalization to a consistent length in order to carry out meaningful geostatistics. A set of equal length 1-metre composites was generated from the raw sample intervals within the resource limits. A total of 212 composites were generated. The statistics of the composites inside the defined mineralized zones for TREE and U_3O_8, which were used for the Mineral Resource estimate, are summarized in Table 26. For its grade distribution analysis, WGM examined the zones as a whole. The results of this study are illustrated in Figures 21 and 22.

The statistical distribution of TREE and U_3O_8 shows relatively good lognormal distributions.

Table 26
Basic Statistics of the One-Metre Composites

Zone	Number	Mean TREE (%)	Mean U3O8 (lbs/ton)	C.O.V.* (TREE)	C.O.V.* (U_3O_8)
Upper Reef	103	0.201	0.462	0.51	0.63
Intermediate Quartzite	55	0.110	0.291	0.52	0.69
Lower Reef	54	0.154	0.943	0.51	0.69

*Co-efficient of Variation

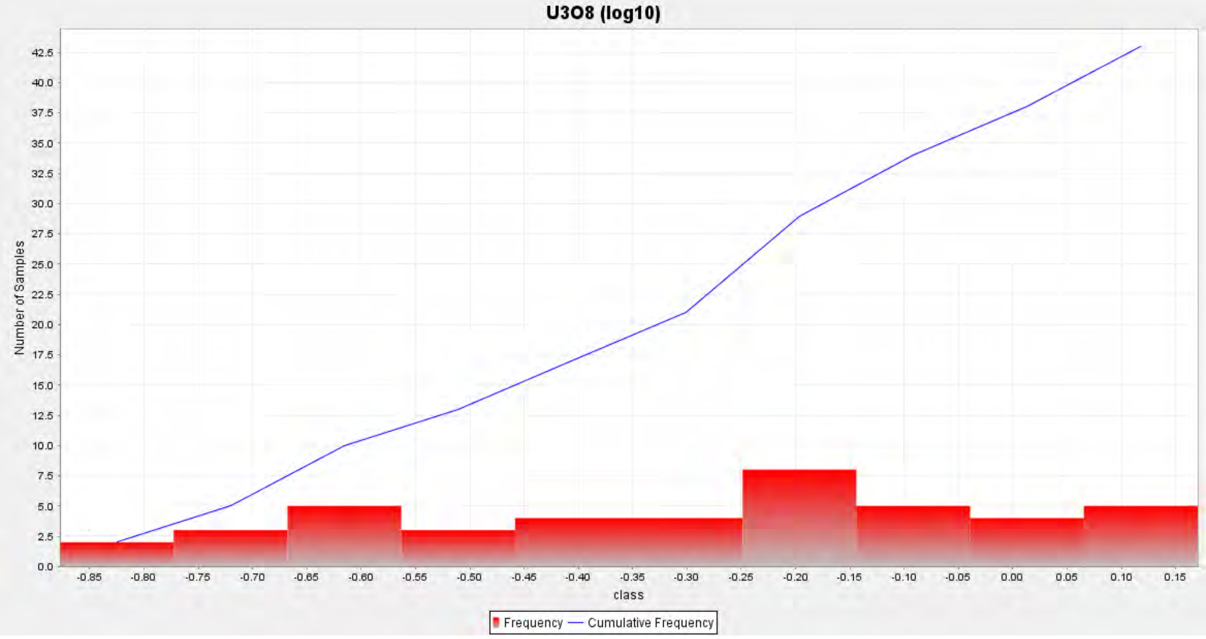


Figure 21. LOG normal histogram, U_3O_8 composites within the entire mineralized zone.

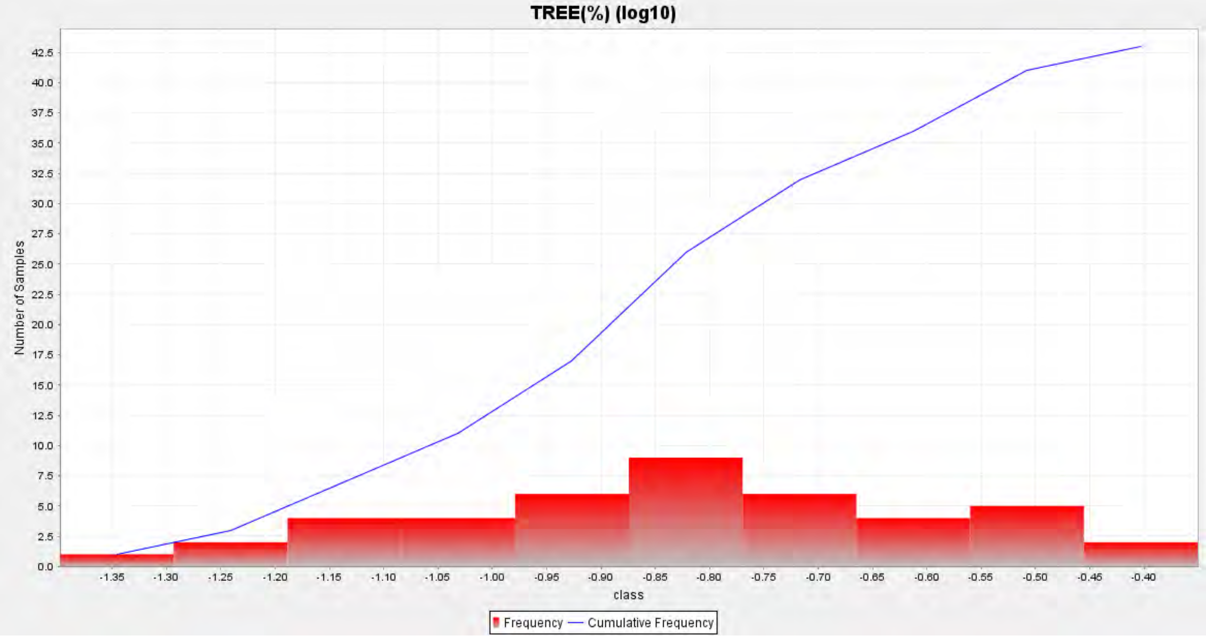


Figure 22. LOG normal histogram, TREE 1-metre composites within the entire mineralized zone

17.2.10.2 Cut-Off Grade and Grade Capping

In its previous report, WGM concluded that the 2008 Preliminary Economic Assessment ("**PEA**") prepared for Pele's Eco Ridge deposit provided useful guidance to Appia concerning its exploitation of the Teasdale mineralization. Specifically, the PEA provided reassurance that, given the current metal price forecasts applicable within a reasonable development timeframe, the grade of the Teasdale uranium-REE mineralization was sufficient to support an economically viable mining operation. WGM concluded that the challenge for Appia would be to demonstrate that sufficient tonnage exists to justify mine development. WGM did not use a cut-off grade in the 2008 Teasdale resource estimate as the value-matrix of the U and REE contents would be quite complex to model. For this current estimate, as with Eco Ridge, WGM has opted to impose a "value" cut-off grade in its estimate which is akin to a "grade equivalence" to account for the combined grades and value of the U and REEs. WGM's review of the REE and U data indicated that the grades were sufficiently robust and continuous to support mining the entire reef section as a single minable zone, whereas historical mining focused mainly on the Lower Reef section. The variability between individual REEs also favoured a focus on TREE content rather than individual metals, and consequently supported WGM's use of geological constraints rather than a specific cut-off grade.

One major consideration in determining future cut-off grades would be whether or not the ore from this deposit could be processed in a central milling facility that would accommodate neighbouring mining operations in the Elliot Lake camp. This would significantly reduce capital and operating costs. It is clear that a PEA of the Teasdale Zone is needed to better review mining and processing options.

While the resources have been constrained for the resource estimate solely by geological marker horizons (boundaries), the hanging wall and footwall zones, immediately above and below the mineralized zone, include assays greater than or equal to 500 ppm TREEs and/or 200 ppm U. Much of this low grade material will likely be considered internal dilution for bulk underground mining (e.g. room and pillar). These parameters were chosen based on a preliminary review of the parameters that would likely determine the economic viability of an underground mining operation and comparison to similar projects in the area that are currently being mined or are at an advanced stage of study / development.

Due to the low composite sample population, there is insufficient data to support the use of high-grade capping at the Teasdale Zone. Grade capping, also sometimes referred to as top-cutting, assay grades is commonly used in the Mineral Resource estimation process to limit the effect (risk) associated with extremely high assay values since high-grade outliers can contribute excessively to the total metal content of the deposit. Philosophies or approaches to establishing and using a grade cap is variable across the industry and includes, for example, not using grade caps at all, arbitrarily setting all assay grades greater than a certain value to a high grade "limit", choosing the grade cap value to correspond to the 95 percentile in a cumulative distribution, evaluation of Mean Grades + multiple levels of Standard Deviations and the evaluation of the shape and values of histograms and/or probability plots to identify an outlier population. Another rule of thumb is to set the capping level to lower the top 10% of the metal content in the deposit. WGM recommends that further geostatistical investigation be conducted as new drilling data becomes available, however, there is no historical basis for high-grade capping at the operating mines given the laterally continuous nature of the mineralization. Also, the low coefficient of variation ("**C.O.V.**") for both TREE and U_3O_8 1-metre composites would suggest that top-capping is unnecessary. Typically, capping is only warranted if the C.O.V. is above 1.0.

The statistical distribution of TREE shows relatively good lognormal distributions, whereas U_3O_8 appears to be exhibit a more bi-modal distribution.

17.2.10.3 Density / Specific Gravity

A specific gravity factor of 2.85 tonnes per cubic metre (3.14 tons/m^3) was used for volume conversion based on 14 samples tested by Appia at the Actlabs laboratory. WGM has accepted this SG as an approximation as it compares favourably with those from similar deposits in the Elliot Lake area (and was the basis of WGM's 2008 and 2011 resource estimate).

WGM recommends that the SG results, like all assays, should also be stored in an assay database table for ease of use and comparison purposes.

17.2.11 POLYGONAL MODEL PARAMETERS, GRADE INTERPOLATION AND CLASSIFICATION OF MINERAL RESOURCES

17.2.11.1 General

The Mineral Resources have been estimated using the Polygonal method whereby a circular area of influence is assigned to each drill hole composite, and from which a volume can be calculated using the true thickness of the composite interval.

17.2.11.2 Polygonal Model Set-Up and Parameters

The polygonal model was created using the GEMS v.6.2.4 software platform to create two sets of polygons around each drill hole composite. The first set of polygons was generated based on a 140 m radius of influence and the second set on a 280 m radius. The area of the polygon was determined by the area of influence deemed appropriate for the individual drill hole based on drilling density. The thickness of the polygons, and thus volume, was determined by the hanging wall and footwall contacts of the composite.

Polygon data, including area, volume, density, tonnage, grade and hole-id, was stored in a multi-tabled workspace in GEMS.

17.2.11.3 Grade Interpolation / Bed Composites

Variograms were generated in an attempt to characterize the spatial continuity of the mineralization in the defined zones, however, due to the relatively small assay database, meaningful variograms could not be computed. The long history of mining of the Elliot Lake uranium deposits provides for a very clear understanding of the geology and the geometry of the mineralization. Consequently, the area of influence and orientation of the polygons were based on this geological knowledge, as opposed to variograms. Grades were thus assigned to the polygons based on a single length-weighted average bed composite as described in Section 17.4.2.

17.2.11.4 Mineral Resource Classification

To categorize the Mineral Resources, WGM classified each of the smaller polygons (140 m radius) as Indicated, and the larger sets of polygons (280 m radius) as Inferred. Smaller polygons which did not intersect adjoining smaller ones, were automatically downgraded to the Inferred category due to insufficient drilling density, thus eliminating the less than ideal "bull's eye" effect. In the case of drill hole AEC12-16, the associated Indicated Resource polygon was downgraded due to poor survey data as described earlier.

All drill holes were included in the resource estimates as the inclusion of the Upper Reef ("**UR**") and the Intermediate Quartzite ("**IQ**") with the Lower Reef ("**LR**") meant that none failed to exceed the minimum 2.44 m vertical thickness used historically when only the LR was mined. WGM recommends that subsequent studies on the Property include preliminary underground mining studies to determine the appropriateness of a significantly greater (+/- 9 m) mining height in light of recent developments in back stabilization and the design of mining equipment for moderately inclined orebodies. Such studies should also consider the potential for losses in mining recovery due to mineralized rock left in situ as supporting pillars.

Resource polygons associated with drill holes AEC12-12 and 14 were assigned zero grade and volume as a result of ground faulting and insufficient mineralization. The polygons associated with AEC12-1 and 1a were downgraded to Inferred Resources due to their close proximity to these holes. For illustration purposes, these zero grade polygons have been omitted from the figures below. WGM suggests that additional drilling be carried out near this cluster of holes to determine the exact nature of the mineralized disturbance/ground faulting.

The Mineral Resource estimates contained herein do not account for mineability, selectivity, mining loss and dilution.

Figure 23 shows the interpolated polygons and categorization on the inclined plane. The visual comparison of polygonal model grades with the 1-metre composite grades shows a reasonable correlation between the values. The orientation of the polygons follows more or less the plane of mineralization. At this early stage of the Teasdale resource model, it is doubtful that block modelling of the resource would significantly improve the interpolation.

APE REV / APE_14_Inclined_Plane_Grade_Model.cdr
Last revision date: Thursday 25 July 2013

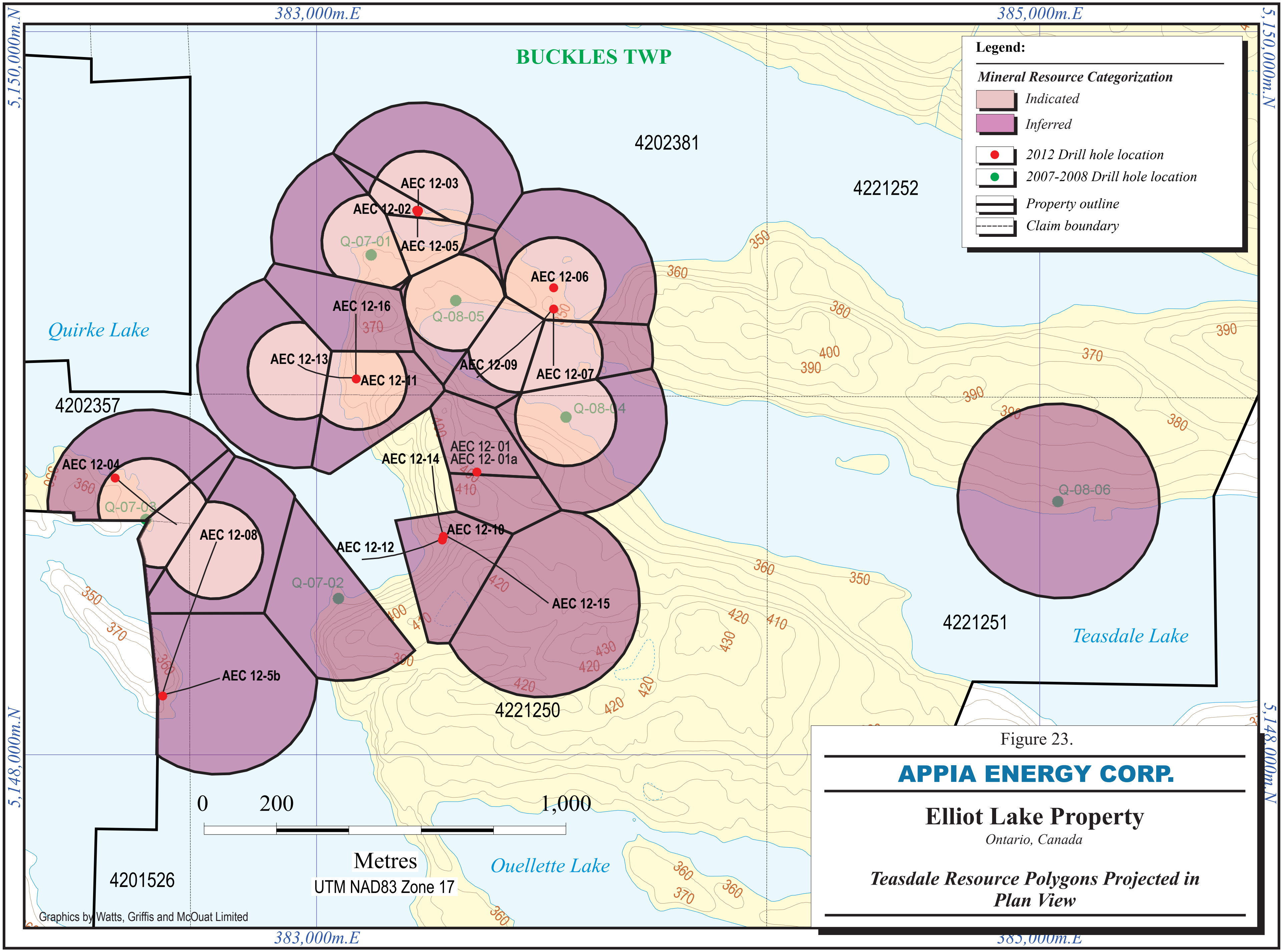


Figure 23.

APPIA ENERGY CORP.

Elliot Lake Property

Ontario, Canada

Teasdale Resource Polygons Projected in Plan View

Watts, Griffis and McOuat

17.3 BANANA LAKE ZONE

WGM refers the reader to the 2011 Technical Report which details the NI 43-101 compliant Mineral Resource estimate for the Banana Lake zone (Workman and Breede, 2011). The Mineral Resource estimate is based on a total of seven (7) diamond drill holes, the results of which are summarized in Table 27.

Table 27
Banana Lake Zone Mineral Resource Estimate - April 1, 2011
(using 0.6 lb U_3O_8/t cut-off)

Category	Tons (‘000)	S.G. (tons/m^3)	lb U_3O_8/t	Total lbs U_3O_8 (‘000)
Inferred Resources	30,315	3.14	0.912	27,638

Notes:

1. Effective April 1, 2011
2. Mineral Resources which are not Mineral Reserves do not have demonstrated economic viability. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.
3. The quantity and grade of reported Inferred Resources in this estimation are uncertain in nature and there has been insufficient exploration to define these Inferred Resources as an Indicated or Measured Mineral Resource and it is uncertain if further exploration will result in upgrading them to an Indicated or Measured Mineral Resource category.
4. The Mineral Resources were estimated using the Canadian Institute of Mining, Metallurgy and Petroleum standards on Mineral Resources and Reserves, Definitions and Guidelines prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council November 27, 2010.
5. S.G. of 2.85 tonnes/m3 (or 3.14 tons/m3) was used.
6. All tonnage and total lbs U_3O_8 amounts rounded to nearest thousand or thousandth. Totals may not add up due to rounding

No additional work has been carried out on the Banana Lake Zone by Appia since the disclosure of the foregoing resource estimate.

17.4 ESTABLISHED HISTORICAL PRACTICES

WGM was unable to find information concerning resource and reserve estimation practices in the Elliot Lake mines. A document was reviewed concerning the practices used at the Agnew Lake Mine where "geological reserves" were calculated using a cut-off grade of 0.75 lbs U_3O_8 per ton (0.38 kg/t). "**Proven Reserves**" were restricted to reserves located within 200 feet (61 m) of underground workings and were developed on two or more sides (Agnew Lake Mines, 1980). "**Probable Reserves**" were uranium-bearing beds located within 200 ft (61 m) of workings, but were only developed on one side, or alternatively, were uranium-bearing beds with drill hole intersections less than 400 ft (122 m) apart. "**Inferred Reserves**" were defined as uranium-bearing beds with drill hole intersections greater than 400 ft (122 m) apart.

WGM is of the opinion that the foregoing classifications correspond to the CIM-equivalent definitions for Proven Reserves, Probable Reserves and Inferred Resources. WGM's discussions with former Elliot Lake mine workers, Bob MacGregor, P.Eng. and Alan MacEachern, P.Geo., lead it to conclude that similar estimation practices were used for the Elliot Lake mines. The reliance on data from widely spaced drill holes was common practice at the time, and supported by the uniformity of the ore and its stratiform character. WGM is of the opinion that the aforementioned spacing of data points adequately supports the foregoing classifications given the excellent grade and thickness continuity of the Elliot Lake ores.

Although WGM has not seen specific mention of a dilution grade used for the conversion from geological reserves to minable reserves, a document by Agnew Lake Mines Ltd. concerning the methods used in the reserve calculation, dated 18 January, 1980, indicates that the practice at the Agnew Lake Mine was to use a zero grade for dilution purposes (Agnew Lake Mines, 1980). According to MacEachern, the grade of the hanging wall (HW) and footwall (FW) dilution at the Denison Mine ranged from 0.00 to 0.60 lbs U_3O_8/ton depending somewhat on the lithology of the wall rock. The amount of dilution from the hanging wall HW + FW depended on the particular reef(s) being mined and the mining method used, with 10% being an overall average (total dilution). The grade of the dilution at Denison ranged from 0.00 to 0.60 lbs U_3O_8/ton depending somewhat on the lithology of the wall rock. In respect to current exploration targets, room and pillar mining of the >6 m Lacnor Reef would allow for dilutions of approximately 5 - 7% whereas mining of the 9-metre thick Teasdale Lake Zone would necessitate lower dilutions.

18. ENVIRONMENTAL

WGM believes that a review of the current status of environmental restoration and impact mitigation activities in the Elliot Lake is needed to ensure that Appia is fully informed regarding the potential collateral costs of mining in this area.

Approximately 2 years after the start of mining termination operations in the Elliot Lake camp, the Atomic Energy Control Board ("**AECB**") decided in October 1992, that proposals submitted by Rio Algom Limited and Denison Mines Limited for decommissioning several sites should be referred to the Minister of Environment Canada for public review. At this time, the decommissioning of mining projects was becoming more and more of a concern due to escalating costs to the Canadian public for the clean-up up of "heritage" sites. Given the fact that the Elliot Lake projects involved uranium, a commodity that was enshrouded in political intrigue and a certain degree of societal fear, and the newness of the public review process, the decommissioning of the Elliot Lake uranium tailings was foreseen as a major undertaking.

Plate 14: Aerial views of the Denison Mine before and after reclamation (from Canadian Nuclear Safety Council (photos from the Denison Environmental Services website).

In hindsight, the engineering of the decommission process appears to have reflected a desire to eradicate any memory of the previous mining at Elliot Lake without any regard to the value of the remaining uranium resource, and without any consideration for the future development of this resource. In today's context, this approach seems short-sighted, however the author of this report vividly remembers the atmosphere prevailing at the time as one of relegating Elliot Lake to the past and looking towards the future through developments in the Athabasca Basin, Saskatchewan.

During early 1993, the terms of reference and operating guidelines for the review were proposed and a review panel was established. In recognition that the Ontario Government had accepted the responsibility for environmental remediation work at many of the Denison and Rio Algom mines[12], the review was limited to Denison's proposal to decommission its Denison and Stanrock mine tailings facilities and Rio's proposal to decommission its Quirke and Panel mine tailings facilities.

By October 1993, draft guidelines had been issued by the panel for the preparation of an Environmental Impact Statement ("**EIS**") by each proponent. These statements require a description of the existing Elliot Lake tailings management areas, the proposed method for long-term management of the tailings, and the potential environmental and health impacts of these proposals.

An EIS was expected early in 1995 from each of Denison and Rio Algom. Intervener Funding was provided to various public groups to allow them to adequately prepare and present input into the environmental review process. In addition to the release of radioactive nuclides such as radon and radium, one of the major concerns was the potential for acid generating tailings and waste to contaminate the local watershed. Some of the other concerns were as follows:

- drying of the tailings and contaminant dust generation;
- uncovering and oxidation of the tailings and resultant acid generation;
- water containing solubilized heavy metals seeping out and entering the Serpent River water system, and related impacts on water quality, sediments and fish; and,
- excessive inadvertent exposure of intruders, human and wildlife, to gamma radiation from tailings surfaces.

In the case of tailings treatment, water cover to a minimum depth of 5 cm was seen as an important means to prevent dust generation and oxidation. According to Health Canada, predictive modelling suggested a need for the sites to be monitored for the next 1,000 to 10,000 years beyond the present based on the long term radiological impact of the tailings, primarily driven by the 1,600-year half life of Ra-226.

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[12] As the mining operations drew down economically exploitable resources, it was apparent that only the high prices negotiated under long term sales agreements with Ontario Hydro could keep the Elliot Lake mines in operation. Public pressure forced Ontario Hydro, an Ontario Crown Corporation, to exercise options to terminate its contracts with Denison and Rio Algom. In exchange, the Ontario Government agreed to assume the responsibility for site remediation work at several, but not all, of the former producers' mines.

Health Canada issued a summary of its finding in a summary document entitled "Decommissioning of the Quirke/Panel & Stanrock/Denison Uranium Mine Tailings Management Areas in Elliot Lake, Ontario". It contained the following statement dated November 16, 1995 by Mr. David Grogan of Health Canada:

> "Based on facility design and expected performance, chemical contaminants are not expected to be an issue. While some exposure parameters (e.g. drinking water consumption, air intake, fish consumption) are not properly used, they do not appear to impact negatively on the choice of the decommissioning option. The conclusion that chemical contaminants are not an issue is dependent upon the facility operating as expected. Monitoring of the performance is essential to ensure water quality is not adversely impacted. No significant radiological impacts are anticipated from the decommissioning options chosen by the proponents."

The Elliot Lake environmental assessment panel submitted its recommendations to the Federal Government during June, 1996 concerning the plans by the two mining companies to decommission the mill tailings sites. The panel agreed with the decommissioning proposals set out by both companies. The review panel recommended certain conditions for closing and reclaiming the Quirke, Panel, Denison and Stanrock tailings facilities.

Nearly a year later, in April 1997, the Federal Government agreed that the proposals submitted by Denison and Rio Algom, and the panel's recommendations should form the basis of the decommissioning licences for the uranium waste management areas. Approval was recommended for the licensing process to proceed, and at the end of the year decommissioning was proceeding at the specific sites in compliance with regulatory guidelines.

Decommissioning operations occurred from 1990 to 1998, and by late 1999 the major site decommissioning and reclamation work on the Rio Algom facilities (Stanleigh, Quirke, Panel) and the Denison facilities (Stanrock/Can-Met, Denison) was essentially completed. Waste management and tailings management areas were stabilized, and most have been flooded (tailings at Stanrock were saturated to reduce acid generation but have a dry cover). Planned interim monitoring and active management will be maintained until the effluent meets discharge criteria without treatment. At that time, the sites will enter into a phase of long-term monitoring with care and maintenance.

During April 1999, the AECB amended the decommissioning licences for the Denison and Stanrock mines to expand the site boundaries to include areas identified in 1998 that exceeded the clean-up criteria established for the two sites.

In addition to the Rio Algom mines covered under the AECB decommissioning licence, the Spanish American, Milliken, Lacnor, Nordic, Buckles and Pronto Mine facilities were not licensed for remedial action. These facilities were licensed by Rio Algom during 1995 to meet AECB requirements to control radioactive materials. Rio Algom submitted an environmental assessment report in 1999 that was reviewed that same year. During 2000, a revised report was expected to be submitted to the AECB with licensing approval expected later that year.

Rio Algom and Denison Mines are also monitoring the Serpent River and its watershed to assess the environmental impacts of their operations and tailings facilities on the entire Serpent River system. Up to the end of 2004, Denison and Rio Algom have committed over $75 million to decommissioning and waste management. Periodic sampling of background and receiving waters is carried out, as well as studies every five years on the biota in the watershed and the man-made tailings environments. Water quality monitoring is on-going. Collective annual costs to Denison and Rio Algom for these activities are approximately $2 million.

The first of the five-year biota assessments was completed in the fall of 1999 and a report was issued in 2000. The biological activity shows that the decommissioning efforts have been successful. Near-field environmental impacts on the watershed were detectable, as expected, in the form of above-background levels of salts, total dissolved solids and some metals. Nevertheless, the local fish, benthic invertebrates and wildlife displayed no adverse effects. The second stage of data collection was completed during 2004 and a report summarizing the findings was released in 2005.

The sustained effort to restore the Elliot Lake watershed to its original condition has been costly for all concerned. Any mine development activities by Appia should be undertaken in this context. While the previous mine sites represent areas already impacted by industrial activity, new mine development will likely draw attention which may be disproportionate to the impacts contemplated by mine planners. First Nations communities that are located in watersheds down-stream from mining operations may spawn groups that are especially resistant to uranium mining. Alternatively, Appia’s engagement of such groups may define substantial grounds for co-operation since the previous mining activities have not produced adverse health consequences although there have been unsubstantiated claims to the contrary.

Notwithstanding opposition to a renewal of uranium mining in the Elliot Lake area, many of the residents of Elliot Lake may in fact welcome the industrial activity as a means of increasing property values and stimulating the local economy. The overall success of the site restoration and impact mitigation work carried out by the Province and by the miners should be seen as clear evidence that impacts can be managed in an environmentally responsible manner. Careful and enlightened interaction between company representatives and the public is required. The Canadian Nuclear Safety Council, in considering Rio Algom's application concerning consolidating the site management under its existing Waste Facility Operating Licence, chastised Rio Algom for not doing enough to keep the Public informed of its activities (Canadian Nuclear Safety Commission, 2004). Rio Algom has responded with bus tours of the sites several times a year and biannual inserts in the local newspapers up-dating the community on its efforts.

As of the date of this report, Denison and Rio Tinto maintain a joint webpage at http://www.denisonenvironmental.com which is a Denison Environmental Services ("**DES**") webpage providing information on various mining legacy projects undertaken by the company. DES provides information concerning the history and rehabilitation of the former uranium mine sites in the Elliot Lake area. The website contains various presentations made in respect to DES activities to local communities and up-dates/maintains means of communication between residents and DES.

Clearly, future uranium production from this area will require an inclusive approach that builds upon the experiences of the past. There is no compelling environmental reason why uranium mining cannot resume in the Elliot Lake area and indeed the local community appears to want the employment opportunities that such activities would bring. This view has been echoed to some extend by officials in the Ontario Ministry of the Environment. Elliot Lake town officials have recently proposed the sale of building lots in the greater Elliot Lake area, including the Quirke Lake area, and Appia needs to ensure that all parties are adequately informed concerning its mineral exploration activities and the mineral development rights that accrue to its mining claims. Appia should explore alternatives to conventional surface milling and processing that would allow sub-surface leaching as was used in the past and/or back-filling mine openings with tailings.

19. OTHER RELEVANT DATA AND INFORMATION

19.1 GENERAL NOTES ON THE URANIUM MARKET

In Workman and Breede (2011), WGM presented a review of Canada's energy history as it applies to the commodity market for uranium. The reader is encouraged to read this document which is only summarized here.

The uranium market is neither transparent nor well publicized in the same sense that information on the copper or gold markets is easily obtained. The market, such as it exists, is dominated by transactions negotiated in private between uranium miners and nuclear fuel producers and energy utilities. The market is heavily leveraged on public acceptance of nuclear electric energy, and major reactor incidents can have strong short term effects on uranium pricing. The uranium market is also very politicized, and it can be difficult to unravel the events by which the market has developed. Great attention is commonly focused on the uranium Spot Price, yet the spot market is the discount market where discretionary (and usually small) purchases are made. The casual observer is consequently presented with rapidly up-dated information on Spot Market pricing which can be quite volatile. Most energy utilities seek the security of assured uranium supply through long-term contracts which are seldom made public, and Term Market pricing is consequently more stable as a result. The surging market for uranium seen in 2006 and 2007 was very similar to that which existed 40 years earlier. The international reaction to the disaster following the Japanese tsunami is uncannily similar to that which followed the Three Mile Island accident, though the scale is hardly comparable. Other aspects of the uranium industry are very different. With some knowledge of the historical legacy comes some understanding of the current potential for escalating prices in the uranium market which are of vital importance to projects such as that envisioned by Appia.

There have been several instances of rapidly escalating uranium prices in the past. As Workman and Breede noted, following the deregulation of uranium prospecting in 1948, the Government of Canada set a minimum U_3O_8 purchase price of $2.75 per pound, guaranteed for a period of five years, and by 1956 more than 10,000 new radioactive occurrences were reported to the Atomic Energy Control Board of Canada. Despite an embargo imposed by the United States of America ("**US**") in 1967 to protect US domestic uranium producers from foreign suppliers, the growing demands of new nuclear-electric utilities and government stockpiling programs resulted in a rapid global escalation of uranium prices during the 1970s which encouraged exploration. In infamous Uranium Cartel that Canada participated in with France and Australia was effective in establishing a U_3O_8 floor price for only 18 months in

1972-73, after which the global price escalated beyond the cartel's $4.90 benchmark. The uranium price increase occurred during a time of soaring oil consumption, as well as rapidly escalating oil prices as a result of the founding of OPEC and the increased power of those member states to establish production schedules and floor prices for petroleum beginning in 1973. It is difficult to believe that in March, 1970, a US Presidential Proclamation had restricted the flow of Canadian oil into US markets. The linkage of uranium prices with oil prices is imperfect, but nevertheless convincing in the longer term.

In June, 1976, the Canadian Government announced revised uranium resource estimates in a report entitled "1975 Assessment of Canada's Uranium Supply and Demand", using a maximum price of US $40 per pound of U_3O_8, equal to the Spot Market price at the time. The actual market price for uranium remained above this threshold until January, 1980, averaging $42.29 during this 43-month period. During the same timeframe, the Term Market price averaged $42.62, and neither the Spot nor the Term prices seemed to be much affected by the Three Mile Island nuclear accident in Pennsylvania on 28 March, 1979. In fairness, this was a minor accident resulting in no loss of life, and even 35 years later no illnesses have been attributed to the event. A minor release of radioactivity occurred in the form of steam charged with radioactive krypton gas (Kr_{85}) and minor amounts of iodine (I_{131}) even though the plan suffered a partial meltdown. Ironically, this relatively minor incident was widely publicized internationally. The movie "The China Syndrome", concerning a nuclear disaster, was released just 12 days before the incident and became a blockbuster. The accident and the movie combined to have far-reaching effects on public opinion, particularly in the United States where construction plans for several reactors were cancelled and public concern increased about society's dependence on nuclear reactors for electrical energy.

During the 1970s, the price support mechanisms implemented by Canadian and other governments had a profound effect in stimulating uranium exploration and mining. Although the effects of over-stimulation were recognized well in advance, and the accumulating global stockpile of uranium in various forms was considerable, the rapid downturn in prices at the end of 1979 and the abandonment of Government support policies was not fully anticipated. The downturn in uranium prices accelerated as economic growth stalled under the weight of high interest rates in 1981-83, run-away Government deficits, and collapsing commodity prices. The end result of revised energy demand outlooks and cancelled or delayed reactor construction, the accumulated uranium stockpiles have taken more than two decades to consume. The net result was depressed prices which bottomed in October and November, 1991 with a Spot Market price of $7.40 however the industry was sustained by blended pricing structures which prevailed under previously negotiated contracts.

Public sentiments towards nuclear energy have exerted an effect on nuclear programs as had been seen previously as a result of the Three Mile Island accident in Pennsylvania during 1979. The Chernobyl nuclear reactor accident on 26 April, 1986, is widely recognized as a true disaster caused by either operator error or poor reactor design. It directly resulted in 31 deaths and affected tens of millions of people to a greater or lesser degree over a very wide region. Chernobyl may have had a weak affect on uranium prices - over the 2-year period following the accident, the Spot Market price declined 9%, however this should be viewed in the context of what was already a weak and declining market.

The low prices for uranium that prevailed during the late 1980s and the 1990s, together with rising production costs, saw the closure of most of the uranium mines and mills in the US, and the closure of all uranium mines in Canada outside of Saskatchewan. Perhaps nowhere was the change in uranium production more dramatic than in the United States as shown in Figure 24. The collapse in US mine output in response to falling prices was dramatic, largely due to the lower grade of US uranium resources - less than 1,000 tonnes of uranium was produced in 1992. The mills that these mines supplied were also forced to shut down by a combination of feed shortages and cost pressures. During 2004, six mines were operating in the US compared with 430 in 1979. All production was from in-situ leach plants – no conventional uranium mills were operating.

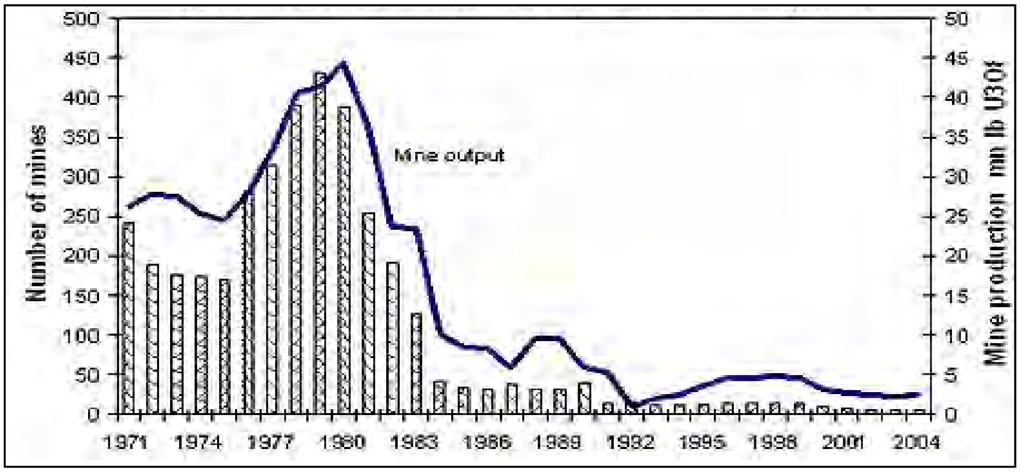

Figure 24 United States Uranium Mine Output and Operating Mines (1971-2004). The vertical indicate the number of operating mines using the scale at left and mine production is shown by the dark line against the scale at right – *from the US AEC.*

The 1990s also saw the gradual depletion of uranium stockpiles. An upwards 'blip' in uranium prices occurred during 1995-96 when the U_3O_8 price rose briefly above $16 before gradually falling off to about $8.75 by November, 1998 and $7.10 by December, 2000.

The 2006-07 rise in uranium pricing did not initially achieve much in the way of recognition. It is possible that years of inattention left the investment community lethargic as the increase was initially thought to be a minor bump similar to that which occurred in 1995. However, the fundamentals affecting the uranium market were very different, mainly due to the exhaustion of the global stockpile, limited growth in uranium production, growing resistance to the conversion of highly enriched uranium ("**HEU**") to nuclear fuel, growing energy requirements in the developing world, and re-emerging demand for nuclear-electric energy sources in existing markets. For the first time, nuclear power was seen as "green" energy as it alone offered the potential to satisfy growing base-load energy demands without contributing to CO_2 emissions.

It is also important, perhaps vitally so, to recognize that the international market is now largely unfettered from the political interferences and protectionism that prevailed earlier, and typified by US actions to protect its own producers. Conventional wisdom holds that hedge funds participated in uranium purchases during 2007 and were largely responsible for the volatility seen in the Spot Market price at that time. The fall in Term contract prices seen during 2008 and the partial recovery seen in 2009 indicates the true strength of the market which at the time appeared to be corrective in the range of \$50 to \$60 per pound of U_3O_8.

Between June 2010 and February 2011, the Spot Market price escalated from \$40.75 to \$72.00 per pound of U_3O_8 and the Term Market price increased from \$58 to \$73 marking gains of 77% and 26%, respectively. The Fukushima Daiichi event on 11 March 2011, the result of a 13-metre tsunami and not a nuclear accident in any real meaning of the term, had a profound effect on public opinion much as the Chernobyl disaster had earlier. Erosion in the confidence of nuclear power led to suspended projects, shut-down reactors, political posturing by anti-nuclear politicians and weakening of uranium prices. One year later, the Spot and Terms prices were \$51 and \$60, for respective declines of 29% and 18%.

Prices in 2012 and thus far in 2013 have been soft, and the uranium Spot Market price according to UxC is US \$39.50 per lb of U_3O_8 as of 10 July, 2013, and the Term Market remains stable at \$56.50 where the price has remained for several months. Uranium transaction data in difficult to acquire on the internet, however the available information suggests that the number of Spot Market transactions thus far in 2013 is substantially lower than in previous years. Together with the stability of the Term Price, the transaction data suggests that producers are either unable to commit surplus supply to the discount market, have no surplus to commit or are unwilling to make supply available at the prevailing price. In any case, the movement towards lower prices appears to have run its course.

One factor that is certainly weighing on the uranium market is the end of the US-Russian Megatons to Megawatts Program ("**MMP**"), an agreement under which 475 tonnes of highly enriched (weapons grade) uranium ("**HEU**") from approximately 19,000 Russian nuclear warheads is being down-blended with depleted uranium to produce commercial-grade, low-enriched uranium ("**LEU**") for use in nuclear power plants. Initiated in 1993, the 20-year program which terminates at the end of 2013, has provided LEU to the US for use in nuclear power plant since June, 1995. This LEU currently provides for over 10% of the electricity requirements of the US at a cost of approximately US $8 billion.

No plans have been enunciated by either party to extend the MMP agreement, however USEC Inc. has a multi-year contract with Russia's Techsnabexport ("**TENEX**") for a 10-year supply of LEU. Under the terms of the agreement, the supply of LEU to USEC will begin in 2013 and ramp up until it reaches a level in 2015 that is approximately 50% of the level currently supplied by TENEX to USEC under the MMP. With Russian approval, there is an option for the US to increase the quantities of LEU acquired up to the same level as the previous HEU program, however it is vital to note that, by all measures, this is a conventional purchase agreement unlike the MMP. The quantities supplied under the new contract will come from Russia's commercial enrichment activities and not from the down-blending of warhead material. WGM presumes that Russia will supply only that additional LEU in excess of the basic agreement that is available and surplus to its own internal needs. Yet Russia itself is not self-sufficient in uranium. In 2006, Tekhsnabexport and the Russian-Kazakh-Kyrgyz uranium mining venture "Zarechnoye" signed a contract to enable the delivery of $1 billion worth of uranium to Russia between January, 2007 and 2022. The Zarechnoye Mine was expected to begin uranium production in the third quarter of 2006 at a design capacity of 1,000 tons of U_3O_8 per year from a deposit estimated to contain a resource of 19,000 tons. In the project announcement, it is clear that Russia expects to receive as much as an additional 5,000 tonnes per year from Kazakhstan. The country increased its production by 26% in 2007 to 6,637 tonnes of yellow cake, up from 5,281 tonnes the previous year and has remained at the forefront in increasing output and becoming the world's leading producer accounting for 37% of global production with 19,451 t of uranium (22,939 t of U_3O_8) produced during 2011, an increase of 9% over 2010 (USGS Minerals 2011 Yearbook). Production is forecast at 30,000 tonnes in 2030 (MacLeod Dixon, 2008). Given the degree of State control of the industry in Kazakhstan, which may conflict with the interests of foreign investors, it will be interesting to see whether these ambitious production targets can be achieved

All followers of the market, including uranium commodity specialists, are forecasting higher prices at the end of 2013 and significantly higher prices by 2016 due to tightened supply-demand dynamics. This is mainly the result of new reactor builds which are out-pacing the number of old reactors being retired, and the termination of the US-Russia MMP. In Japan, where only two of the nation’s 50 reactors are on-line, the restarting of nuclear power stations is a key government goal to reduce the import bill for fossil fuel to run conventional stations. On 8 July, 2013, operators applied to restart 10 of the idle reactors and some units are projected to restart within the next year. Mr. Kenzo Oshima, a commissioner of Japan's Nuclear Regulation Authority, told Reuters "It is hard to imagine that all the applications would be rejected, though we don't know what the outcome will be at the moment." The restart of Japan’s reactors is seen internationally as a significant vote of confidence in nuclear energy. A collateral benefit to uranium markets is the fact that a restart means that Japan’s considerable nuclear fuel stockpiles will remain committed to Japan’s needs and will not be dumped on the market.

19.2 NEW URANIUM SUPPLY

According to the World Nuclear Organization, known uranium resources have increased almost threefold since 1975, more or less in line with expenditures on uranium exploration. Increased exploration expenditures in the future are likely to result in a corresponding increase in known resources, even as inflation increases the costs of mining and recovery thereby eliminating some resources.

Australia, with approximately 31% of the world’s uranium resources, remains a potential source of new supply. Until recently Australia upheld its Three Mine Policy initiated during the late 1970s, a law initiated by a national Labour government and vigorously pursued by the various State governments that limited the country’s uranium industry to a maximum of three operating uranium mines at any one time. The position of the Federal Government softened some time ago as it became clear that nuclear energy offered an increasingly greenhouse gas conscious world a source of relatively clean energy, notwithstanding environmental concerns related to the disposal of mine tailings. South Australia, with its giant Olympic Dam copper-uranium-gold deposit, was the first state government to move forward, permitting Australia’s fourth uranium mine, SXR Uranium One’s Honeymoon Mine, an in-situ leach operation (400 t U_3O_8/year for 7 yrs). BHP Billiton Ltd., the owner of Olympic Dam and a significant uranium producer on its own, announced plans to triple the mine’s output of uranium oxide to 15,000 tonnes per year, notwithstanding the impact of the accident in 2010 that seriously damaged the main production shaft. This plan was set aside in the wake of the Fukushima Daiichi disaster. Recent activity shows that the South Australia government has essentially abandoned the Three Mine Policy, a shift with uncertain impacts, but it seems clear to WGM that Australia's uranium production is likely to increase over the next decade as many deposits have been delineated and brought beyond the feasibility study stage – they are ready for development pending only completion of whatever new permitting process emerges. These deposits include Jabiluka in the Northern Territory, Yeelirrie (owned by Cameco) and Kintyre in Western Australian, and Valhalla in Queensland. Collectively, the known resources in Australia total some 2 Mt of U_3O_8 ranking it first among uranium resource countries. Australia has the potential to produce more than 20% of world mine production, and its federal government believes that the stated resource base substantially understates its uranium potential. As the country has no domestic nuclear energy industry, all of this uranium will be for export, and China has been identified as a major new market for Australian uranium.

WGM and others believe that renewed interest in nuclear power, partially due to energy market growth and partially due to concerns over the greenhouse gas emissions of alternative methods of generating base electrical load, will result in an escalation of uranium prices and consequently an increase in exploration and development spending on uranium projects. A range of undeveloped or moth-balled deposits are present in the US, the largest being the Mount Taylor deposit in the Ambrosia Lake district of New Mexico. Developed by Gulf Minerals Resources Co. in the late 1970s, the deposit has never seen full production. As with other metals, exploration expenditures will lag increased commodity prices, but will thereafter tend to track price trends quite closely. Since the mid-1980s, uranium exploration expenditures have been at a virtual standstill except in the Athabasca Basin where a collection of established producers and well funded junior companies have persisted due to the high grade character of the unconformity-type deposits (high reward/risk ratio). Improved exploration techniques and equipment will assist the industry in finding ever more deeply concealed deposits. Explorers are also active in countries which were not open to western companies during the 1970s uranium boom, countries such as Kazakhstan, Mongolia, China and Russia. As mentioned, Australia will be a major focus because of its known potential and the change in uranium politics.

Recent discoveries by uranium explorers will add new supply to the market which will likely continue over the next decade. This is occurring both in established camps, such as several recent discoveries in Canada's Athabasca Basin, and in emerging areas such as the Dornod Deposit being developed in Mongolia (4-5 M lbs U_3O_8 per year for 10 years). Kazakhstan remains a major producer with ample opportunity to increase its output. Intensive exploration activity in Canada is expected to result in new high grade discoveries and some are currently developing momentum, such as the Fission Uranium-Alpha Minerals Patterson Lake South deposit.

Although uranium mining in Gabon ceased in the late 1990s, and in general French uranium production has not been significant for most of the last 20 years, several new discoveries have been made in Gabon which are estimated to total approximately 26,000 tons of uranium. Areva, the main explorer in the country, maintained in 2009 that it required 30,000 tons to justify mine development and it is uncertain that this hurdle has been met. In neighbouring Niger, many companies are active including those from France, Russia, China, India and Korea. In northern Niger, Areva's Imouraren project is slated for production in 2014 or 2015 at an annual rate of 5,000 tpa of uranium. In southern Niger, China National Nuclear Corp. operates the Azelik uranium mine which has been plagued with labour stoppages. It is expected to ramp up production to 2,500 tpa in 2015 when it is expected to have a total resource of some one million tonnes of uranium, and to a level of 4-5,000 tpa by 2020.

In Africa, new mine development in Namibia appears to be a certainty at Rio Tinto's 160 Mt Z20 deposit and Swakop Uranium's 205 million tonne Husab Mine (497 ppm U). A planned production rate of 15 Mlbs/year of U_3O_8 is planned for the Husab Mine which is 90% owned by a consortium of companies from China. Rio Tinto recently suspended temporarily mining operations at its money-losing 2,700 tpa[13] Rössing Mine. In its lifetime, Rossing has produced more than 266 Mlbs of U_3O_8 as of year-end 2011 and the Z20 deposit is potentially of similar size and grade (350-400 ppm U). Other, albeit minor, new sources in Africa may be developed in such countries as in Botswana (A-Cap Resources's Letlbakane project), the Central African Republic (Areva's currently suspended Bakouma Mine), the DRC (Areva's Shinkolobwe Mine) and Mali (Rockgate Capital Corp's Faléa Project). The Letlbakane deposit (45 Mt averaging 180 ppm U) is low grade by all accounts and is forecast to produce 2.2 Mlbs U_3O_8 (846 t U) per year at an initial cost of US $48.41 per lb U_3O_8 according to a recent preliminary economic assessment.

Leading uranium projects in South America include U3O8 Corp's Kurupung exploration project in Guyana (>16 Mlbs U_3O_8 at 0.08-0.09% U_3O_8) and the Santa Quitéria phosphate deposit in Brazil (est. 1,500 tpa uranium production), as well as a uranium by-product project associated with copper production at Codelco's Radomiro Tomic Mine. U3O8 Corp also has uranium interests in Columbia and Argentina but these projects are not advanced. Considerable anti-uranium mining pressure exists that is holding back projects in Argentina

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[13] The nameplate capacity at Rössing is 4,500 tpa of uranium.

and a proposed mine in a UNESCO world heritage site in the Quebrada de Humabuaca area has been a lightning rods for protest. The potential for a few small historical producers to reopen if prices increase cannot be ruled out (eg. the Don Otto Mine).

Despite the clear potential for new mine development, many past producers are being examined due to looming supply shortages. About 20% of US uranium production up until the mid-1990s came as a by-product from central Florida's phosphate deposits, after which time the uranium production became uneconomic. Higher anticipated uranium prices are resulting in a re-examination of this resource for its potential to produce 400 t U/year as well as a similar lower-grade uranium-REE deposit in Morocco. Coal ash is another easily-accessible though minor uranium resource in many parts of the world, including in China where the uranium content of the ash averages approximately 210 ppm in Yunnan Province. The Xiaolongtang power station ash heap contains more than over 1000 t of uranium, with continuing annual contributions to the heap of about 190 t of uranium. Metallurgical testing is underway to determine the economics of acid leaching which can recover about 70% of the uranium.

It is forecast that widespread use of fast breeder reactors could increase the utilisation of uranium 50-fold or more. Breeder reactor are said to produce more fuel than they consume, but their construction in the US and elsewhere was cancelled because of the fears surrounding their use of plutonium. These reactors, supplied with natural or depleted uranium in a "fertile blanket" can be operated so that each tonne of ore yields 60 times more energy than in a conventional reactor. The security concerns surrounding the implementation of breeder programs and the handling of plutonium-enriched fuel make it unlikely these reactors will be widely used, however, it is interesting that Canada's CANDU reactor can in its present form consume plutonium-laced or mixed oxide fuel, commonly referred to as "**MOX**".

The world's power reactors, with a combined installed capacity of some 375 GWe, annually require about 68,000 tonnes of uranium. While this capacity is being run more productively, with higher capacity factors and reactor power levels, the total uranium fuel requirement is increasing. The growing number of reactors in service, and increases in overall efficiencies are being offset by a trend for larger capacities and higher fuel burn-up, and consequently fuel demand is more or less steady. According to the World Nuclear Organization, the electricity generated by nuclear power increased 3.6-fold from 1980 to 2008 while the uranium used increased by a factor of 2.5.

The amount of natural uranium (0.7% U_{235} 99.3% U_{238}) required for a given amount of reactor fuel (+/- 4% U_{235}) has been reduced somewhat through reducing the amount of the readily fissionable 0.7% U_{235} in the depleted uranium tails from enrichment plants. The reprocessing of used fuel from conventional light water reactors also utilises present resources more efficiently. In 2009, it was estimated that 75% of reactor fuel requirements were being met by direct mine output. The balance was being met from secondary sources such as commercial stockpiles, nuclear weapons stockpiles, recycled plutonium and uranium from reprocessing used fuel, and some from re-enrichment of depleted uranium tails.

Lastly, the impact of thorium as a reactor fuel should be factored into long-term uranium pricing scenarios. The thorium fuel cycle has some attractive features, and although it is not yet in commercial use as uranium is the only fuel currently used, research in India has advanced our knowledge of its potential. Thorium is approximately three times as abundant in the earth's crust as uranium. In 2009, the IAEA provided an estimate of global thorium resources at 3.6 million tonnes however this excludes data from much of the world where little or no exploration for thorium has occurred. The IAEA estimates that the actual thorium resource available may be as much as 6 Mt. At this time, thorium can be used as a fuel for CANDU reactors or in reactors specially designed for this purpose. The heavy water moderator inherent in the CANDU design make it very efficient in respect to neutron-capture provided the reactor is started using a fissile material such as U_{235} or Pu_{239}. The Th_{232} atom isotope captures a neutron in the reactor, thus becoming fissile U_{233}, which continues the chain reaction. In the future, some advanced reactor designs are likely to be able to make use of thorium on a substantial scale, however at this time the prevalent light water reactors are unable to use thorium due to their lower neutron efficiency. The current reactor population and current production technologies create considerable inertia working against the radical shift that thorium reactors would bring, and thus the substitution of thorium for uranium is not seen as having a measurable impact on uranium markets in the foreseeable future. The higher uranium prices needed to bring about a changeover to thorium would presumably result in a flood of new uranium supply at lower prices which would reduce the economic incentive for change. Other factors favouring thorium aside, such as its inability to produce plutonium as a by-product and enrichment issues attached to the use of uranium fuel in light water reactors, WGM does not foresee thorium use as a threat to development of such uranium projects as Appia's Teasdale deposit.

19.3 URANIUM PRICE OUTLOOK

Any analysis of the uranium market must begin with an analysis of price trends. However, while the pricing of other mineral commodities is transparent due to the existence of metal retailers and a terminal market/warehouse for which prices quotes are readily available, the pricing of uranium has commonly been determined in closed door negotiations between producers and energy utilities. WGM's review of information pertaining to the main Elliot Lake producers, Denison Mines Limited and Rio Algom Limited, revealed that the companies were willing to divulge delivery prices during the early 1960s, however this willingness apparently disappeared as prices escalated such that the companies only advertised sales volumes[14]. For confidentiality reasons, Cameco does not publish its contracted delivery prices in its annual reports. As a result, it is very difficult to get an unambiguous average price for the majority of uranium deliveries, whether as U_3O_8 or as reactor fuel (UO_2).

At about the time Appia's Elliot Lake project was initiated, the Spot Market price on 6 June, 2007 was US$135 per lb of U_3O_8 versus approximately $40 at the time of writing this report in June, 2013, a price retreat of approximately 70%. The less volatile Term Market price at that time was $95 versus $57 at this date, a retreat of 40%. However, it important to note that the Spot Price is the discount price and not the price at which most uranium is delivered to energy utilities. As of the date of this report, the Spot and Term prices are still recovering from the effects of the Fukushima Daiichi tsunami disaster in 2011 which was used by the media, anti-nuclear activists, and opposition governments to cast unjustifiable dispersions on the nuclear energy industry. Even Japan, having understandable concerns about reactor safety in an area with clear geohazards concerns, has re-stated its support of the industry and applications are being studied for the restart of 10 reactors idled since early 2011.

Utilities are intermittently buying in the Spot Market to supplement nuclear fuel supplies stockpiled at reactors and to satisfy the need for initial cores at new reactor sites. Substantial new electrical capacity development is going ahead at various nuclear sites and continuing upwards pressure on prices seems certain. The US-Russian HEU Agreement, officially named the "Megatons to Megawatts Program", terminates at the end of 2013. Under this agreement, 500 tonnes of highly enriched (weapons grade) uranium (**"HEU"**) from approximately 20,000 Russian nuclear warheads is being down-blended to produce

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[14] The lack of transparency in the uranium market was once more given force by a recent decision concerning uranium by the Canadian Government, namely: "commencing in 2002, Natural Resources Canada has decided to suspend the publication of the Average Price of Deliveries under Export Contracts for uranium for a period of three to five years, pending a policy review and assessment of market conditions" (NRCan website).

commercial-grade, low-enriched uranium ("**LEU**") for use in nuclear power plants. The LEU is shipped to the US where it is used as nuclear power plant fuel to generate electricity. Initiated in 1993, the 20-year program first shipped LEU to the US in June 1995. At this time the warhead-derived LEU from Russia provides over 10% of the electricity requirements of the US. No plans have been enunciated by either party to extend the agreement.

Despite the anti-nuclear political rhetoric present in Europe, and until its recent reversal, Japan (which accurately WGM predicted), many new nuclear-electric development projects are proceeding more or less on schedule. Both China and India, neither self-sufficient in fossil fuels, are constructing new reactors as quickly as possible due to rapidly escalating energy requirements – India's will triple within 25 years.

The last 20 years in uranium market has also witnessed a change-over of Russia from a major exporter of surplus uranium from its U_3O_8 and HEU[15] stockpiles to a net importer and is seeing the growing commitments of China and India to nuclear energy as competing with its own interests. Developing nations are increasingly turning to nuclear as a reliable provider of "green" energy. Even the energy-rich United Arab Emirates is building its first of several planned reactors as a means of providing energy for water desalination plants while freeing future oil and gas production for export rather than domestic consumption. Saudi Arabia is planning the construction of a group of reactors for the same purposes. The future market for small modular reactors for remote communities and industrial complexes seems assured to contribute to uranium demand. These factors must be taken into consideration in order to have some understanding as to the potential for a project such as is envisioned by Appia.

The uranium price outlook is a function of supply and demand like any other mineral commodity. Unlike other commodities, the demand for uranium is governed by reactor requirements which are known both now and for the future lifetime of the nuclear plant. A typical 1,000 MW light water reactor requires approximately 800,000 lbs of U_3O_8 annually, depending upon its load factor and burn-up rate. A typical 1,000 MW plant can supply the electricity needs for a city of about 600,000-1 million population.

During 2010, a typical 1,000 MW nuclear reactor came with a price tag of approximately US $5.3 billion ($5,300 per kW according to the World Nuclear Organization), and required the better part of a decade to permit and construct. For a light water reactor requiring enriched

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[15] HEU – highly enriched (weapons grade) uranium containing as much as 90% U_{235} in contrast with natural uranium which contains approximately 0.7% U_{235} and 99.3% U_{238}. Note that light water nuclear reactors require 3-5% U_{235} in contrast with CANDU reactors which accept uranium fuel in its natural isotopic ratios.

uranium (LEU), the costs associated with acquiring one kilogram of uranium oxide (UO_2) fuel in June 2013 was:

U3O8 purchase:	8.9 kg @ $130/kg	$ 1,160.
Conversion to U:	7.5 kg @ $11	83.
U_{235} Enrichment:	7.3 SWU @ $120	880.
Fuel Fabrication:	per kilogram	240.
	Total *	$2,360.

*source World Nuclear Organization (www.world-nuclear.org)

It is clear from the foregoing analysis that the initial purchase price of U_3O_8 is only about half of the cost of the manufactured uranium fuel except in the case of heavy water reactors which do not require enrichment.

Unlike some forms of power generation, power reactors can't usefully be throttled back to consume less uranium - they are operated at 100% output, and therefore energy utilities are mainly concerned about nuclear fuel supply rather than price risk. Various estimates exist as to the impact of fuel costs on the economic viability of nuclear reactor, however there is a general consensus that costs would need to exceed $200 to $500 per pound U_3O_8 before sufficient negatively effects would be experienced to affect decisions regarding reactor construction. It is estimated that a doubling of the U_3O_8 price from $70/lb to $140/lb would increase fuel costs from 0.75 cents per KWh to 1.05 cents per KWh. In the current pricing environment this appears to allow sufficient room for uranium prices to increase significantly from present levels.

Figure 25 shows the history of average annual Spot and Term Market prices for U_3O_8 beginning in 1968. The gap between term contract and spot prices reflects persistent oversupply during the 1970s and 1980s. The difference between spot and contract prices was at its smallest during 1980. It decreased temporarily in 1997 and again in 2004. The historic and unusual decoupling of market price and metal inventories during the late 1970s as a result of government stockpiling programs was very different from more recent developments in uranium markets which, in 2006, saw virtually no surplus inventory available. During mid-2007, spot prices were approximately $30 higher per pound of U_3O_8 than the term contract price, an inversion that WGM attributed to the absence of substantial quantities of new uranium available for long term supply, and thus a simple shortage of long term contracts to set a new benchmark price.

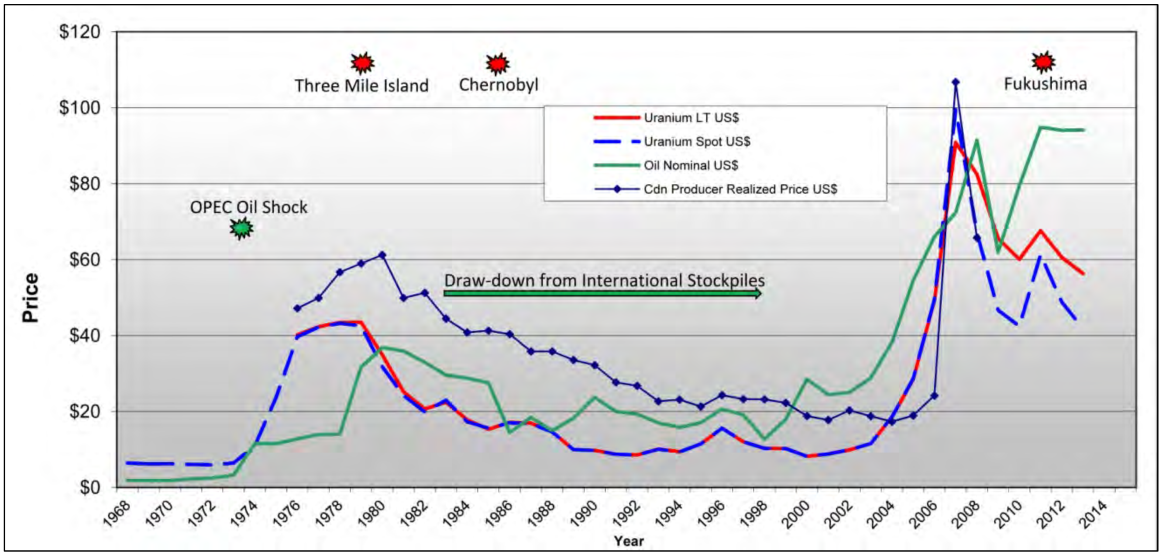

Figure 25 Graph of annual average Spot and Term U_3O_8 prices against actual price received by Canadian producers and the annual average oil price. Data for 2013 is current averaged to mid-May. Major incidents in the energy sector are shown symbolically. Although NRCan no longer publishes realized price data, evidence suggests producers have been substantially protected from the post-2008 volatility.

The speculative participation of hedge funds in buying up the available uranium was widely seen as a catalyst to the 2006-07 price run-up. Interestingly, as uranium prices recovered during the period November, 2010 through January, 2011, the spot and term prices again inverted with the discount price up to $5.00 more per pound than the $65 term price. In this case, however, the term price rose to match and briefly exceed the discount price before the fall-off experienced in mid-March as a result of the tsunami in Japan.

The following graph, Figure 26, shows the foregoing chart adjusted for inflation using the Canadian consumer price index and using 2006 dollars. The peak average annual realized price by uranium producers achieved during 1980 was equal to approximately C$133 (US$116) per pound of U_3O_8 in 2006 dollars, and was realized on the basis of contracts negotiated in previous years. The sales prices achieved for 1980 came two years after the peak spot price. The earlier negotiations occurred at a time when energy utility companies were deeply concerned about continuing price escalations. It is, however, noteworthy that spot market activity at the time of delivery accounted for only a very small fraction of total market sales, thus not truly reflecting the amount of uranium actually available in the market from accumulating stockpiles. It is also interesting that spot market activity did not significantly increase until into the third year of declining realized prices.

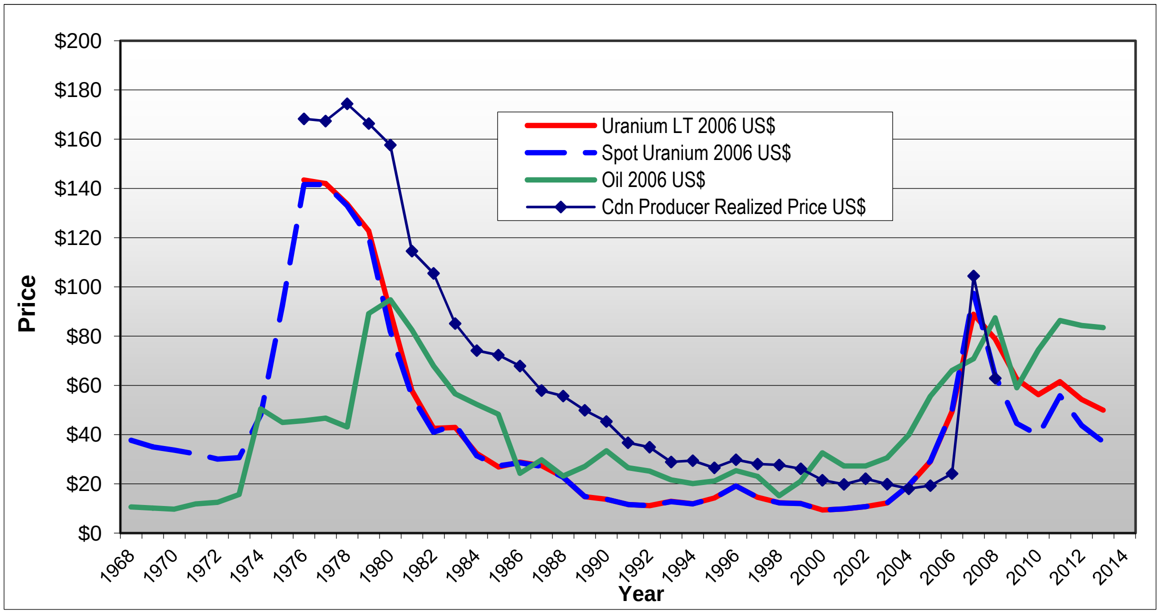

Figure 26: Average annual uranium and oil prices in constant 2006 US Dollars. Note that the decline in realized prices to producers lagged the fall of market prices by approximately 3 years. The peak price of $120 per pound of U_3O_8 experienced during 2007 remained below the inflation-adjusted highs realized during 1976 through 1979. Also note the large disconnect currently between oil prices and uranium prices.

Figure 26 demonstrates how having long term supply contracts generally resulted in domestic producers being well protected, at least initially, from the slide in uranium prices that occurred during the early 1980s. The data also show that, despite the price drop following the tsunami-triggered Fukushima nuclear disaster in Japan, the average uranium price recovered from the initial shock well into 2011 before more recently falling off.

While the underlying data is the same as that used for the foregoing chart, the profound effects of the inflationary 1970s and early 1980s can be easily seen. The graph also shows that the price escalation in 2007 did not come close to the inflation-adjusted uranium price experienced in the 1970s. This suggests that the current uranium price seen today is extremely inexpensive in historical terms and costly to producers in terms of lost revenue to offset production costs. The graph supports WGM's view that the current pricing environment is unsustainable and that prices will sharply rise.

Perceptions of the direction and extent of price movements differed in the early 1980s, and again in recent years, when viewed in the different currencies. The current price of uranium must be adjusted somewhat because of the fluctuating value of the US dollar against some currencies. The cost of reactor fuel should therefore be reviewed against a basket of currencies including the Canadian dollar, major European currencies, the Chinese Renminbi, the South Korean Won and the Japanese Yen. While WGM has not carried out this type of review, it nevertheless seems reasonable that in any real terms, currency price fluctuations have little impact on the uranium fuel operating costs of power reactors.

At the same time as uranium demand is increasing, uranium mining projects are being slowed due to perceived social, political and capital risks. The permitting process for new uranium mining projects in greenfield areas is notorious for its delays, so sharp increases in production are unlikely. If the dynamics of the last uranium boom can be used as a model, lagging production will cause uranium prices to stabilize at a much higher price than has been seen recently. Evidence from various sources was reported by the World Nuclear Association to show that $40 was a marginal producer price in 2010, illustrating that many producers were under stress at that time.

Despite the knowledge that significant undeveloped uranium resources exist, serious shortages of U_3O_8 and reactor fuel have been forecast by commodity analysts. This view is based on a multiplicity of reasons such as:

- the long time-line between deposit delineation and mine permitting in most countries, driven in part by matters of corporate social responsibility, environmental impact mitigation and public resistance to uranium mining;
- the politicization of decisions relating to the due course approval of uranium mining projects and pandering to often ill-informed public opinion;
- legitimate environmental concerns in some areas centring on surface and groundwater contamination;
- unreasonable restrictions on uranium mine development and outright bans on uranium mining in some jurisdictions (e.g. affecting the Coles Hill deposit[16], Virginia, USA);
- the relatively low grade of major uranium deposits in some areas or the technical challenges of underground mining (e.g. Cigar Lake, Saskatchewan);
- infrastructure challenges in many uranium-rich provinces such as SW Africa, Mongolia and Kazakhstan;
- resource sovereignty demands requiring production royalties that render the deposits uneconomic at anticipated prices;

According to the U.S. Energy Information Administration's ("**EIA**") May, 2012 annual report, the owners and operators of U.S. civilian nuclear power reactors purchased the

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[16] According to a Virginia Energy Resources (TSX-V VUI) corporate presentation, the Coles Hill deposit in Virginia, USA is estimated to contain a NI 43-101 compliant resource of 133 Mlbs of U_3O_8 at an average grade of 0.056% U_3O_8 (1.14 lbs U_3O_8/ton) using a 0.025% U_3O_8 cut-off.

equivalent[17] of 58 Mlbs of U_3O_8 from U.S. and foreign suppliers during 2012, an increase of 3Mlbs (5%) over the previous year, at a weighted-average price of \$54.99 per pound U_3O_8. The 2012 weighted-average price represented an increase of 442% compared with the 2001 weighted-average price of \$10.15 per pound U_3O_8. Only 17% of the 2012 delivered uranium was of U.S. origin and this was purchased at a higher price of \$59.44 per pound. The uranium sourced from foreign sources accounted for 83% of deliveries at a weighted-average price of \$54.07 per pound of U_3O_8 with Australian and Canadian uranium together totalling 35% percent of the delivered amount. Uranium originating in Kazakhstan, Russia and Uzbekistan accounted for 29% and the remaining 19% originated from Brazil, China, Malawi, Namibia, Niger, South Africa and the Ukraine. During 2012, 14% of the U_3O_8 was purchased under spot contracts at a weighted-average price of \$51.04 per pound. According to the EIA, the remaining 86% was purchased under long-term contracts at a weighted-average price of \$55.65 per pound of U_3O_8.

While the current low prices have seen U.S. utilities increase their stockpiles from an average 111.2 Mlbs U_3O_8 equivalent during 2008-2011 to 120.7 Mlbs in 2012, and increase of 8.6% according to EIA data, the inventory represents approximately 2.5 years consumption[18] which is significantly less than the time required to discover and bring new deposits into production. According to the EIA, U.S. utilities have signed contracts for delivery during 2013 of a minimum of 45.9 Mlbs of U_3O_8 and a maximum of 48.8 Mlbs of U_3O_8 which will leave the utilities with unfilled requirements totalling 8.8 Mlbs during 2013 for which purchases will be made on the Spot Market if inventories are not depleted. The unfilled requirements currently total 17.6 Mlbs for 2014 and are approximately 125 Mlbs for 2018.

Given the position of the US as the world's largest consumer of U_3O_8 and the benchmark set by its most recent U_3O_8 purchases, it is exceedingly difficult to believe that future uranium prices will pull back from their current levels. Indeed, the evidence proves that the current Spot Market price has little to do with the average prices currently being realized by uranium producers. During 2012, a period during with the Spot price averaged \$48.81 and the Term Price averaged \$60.58, U.S. utilities signed 34 new purchase contracts with deliveries in 2012

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[17] The equivalent uranium refers to deliveries of UF6, enriched uranium and other forms that are expressed in equivalent U_3O_8 form.

[18] According to the EIA, U.S. civilian utilities will consume approximately 50 Mlbs of U_3O_8 equivalent during 2013, far in excess of its domestic production of 4.15 Mlbs in 2012. Market requirements are expected to be approximately 47.8 Mlbs in 2014, rising to 55.6 Mlbs in 2015 and averaging 50.6 Mlbs of U_3O_8 equivalent per year during 2016-2022. Total U.S. production capacity is 27.6 Mlbs if all mines and in-situ leach capacity is included for current producers, projects on stand-by, deposits that are in development and new operations that are licensed but not yet operating.

of 12 Mlbs of equivalent U_3O_8 at a weighted-average price of $55.16 per pound, 31 being Spot Market contracts and three being Term Market contracts. The growing contract deficit experienced by U.S. utilities adds pressure to supply sources that will see significant pressure as the U.S.-Russia HEU agreement ends. In this context, and with the looming threat of uranium shortages, Appia should experience a significantly higher price regime than exists in mid-2013 if the Spot Market price is falsely used as the sole commodity price indicator.

Most uranium market watchers expect increased prices to bring new production into the market. Respondents to a recent survey by the U.S. EIA identified reserves of only 51.8 Mlbs of U_3O_8 at prices of less than $30 per pound, however the reserves increased to 304 Mlbs at prices as high as $100 per pound of U_3O_8. Notwithstanding hypothetical availabilities, RBC Capital Markets currently forecasts Term Market U_3O_8 prices rising to $75/lb in 2016, and an $80 price persisting beyond that to at least 2020. CIBC Metals & Minerals have forecast Term Market prices for U_3O_8 during 2014 and in the longer term of $65 and $70, respectively. Market analysts Raymond James has forecast U_3O_8 prices to average above $60/lb in 2013, a hurdle that may currently seem unlikely, and prices in excess of $70/lb in 2014 and 2015 before settling to $70/lb in the long-term. In contrast, BMO Research has slowed the expected rate of restarts in Japan and increased secondary supplies due to excess enrichment capacity allowing the enrichment companies to underfeed their plants. Reflecting the reduced demand projections for primary mined uranium, BMO Research has further delayed several start-up and expansion projects in Australia, Kazakhstan, Niger, Namibia and Tanzania. The net impact is a slight increase in forecast near-term oversupply that results in BMO Research now expecting the turnaround in the uranium price to take longer than previously anticipated. It is now forecasting the uranium price to average US$52/lb U3O8 in 2014, down from US$70/lb previously and US$60/lb in 2015, down from US$70/lb previously. BMO maintains a forecast of the uranium market entering a sustained deficit from 2018 onwards. Despite variances in the short term, there is a general consensus that the longer the uranium price recovery is forestalled, the steeper the recovery will be.

19.4 RARE EARTH METAL PRICE OUTLOOK

The search for REE deposits was energized by ever-increasing demand for these metals in a wide range of ‘high-tech’ applications. China’s decision to restrict exports of REEs to meet its own domestic needs initially caused price spikes, but at the end of 2011 and during 2012, prices fell. Domestic consumption of rare earths in the United States (and imports) declined significantly in 2011 compared with that of 2010. Owing to declining supply from China, prices for most rare-earth products increased significantly in the third and fourth quarters of 2011. Consumption generally decreased for cerium compounds used in automotive catalytic converters and in glass additives and glass-polishing compounds; rare-earth chlorides used in the production of fluid-cracking catalysts for oil refining; rare-earth compounds used in automotive catalytic converters and many other applications. Rare-earth metals and their alloys used in armaments and base-metal alloys also saw price declines. Consumption was more stable in lighter flints, permanent magnets, pyrophoric alloys, and superalloys, but decreased for yttrium compounds. Demand remained stable for rare earths in many other applications, especially automotive catalytic converters, permanent magnets, and rechargeable batteries for electric and hybrid vehicles.

Molycorp's rare-earth separation plant at Mountain Pass, California resumed operation in 2007 and continued to operate throughout 2011 producing bastnäsite concentrates and other REE intermediates and refined products from mine stockpiles. Molycorp commenced with its Project Phoenix in 2011, a plan to reopen mining operations and to build new processing facilities at the mine, and commenced new mining operations during 2012, returning the US to REE producer status (Figure 27). Molycorp also acquired Canadian REE technology company, Neo Material Technologies, a leading REE fabricator with global operations including facilities in China. The Molycorp processing plant at Mountain Pass was in operation in the first quarter of 2013. US consumption however decreased 25% in 2012 over 2011, partly due to substitution. In it's 2013 1st quarter report, MolyCorp reported that it sold 3,274 tonnes of REE product from its various global operations at an average sales price of US $44.71 per kilogram. The product from the Mountain Pass Mine, of which approximately 47% is Ce, 32% is La, 21% is Nd and Pr and less than 5% is other REOs, was sold at US $23/kg. Although revenues were up 9% over 2012 levels, the company posted a net loss of $0.33 per share due to soft REE prices. Molycorp plans to ramp up to the facility's initial planned annual run rate of 19,050 mt of REO equivalent by mid-year 2013 yet it is not forecast to develop profitability until 2015.

The considerable excitement that gripped rare earth metal explorers, a relatively recent phenomenon, forced governments to suddenly assess the strategic importance of REEs to key industrial applications. The Australian Government's review of national mineral activities for 2009 does <u>not</u> mention a single rare earth project even though several major discoveries were moving towards production. The Mount Weld Mine, owned by Lynas Corp., is now in production and the company's REO refinery in Malaysia is ramping up to an 11,000 tpa rate of REO production, to be doubled when its phase 2 program is commissioned later in 2013.

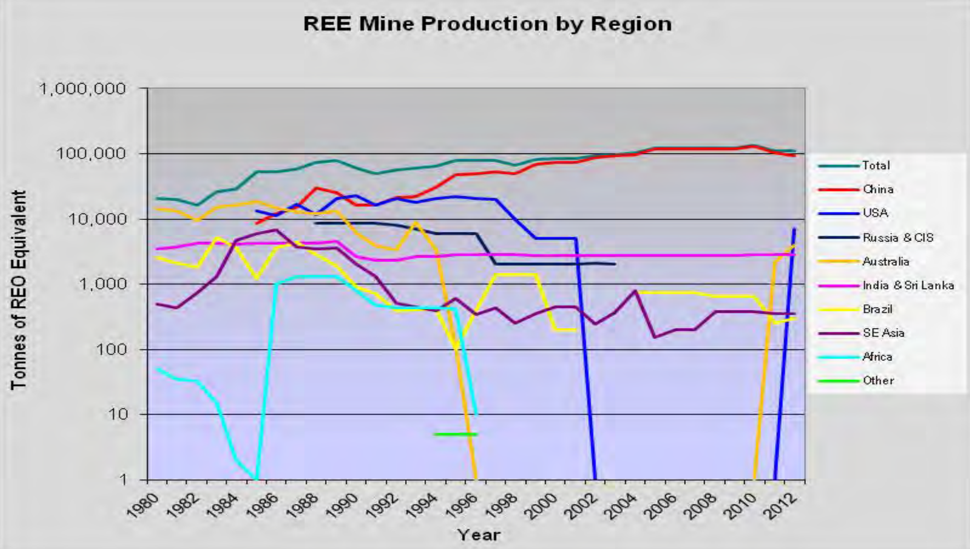

Figure 27: Global REE production during the period 1983 through 2012.

Global exploration efforts to develop rare earths projects surged in 2011, and investment and interest increased dramatically. Economic assessments continued in North America at Bear Lodge in Wyoming; Diamond Creek in Idaho; Elk Creek in Nebraska; Hoidas Lake in Saskatchewan, Canada; Kipawa in Quebec, Canada; Lemhi Pass in Idaho-Montana; and Nechalacho (Thor Lake) in Canada's Nunavut Territory. None of these deposits are thought to be on the verge of a production decision. Economic assessments in other locations around the world include Dubbo Zirconia in New South Wales, Australia; Kangankunde in Malawi; Nolan's Bore Project in the Northern Territory, Australia, and at Steenkampskraal in South Africa. Nolan's Bore is very similar to Mount Weld in respect to the balance of REEs, however, it is unlikely to enter production for some time yet.

Most of the production growth, to the 133,000 tonnes produced in 2011 (110,000 t at this time), has resulted from increased output from China. With the USA and Australia returning to REE production during 2012, any forecasts of future REE prices must incorporate the impact of increasing demand and alternative supplies in the global marketplace

The major uses of the rare metals are summarized in Table 28. In addition to the uses listed, REEs are also used in the defence industry in many applications including precision-guided munitions (smart bombs), rangefinder lasers and target designators, detection devices for underwater mines, communications, aircraft control mechanisms, high-temperature ceramics in jet engines, information displays, radar systems, coatings, optical equipment, sonar applications and in electronic counter measure technologies.

Table 28

Major Industrial Applications of Rare Earth Metals and Compounds

Element	Symbol	Market Price *	Applications
Lanthanum	$_{57}\mathrm{La}^{139}$	$16.50/kg $14.00/kg	Catalyst used in the cracking of hydrocarbons to produce fuel, fuel calls and batteries, in optical glass to modify the refractive index, NiMH batteries for computers, in phosphors for X-Ray films. Used to reduce radiation dosages in MRI, CAT and sonogram imaging techniques.
Cerium	$_{58}\mathrm{Ce}^{140}$	$16.00/kg $7.25/kg	Catalytic converters, additive for diesel fuels. Polishing compound for high performance glasses (television screens, mirrors, optical glass, disk drives and silicon microprocessors). Decolouring agent for glass and photographic filters. In high-strength, low alloy steels, used to improve performance in chrome plating baths. Used with Tb in phosphors in tri-colour lamps and compact fluorescent lighting. Used with Zr in high-performance insulating ceramics (Space Shuttle).
Praseodymium	$_{59}\mathrm{Pr}^{141}$	$80.00/kg $78.00/kg	Colouring pigment in ceramic tile/glass. High-quality mirrors. Used with Nd in photographic filters to reduce certain wavelengths of light. Pollution-control catalysts. Used to make electric motors lighter.
Neodymium	$_{60}\mathrm{Nd}^{144}$	$73.00/kg $62.50/kg	Nd-Fe-B magnets for mobile phones, portable CD players and computers. Nd capacitors in mobile phones. Nd-lasers for surgery and in manufacturing sector. Strong magnets for MRI units. Anti-glare automobile glass and mirrors, CRT glass. Sky-blue colouring pigments in ceramics and glass.
Promethium	$_{61}\mathrm{Pm}^{145}$	n.a.	Very scarce – no stable isotopes – longest half-life (Pr^{145}) is 17.1 years.
Samarium	$_{62}\mathrm{Sm}^{150}$	$33.00/kg $9.00/kg	Filter glasses for Nd-lasers. Used to stabilize the high-temperature performance of REE magnets (Sm-Co magnets are the strongest available). Used with titanates as dielectric compounds in capacitors operating at microwave frequencies. Glass and tile pigmentation.
Europium	$_{63}\mathrm{Eu}^{152}$	$1,350/kg $825/kg	A photon emitter used as the red phosphor in television and computer screens. Used in fluorescent lights to reduce electrical consumption. Used as a luminescent tag in living tissue medical research.
Gadolinium	$_{64}\mathrm{Gd}^{157}$	$132.50/kg $46.50/kg	Magnetic properties make it useful in magneto-optic recording technology – e.g. bubble-memory in super-computers. Enhances imaging in MRI devices. Used in the detection of radiation leaks in nuclear power-plants.
Terbium	$_{65}\mathrm{Tb}^{159}$	$1000.00/kg $700/kg	Improves energy efficiency in fluorescent lamps. Used in magnetic films used for recording data in magneto-optical applications.
Dysprosium	$_{66}\mathrm{Dy}^{163}$	$605/kg $475/kg	Allows electronic devices to be smaller and faster. Added to ceramics to produce high-capacitance miniaturized capacitors. Added to NdFeB high-strength permanent magnets to improve coercivity.
Holmium	$_{67}\mathrm{Ho}^{165}$	uncertain	Very scarce and has few practical uses
Erbium	$_{68}\mathrm{Er}^{167}$	uncertain	Used in amplifiers for optic data transmission. Medical and dental lasers. Only stable pink pigmentation for glass (sunglasses and decorative glass).
Thulium	$_{69}\mathrm{Tm}^{169}$	uncertain	Rarest of the REEs – similar chemistry to yttrium – can be used in sensitive X-Ray phosphors to reduce the required radiation exposures.
Ytterbium	$_{70}\mathrm{Yb}^{173}$	uncertain	Similar chemistry to Y – when under high stress, increases its electrical resistance by 10x – and therefore used in stress gauges to monitor seismic ground movements.
Lutetium	$_{71}\mathrm{Lu}^{175}$	n.a.	One of the least abundant REEs – Ce-doped lutetium oxyorthosilicate (LSO) is used in detectors in positron emission tomography (PET) applications.
Yttrium	$_{39}\mathrm{Y}^{89}$	$48.00/kg $18.00/kg	Used in oxygen sensors for engines to improve the combustion of fuels. Y-Fe garnets used as resonators in frequency meters, magnetic field measuring devices, tuneable transistors and Gunn oscillators, laser crystals. Stabilizer and mould-former for light-weight engine turbine. Stabilizer in rocket nose cones. In ceramics used for melting radioactive metals. Used as nozzles for jet casting molten alloys. Used as a primer for other metallic coatings (e.g. titanium coatings).

* Note: The price for the metal is shown above the price for the 99% pure oxide form (source www.Metal-Pages.com, 18 July, 2013)

The recent FOB market price (metal and oxide form) as of July, 2013 is indicated in US Dollars (*source* www.metal-pages.com). These prices reflect significant declines from the price peack experienced in early 2011. The drops ranged from 75% to 85% in the case of Ce and La, respectively, to 58% in the case of Nd, 43% for Pr, 54% for Gd and 90% for Sm, the latter dropping from $91 per kilogram for samarium oxide to $9.00 per kilogram. The decline for some heavy rare earth metals was less severe with Tb down 10%, but Y experienced a significant 83% drop in value. Dysprosium and Eu experienced price increases of 2% and 25%, respectively.

During the early part of the decade, mineral economists and metals market forecasters predicted growth in REE demand that in reality has fallen short of expectations. WGM believes this is largely due to the impact of the global financial crisis that initially affected the markets during late 2008 (and continues today). As a result, metal demand declined in the west while Chinese growth continued more or less unaffected due to its population and growing economy. India also contributed to increased demand. The market has certainly grown, but clearly not as expected a few short years ago.

On the supply side, the growth in demand was not balanced by increased supply. Having been driven out of production by China's aggressive price cutting in the 1990s, evidence suggests that potential producers required time to explore new discoveries, establish resources, design new mining operations and secure the necessary operating permits to allow the mine to be constructed. More recently, economic uncertainty has somewhat impeded the ability of companies to raise capital for projects. As a result of the foregoing impacts, REE demand has slowly out-stripped supply and created an imbalance. China, with approximately 95% of global REE production as a result of its aggressive actions against competitors, is now faced with the possibility that it may not be able to satisfy its own fabrication demands. Even less is its ability to meet the global supply shortage that it created. During 2010, China reacted by reducing rare metals exports to Japan, a major manufacturer of products containing REEs, and REE prices reacted accordingly. China remains confronted with the problem of balancing competing interests for the foreseeable future.

The need for increased REE production has not gone un-noticed by the international mining community. Typically, the junior mining sector was quick to respond to forecasts made a decade ago regarding the current situation. Old projects and more recent discoveries have been revived, and fresh venture financing has been found to support renewed exploration projects. Unfortunately, the financial crisis has negatively impacted several major REE projects that sought financing during the crisis. Within the industry, it was recognized that the main beneficiaries of the supply imbalance would be the one or two companies that were the first into production. That race was won by Molycorp, owner of the Mountain Pass Mine in California and Lynas Corp. owner of the Mount Weld Mine in Western Australia. At this time, investment has been slowed and Arafura Resources' Nolan's Bore REE mine developments are behind the schedule originally envisioned by its owners although bulk sampling, test concentrate production and pre-leach testing has been completed.

Given the current price decline in REEs, it is doubtful that significant new production will commence anytime soon unless the deposit is significant enriched in heavy REEs or the REEs can be produced as a co-product with another metal that contributes significantly to the revenue stream. The junior mining sector is poised to bring a handful of REE projects into production during the next decade, however a great amount of hyperbole has surrounded many of these projects to the affect that the economic realities are often obscured. Clearly, some due diligence is needed by any investor or company interested in this mineral sector.

China's minerals infrastructure that supports the production of rare earth metals is thought to be the world's strongest. Previously, China's position was in the top three, with the other two comprising the United States and Japan. However, in the last decade, China's output has soared, with the major effect of lowering prices and driving its competitors out of the market. In 2007, China was responsible for 96.8% of global rare earth metal production, most of which is from mines located in Inner Mongolia. The Inner Mongolian Baotou Steel Union Co., Ltd. is the largest rare-earth metal manufacturer in China. Even though about 42% of global REE resources and reserves are situated outside of China, its cheap labour and Government subsidies ensured that Chinese companies were well supported in respect to investing in new mines and processing plants during the 1980s. This infrastructure included rare earth metals research and development laboratories that worked to undercut China's rivals. In the early part of the 1990s, China could produce neodymium very inexpensively for the market, resulting in a price drop from $11.70 per kilogram in 1992 to $7.40 in 1996. In a relatively short time, the REE market volume increased from 40,000 tonnes per annum (tpa) to 125,000 tpa. For nearly 20 years, China has pursued a policy to make it the "*OPEC of rare earth metals*". This goal is probably unachievable given the participation of the US and Australia as significant producers supported by REE off-take agreements.

According to the Peterson Institute for International Economics (www.piie.com), China's rare metal industry could be characterized by what industry observers call "disorderly competition" and "price chaos". Local firms have engaged in a price war leveraged on expanded production. In 2008, China's annual smelting capacity for REE metal production exceeded 200,000 (short) tons, which at the time was more than double global demand. In August, 2009, the Ministry of Industry and Information Technology issued a draft policy recommending an annual export quota of 35,000 tons, improvement in mining and environmental practices and a potential ban on exporting five REEs seen to be in short supply and essential to China. The goal seems to have been to consolidate the domestic industry and stabilize prices while trying to attract investment in downstream applications and fabrication.

This "disorderly" competition from Chinese producers was the principle reason for the closing of the Mountain Pass Mine in California at which time overproduction killed the market and drove out higher cost producers. A very different market exists at this time, especially since 2007, with China reducing its REE exports and potentially restricting the export of some metals entirely. China has apparently pursued this policy for two reasons; firstly to assure itself of a supply of metals vital to its defence industries and manufacturing sectors, and secondly to pressure western manufacturers to establish production facilities in China. The 22 September 2010 embargo of REE exports to Japan in retaliation for Japan seizing a Chinese trawler has caused ripple effects through the industry since Japan was completely reliant on Chinese sources for metals used in the production of REE magnets. Japan's position as a major supplier of magnets to the West has provoked the US Government to consider a bill to subsidize the revival of its domestic REE industry. Due to the increased demand for REE metals, Molycorp Inc. has invested approximately $500M in the reopening of the Mountain Pass Mine. Over the past decade, the United States has imported about 87% of its lanthanide metals from China. This clearly will change with Molycorp's "mine-to-magnets" strategy, including the expansion and modernization of its Mountain Pass processing facility and its acquisition of Neo Material Technologies. Molycorp expects to be one of the world's most integrated producers of rare earth products, including oxides, metals, alloys and magnets.

Several potential producers are advancing projects towards mining. One is the Nechalacho Project at Thor Lake in the Northwest Territories 100% owned by Avalon Rare Metals Inc. ("**Avalon**"), a Canadian junior company. This deposit, known for more than 20 years, is emerging as a major undeveloped REE resource, however the project is challenged by its remote location and a lack of infrastructure. Avalon has advanced the project with the view that it is enriched in HREEs, however in order of declining abundance the major metals are Ce, Nd, La, Y, Pr and Sa….. Yttrium is the only HREE metal that is present in concentrations above 0.1%. Nevertheless, the deposit is sizeable at 197 Mt averaging 1.24% $LREE_{TOTAL}$ and 0.22% $HREE_{TOTAL}$. Avalon has been well funded and has no debt. Its plan, assessed through a recent scoping study by SNC-Lavalin, is to construct a separation plant with an intended production capacity of 25,000 tonnes per annum. This plant capacity is intended to handle the presently contemplated production of 10,000 t/a from Nechalacho, any future Avalon production increases, and process material from other potential future producers, especially those producing chemical precipitates rich in the heavy rare earths.

An effort similar to that of Molycorp sees Rareco moving the past-producing Steenkampskraal Mine back towards production in South Africa with a target date a few years in the future. At the same time Arafura Resources is working towards 10,000 t REE/year production from Nolan's Bore deposit.

Japan, a major fabricator of REE-bearing goods, imports more than 10,000 tons of rare earth metals per year, while about a fifth of the country's total annual consumption is believed to enter the country through a thriving black import network, without which Japan would already be in a severe supply crisis. China has been lowering its export quotas for rare earth metals by about 6% annually since the start of the decade, with Japan allotted only 38,000 tonnes in 2009. Toyota and Honda alone will consume about that quantity and experts in Australia have predicted a wider global supply crunch within three years as demand surges beyond existing refinery and extraction capacity. In view of the importance of rare earth metals to its economy, the Japanese Government has initiated a search for alternative supply sources in Vietnam, Kazakhstan and elsewhere. The Japanese government has encouraged its manufacturers to adopt a supportive role in the financing and development of REE projects that is not take-over oriented. Japan's official development assistance (ODA) strategy calls for increased support for mining development in foreign countries, infrastructure development in the surrounding areas, active cooperation for technology transfer and protection of the environmental.

Japanese companies that are actively seeking REE projects worldwide include the following:

Sumitomo Corp.	at inception, Sumitomo planned to produce REEs in Kazakhstan through a joint venture established with state-owned nuclear power company Kazatomprom. Using Kazatomprom's facilities, REEs will be removed from uranium ore left over after uranium has been extracted. Annual output was expected to reach 3,000 metric tons in 2010 (slightly less than 10% of Japan's imports), however delays occurred resulting in a revised plan to export 1,500 tonnes in early 2013. The current status is uncertain.
Toyota Tsusho Corp.	plans to spend a total of 40 billion yen on natural resources development, mainly for rare earths, over the next five years. It intends to start extracting the metals from tin ore in Indonesia, and it is also considering developing mines in such countries as Mongolia. By expanding its rare-earth business, the firm hopes to secure stable supplies for

	Japanese carmakers like Toyota Motor Corp.
Marubeni Corp.	will start recycling rare earths through a subsidiary. It hopes to develop efficient recycling technologies in preparation for four or five years down the road, when more hybrid cars will be scrapped
Mitsubishi Corp.	entered a partnership with Neo Material Technologies (“**NMT**”) of Canada to recover by-products such as dysprosium and terbium from the Pitinga tin mine in Brazil. The two companies intended to form a joint venture to acquire the rights to purchase at least 20% of the mine's output. With the take-over of NMT by Molycorp, the status of this agreement is uncertain at this time.
Mitsui & Co.	plans to import a large volume of the rare metal from Canada. The move comes on the heels of the firm's investments in nickel and cobalt - other rare metals essential for manufacturing lithium-ion batteries. Mitsui has obtained exclusive sales rights to lithium produced at a mine that Canada Lithium Corp. owns in the Canadian province of Quebec. After shipping samples to potential customers, Mitsui plans to start importing around 2,000 metric tons of lithium a year from the mine in 2013 for sale to Japanese and South Korean manufacturers of lithium-ion batteries.

The nature of the potential crisis over shortages in rare earth metals is more acutely voiced in Japan which is a major producer of the REE magnets used in everything from high-performance electric motors to jewellery. However, the anticipated shortage of REEs has not materialized principally because the magnitude of the demand is relatively small in terms of the number of mines that can satisfy global demand. Efforts by China to dominate the market have resulted in some substitution of metals which has retarded demand growth. In a recent article “The Coming Rare Earth Metals Crunch” the writers pointed out that the world demand for rare earth metals used in cell phones, hybrid cars, wind turbines and many electronic applications was over 110,000 short tons per year, and projected by the US Geological Survey to grow some 71% to 188,000 tons by 2012. This growth did not materialize, in part due to the financial crisis. Global demand has remained at approximately 110,000 tonnes, and according to the USGS equal to about 0.1% of estimated global resources of 110 Mt. There is clearly no shortage of REEs in general except for certain HREEs which are scarce. The consumption of rare earth metals is expected to grow as the number of uses are found. Each Toyota electric Prius motor requires 1 kilogram (2.2 lb) of neodymium, and each battery uses 10 to 15 kg (22-33 lb) of lanthanum. That number will nearly double under Toyota's current plans to boost the car's fuel economy. The degree to which future prices will rally is uncertain. What is certain is that the major producers have the ability to flood the market with many of the rare earth metals most in demand. In the short term, refinery capacity may prove to have more of an influence on price than resource availability.

19.5 ENVIRONMENTAL

Uranium mineralization in the Elliot Lake area is relatively low grade and contained in highly indurated, conglomeratic host rocks. At an average grade of 2 lbs of U_3O_8 per ton of ore, the production of one ton of uranium oxide produced 1,000 tons of mine tailings. The 11 mill-sites in the Elliot Lake area, with an average area of 230 acres, are estimated to contain a total of over 149.3 million tons of tailings (MNDM), which contain low concentrations of naturally occurring radioactive elements, including traces of uranium, thorium, radium and other heavy metals, especially iron derived from pyrite and pyrrhotite. A significant amount of REEs is thought to be present in the tailings because only yttrium was commercially recovered as a by-product of uranium mining. Radium decay emits the radioactive gas radon (Rn_{226}) and other daughter products, such as bismuth (Bi_{222}) and polonium (Po_{210}), which are potential health hazards.

Given the foregoing contents, it is clear that any future reclamation of tailings must be geared towards:

- reducing direct gamma radiation from the impoundment area to essentially background levels,
- reducing the radon emanation from the impoundment area to the surrounding environment; and,
- stabilizing the tailings pile to prevent it from contaminating the groundwater through erosion, seepage, or water runoff.

Finally, the tailings remedial action must eliminate or minimize the need for additional work during on-going monitoring and maintenance program following reclamation. The two major concerns for tailings remediation involve covering the pile and stabilizing the embankment as the costs incurred by failure of the tailings cover or destabilization of the embankment can be substantial. Groundwater problems resulting from the exit of contaminated water from an inadequately protected tailings pile are difficult to predict, and can be very costly to bring under control. The presence of iron sulphides in the Elliot Lake tailings raises the concern of acid generation as a result of oxidative processes. The tailings contain approximately 5% pyrite and minor pyrrhotite. The oxidation of sulphides lowers the pH of the tailings and results in enhanced leaching of radioactive metals and other trace heavy metals. Potential re-dissolution of radionuclides held as insoluble barium sulphate sludge, a precipitate removed during the control of radium with barium chloride, may also result from exposure to acid water. For several decades, site remediation at Elliot Lake has successfully used a water-cover option to prevent acid generation by excluding oxygen from the tailings while at the same time providing a barrier to radon evolution. This approach requires that the tailings pile be uniformly levelled to eliminate the need for internal dikes, thus also reducing the risk of water release from internal dike failure.

The restoration costs for decommissioned uranium mines in Canada are given in Table 29. These costs, particularly those associated with Blind River deposits (bolded) should serve as a guide to Appia in planning its approach to conventional mine development in the Elliot Lake area. However, the possibility of using existing mine workings as a repository for tailings should be investigated.

Table 29
Restoration Costs of Shutdown of Uranium Production Facilities
in Canada in 1993 Dollars

Mine Name	Production (tonnes U)	Production Cost (M US $)	Tailings (tonnes)	Unit Costs (US $/t tailings)
Beaverlodge	17,500	10.55	5,800,000	0.75
Agnew Lake	**750**	**2.14**	**37,500**	**n.a.**
Madawaska	3,670	n.a.	4,460,000	0.04
Quirke	**43,700**	**29.87**	**46,000,000**	**0.35**
Panel	**9,200**	**16.23**	**15,000,000**	**0.65**
Stanrock	**10,400**	**10.39**	**5,700,000**	**1.71**
Denison	**56,100**	**15.58**	**63,300,000**	**0.65**
Rabbit Lake	58,900	18.51	14,100,000	0.46
Key Lake	74,400	20.39	4,700,000	0.92

Source: IAEA, 2002 – Elliot Lake operations are in bolded text.

In a meeting with the manager of the Ontario Ministry of the Environment (“**MOE**”) office in Sault Ste. Marie on 15 May, 2007, it was reported to WGM that all of the former uranium mines in the Elliot Lake area have been decommissioned and all mining and supporting infrastructure have been removed from the sites. The access roads to all sites are gated to prevent vehicular access. The MOE also reported to WGM the following:

- the Elliot Lake mine sites are under the administrative jurisdiction of the Canadian Nuclear Safety Commission (‘**NSC**”), a federal body established for the regulation and monitoring of all infrastructure in Canada related to the nuclear industry;

- the environmental monitoring of the decommissioned sites falls within the mandate of the Joint Review Commission ("**JRC**") which is composed of representatives from the Ministry of Northern Development and Mines, the Ministry of Natural Resources, the MOE and two federal bodies – Environment Canada and the Department of Fisheries and Oceans;
- the NSC is the main driver in setting objectives for the JRC;
- the MOE is aware that uranium exploration has been renewed in the Elliot Lake area and that considerable expenditures are being made in the search for new uranium deposits;
- there are no land withdrawals in the Appia project area that would negatively impact Appia's exploration plans; and,
- subject to explorers meeting statutory requirements, completing the permitting process and gaining the required approvals, there are no current regulations or policies that would preclude a return to production of any of the decommissioned mines, or the mining of new deposits.

20. DISCUSSION

20.1 GEOLOGY

The Elliot Lake uranium-REE deposits are paleoplacers within which the economic minerals are typically deposited in conglomerates at the base of a sedimentary cycle. The host rocks are contained within the Quirke Lake Syncline, a major east-west trending fold structure located north of the town of Elliot Lake. The deposits are stratabound, commonly occurring in stacked sheet-like bodies of quartz-pebble conglomerate. Mineralization is mostly disseminated along bedding planes and the highest grades are associated with higher concentrations of pyrite and well packed quartz pebbles. The weight of evidence suggests a sedimentary origin for the mineralization. The district wide presence of brannerite (UTi_2O_6), the main economic mineral, and U-bearing phosphates such as monazite ($[Ce,La,Nd,U,Th]PO_4$), xenotime ($Y\text{-}UPO_4$) and other rare earth minerals relates quite well to the weathering of a U-Th-REE enriched (granitic) source. Pyrite and to a much lesser extent, pyrrhotite, are the main minerals associated with uranium, occurring as overgrowths on detrital pyrite grains and on uraninite grains altering to coffinite.

The geology of the Appia claims and the Teasdale Deposit is well understood. Additional drilling is needed and will predictably add to the REE and U resource base notwithstanding the limited impact of small dikes and structures which are present.

20.2 EXPLORATION AND MINERAL RESOURCES

No recent exploration has been completed in the Appia project area prior to 2006. The last major historical exploration programs consisted of deep drilling by Kerr McGee from sites along the axis of the Quirke Lake Syncline. The average hole length was approximately 1,500 metres (5,000 feet). The drilling succeeded in testing the uranium-bearing Matinenda Fm. at points scattered across the basin at a kilometre-scale spacing (or more). The uranium-bearing Lower Reef horizon generally averaging 1.0-1.5 lbs U_3O_8 per ton was consistently encountered – this is in keeping with the general tenor of the deeper mineralization that was mined during the later stages of Elliot Lake's mining history. Most intersections contained a few narrow higher grading sections, commonly exceeding 3-4 lbs U_3O_8 per ton.

WGM drew the following conclusions detailed in Workman and Breede, (2011) based on the earlier exploration programs:

1) Appia's airborne magnetic and MegaTEM electromagnetic survey showed the presence of various structures and dikes within the Quirke Lake basin, but its follow-up IP survey failed to provide useful targets for drilling despite the recommendations of the geophysicist who interpreted the data. No further surface geophysics (mag, EM, IP) was recommended by WGM;
2) WGM's attempts, and subsequent attempts by Appia, to locate the casing of early drill holes were both successful in the Banana Lake and Teasdale areas, and some holes were used successfully by Appia in 2007 and 2008 to wedge deep secondary holes to allow confirmation of previous historical intersections;
3) In the Teasdale Lake Zone, Appia's 2007-08 winter drilling confirmed historical intersections which were concentrated in an area west of Teasdale Lake;
4) Appia's drilling on the Teasdale Zone allowed WGM to confirm the historical resource estimate of former Rio Algom Chief Geologist Doug Sprague as a valid expression of the amount of uranium in the Teasdale Zone;
5) Appia's drilling enlarged the area previously known to contain uranium resources and provided the basis for a NI 43-101 compliant Mineral Resources estimate, including those historical holes which WGM confirmed, comprising an Indicated Mineral Resource of 17.4 million tons (15.8 Mt) with an average grade of 1.10 lbs U_3O_8/ton (0.55 kg U_3O_8/t) and an Inferred Mineral Resource of 48 million tons (43.5 Mt) at the same grade (Workman and Vasak, 2008);
6) Appia's drilling formed the basis of a resource estimate for the Banana Lake Zone comprising Inferred Resources of 33.42 million tons (30.52 Mt) averaging 0.9 lbs U_3O_8/ton (0.45 kg U_3O_8/t), a grade that is 20% higher than the historical estimate of 0.76 lbs U_3O_8 per ton – (Workman and Breede, 2011);
7) Excluding historical holes for which REE assay data was lacking, and including assay data for the Upper Reef, the Intermediate Quartzite and the Lower Reef, WGM re-estimated the Teasdale Mineral Resources as follows in Tables 30 and 31.

Table 30
Summary of Teasdale Zone Uranium and Rare Earth Mineral Resource Estimate

Zone	Tonnes ('000)	Tons ('000)	TREE (lbs/ton)	U_3O_8 (lbs/ton)	Average Thickness (m)	Contained TREE ('000 lbs)	Contained U_3O_8 ('000 lbs)
INDICATED RESOURCES (LOWER REEF-INTERMEDIATE QUARTZITE-UPPER REEF)							
Total	**13,095**	**14,435**	**3.30**	**0.554**	**9.48**	**47,689**	**7,995**
INFERRED RESOURCES (LOWER REEF-INTERMEDIATE QUARTZITE-UPPER REEF)							
Total	**38,507**	**42,447**	**3.14**	**0.474**	**9.82**	**133,175**	**20,115**

Qualifying notes for Mineral Resources are contained in Table 24.

The individual REEs included in the TREE mineral resources are reported as follows in ppm in Table 31. For conversion purposes, 500 ppm = 1 lb/ton.

Table 31
Individual REE Resource Grade Composition Summary for Teasdale

Zone	Light REE (ppm)						**Heavy REE (ppm)**									
	La	Ce	Pr	Nd	Sm	Eu	**Gd**	**Tb**	**Dy**	**Ho**	**Er**	**Tm**	**Yb**	**Lu**	**Hf**	**Y**
INDICATED RESOURCES																
Total	422	745	73.8	247	41.1	1.7	26.2	3.2	14.3	2.3	5.8	0.8	4.6	0.7	5.2	59.4
INFERRED RESOURCES																
Total	401	709	69.9	232	39.0	1.6	24.6	3.0	13.5	2.1	5.5	0.7	4.4	0.6	5.3	56.5

Qualifying notes for Mineral Resources are contained in Table 24.

8) WGM concluded that the value of the REEs present in the Teasdale Zone will largely vest in cerium, lanthanum, neodymium and yttrium which account for 86.3% of the total REEs present in the sample population at large.

9) The commercial viability of REE mineralization was previously demonstrated by the historical recovery of yttrium as a by-product of uranium production at the Elliot Lake mines. These operations proved that separate facilities were not required to leach the REEs, and that once in solution, yttrium could be easily recovered. However the mine operators ignored the other REEs because the market was adequately served by deposits elsewhere.

10) Although the potential quantity and grade of a larger REE resource in the Teasdale Zone was conceptual in nature due to insufficient exploration, and inherent uncertainties exist that continuing exploration might not delineate a mineral resource, WGM concluded that the close association between REEs and U, as shown in Figure 28, indicated good potential that the area of historical drilling largely excluded from the current Appia resource base would contain a significant REE resource, and that a simple linear extrapolation would indicate the potential for a resource of some 400 to 450 Mlbs of total REEs in this historical area at an average grade of approximately 3 to 4 lbs/ton.

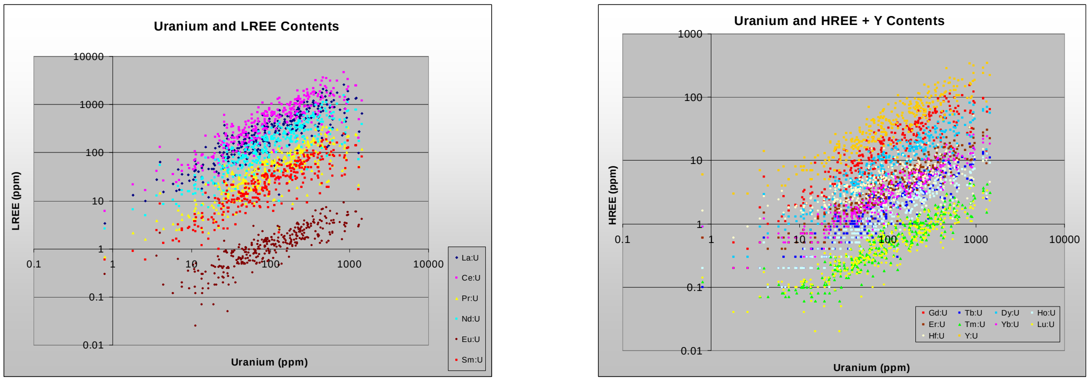

Figure 28: Illustrations of the Close Correlation Between Rare Metals and Uranium Contents in Appia Drill Holes.

11) The PEA announced by Pele regarding its Eco Ridge Project based on a 9,400 tonne per day operation producing 10.7 Mlbs of total rare earth oxides (REOs) and 24.9 Mlbs of U_3O_8 over a 14-year mine life and having and internal rate of return of 47% could be used as a very approximate measure of the economic basis for pursuing the exploration of the Teasdale Deposit due to the similarities of the two projects. A minimum thickness of 1.8 m produced an average thickness of 2.76 m for the main resource zone.

12) The uranium production from all Elliot Lake mines totalled some 362 million pounds of U_3O_8 from approximately 167 million tons of ore predominantly by conventional room and pillar mining of the Lower Reef, generally with a mining height of 2.44 metres[19]. In most instances, the average height was not significantly greater than this. WGM and Appia concluded that the discovery of significant REE mineralization in the Upper Reef constituted an excellent opportunity to develop a larger scale-lower cost underground operation utilizing larger equipment and an average mining height of approximately 9.7 metres.

13) WGM concluded that the Teasdale Deposit was unconstrained by geological parameters or by exploration data, and strongly recommended additional diamond drilling in the amount of 15,405 m budgeted at \$7 million. WGM also recommended additional drilling on the Banana Lake Zone in the amount of 12,800 m budgeted at \$7.6 million.

14) The historical uranium resources present on the Appia claims, as follows, which have not been confirmed by Appia, are not compliant with current statutory requirements under Canadian Securities regulation NI 43-101, but notwithstanding that fact, these estimates by Rio Algom totalling approximately 200 million pounds of U_3O_8 were based on actual mining experience and reasonable assumptions:

Teasdale Lake Zone	17,458,200 tons	at 1.206 lbs U_3O_8/ton	(20,787,200 lbs)
Gemico Block #3 Zone	42,800,000 tons	at 0.38 lbs U_3O_8/ton	(16,264,000 lbs)
Gemico Block #10 Zone	20,700,000 tons	at 0.75 lbs U_3O_8/ton	(15,525,000 lbs)
Banana Lake Zone	175,800,000 tons	at 0.76 lbs U_3O_8/ton	(133,608,000 lbs)
Canuc Zone	7,000,000 tons	at 1.86 lbs U_3O_8/ton	(13,020,000 lbs)
Total	263,758,200 tons	at 0.76 lbs U_3O_8/ton	(199,204,200 lbs)

15) Based on the foregoing, WGM believes there is excellent potential for many of the zones to increase in size as the former estimates were highly constrained by the claim boundaries of the individual companies, a limitation that no longer exists on the Appia Property. In light of Appia's recent work which shows economically interesting REE mineralization occurring in the Upper Reef in addition to what was mined previously, WGM believes this also contributes to the mineral potential.

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[19] The Lacnor Reef, also known as the Lower Reef, was the first conglomerate bed deposited above the basement and was generally thin and discontinuous. In the Lacnor and Milliken Mines, it was mined at an average height of 2.44 m (8 ft.). The Nordic Reef, an overlying conglomerate horizon, was the primary unit mined at all the four of the mines located in the Nordic Channel with an average height of 3.0 m (10 ft.). At the Nordic Mine, mining exploited an upper conglomerate horizon known as the Pardee Reef with an average mining height of 1.5 m (5 ft.), but reaching thicknesses of 3.0 m (10 ft.) at the Stanleigh Mine.

16) In reviewing how low-grade underground uranium miners achieved success in such places as New Mexico, Appia should examine the potential cost savings of constructing a central processing facility in co-operation with other potential uranium producers in the Elliot Lake area. Given the high capital cost considerations, and the obvious similarities of the ore, there would be obvious benefits if the operators could share the cost of processing infrastructure.

17) Lastly, WGM determined that evidence indicated the presence of considerable, readily accessible thorium and REEs present in the tailings from the various Elliot Lake mines. Notwithstanding the environmental, social and political challenges of opening the tailings sites to redevelopment, there are few technical challenges in leaching the metals from the tailings which are estimated to contain approximately 0.87 lbs thorium oxide (ThO_2) and 0.66 lbs REEs per short ton, much of it contained in the mineral monazite (Robertson and Steenland, 1960). The 180 Mt of ore mined and processed would equate to approximately 155 Mlbs of ThO_2 in tailings and 120 Mlbs of REEs, however Prasad and Ruzicka (1992) report that the Denison mill recovered some of the thorium and REEs whereas the Rio Algom mill recovered very little if any, consequently the amount of contained metal will vary from site to site depending on the mine's head grade and the operator. The value of the thorium is highly conditional, largely based on the future development of thorium reactors which have been discussed in scientific media since the 1970s, but as far as WGM is aware, India is the only country with an established research program. Interestingly however, Canada's CANDU reactor can operate on blended uranium-thorium fuel cycles. Despite the thorium reactor's many advantages, only an extreme shortage of uranium fuel would cause a significant technological shift in favour of thorium, and so the Elliot Lake thorium resource must be thought of in a long-term, perhaps very long-term, context.

21. INTERPRETATION AND CONCLUSIONS

21.1 GEOLOGY AND MINERAL RESOURCES

Appia’s exploration to date, carried out under the guidance of WGM, has been successful in confirming historical drill hole intersections in both the Banana Lake Zone and the Teasdale Zone. Deep drilling has outlined a NI 43-101 compliant resource in the Banana Lake area that is higher grading than previously indicated by historical drilling. However, it is in the Teasdale Zone underlying the western portions of Quirke Lake where Appia’s shallower drilling has both confirmed and enlarged a historical resource (Tables 32 and 33).

Table 32
Summary of Teasdale Zone Uranium and Rare Earth Mineral Resource Estimate

Zone	Tonnes (‘000)	Tons (‘000)	TREE (lbs/ton)	U_3O_8 (lbs/ton)	Average Thickness (m)	Contained TREE ('000 lbs)	Contained U_3O_8 ('000 lbs)
INDICATED RESOURCES							
UR	6,733	7,422	4.20	0.484	4.61	31,199	3,593
IQ	3,006	3,314	1.98	0.259	2.27	6,578	0.857
LR	3,355	3,699	2.68	0.958	2.60	9,912	3,544
Total	**13,095**	**14,435**	**3.30**	**0.554**	**9.48**	**47,689**	**7,995**
INFERRED RESOURCES							
UR	18,326	20,201	3.87	0.421	4.33	78,080	8,498
IQ	10,209	11,254	1.64	0.184	2.78	18,464	2,070
LR	9,972	10,992	3.33	0.869	2.71	36,631	9,564
Total	**38,507**	**42,447**	**3.14**	**0.474**	**9.82**	**133,175**	**20,115**

Qualifying notes for Mineral Resources are contained in Table 24.

The individual REEs included in the TREE mineral resources are reported as follows in Table 33.

Table 33
Individual REE Resource Grade Composition Summary for Teasdale

Zone	Light REE (ppm)						**Heavy REE (ppm)**									
	La	Ce	Pr	Nd	Sm	Eu	**Gd**	**Tb**	**Dy**	**Ho**	**Er**	**Tm**	**Yb**	**Lu**	**Hf**	**Y**
INDICATED RESOURCES																
UR	540	951	93.9	313	51.7	1.9	32.8	3.9	17.2	2.7	7.0	0.9	5.5	0.8	6.8	72.9
IQ	256	452	44.9	148	24.4	1.0	14.7	1.8	7.7	1.2	3.1	0.4	2.5	0.4	3.6	30.6
LR	332	596	59.4	201	35.1	1.7	23.2	3.0	14.2	2.3	5.9	0.8	4.5	0.6	3.3	58.1
Total	**422**	**745**	**73.8**	**247**	**41.1**	**1.7**	**26.2**	**3.2**	**14.3**	**2.3**	**5.8**	**0.8**	**4.6**	**0.7**	**5.2**	**59.4**
INFERRED RESOURCES																
UR	498	876	85.9	285	47.2	1.8	29.3	3.5	15.9	2.5	6.5	0.9	5.3	0.8	6.8	67.9
IQ	213	374	37.0	122	20.0	0.8	12.3	1.4	6.4	1.0	2.6	0.4	2.2	0.3	3.3	26.5
LR	417	747	73.9	249	43.4	1.9	28.5	3.6	16.4	2.6	6.6	0.9	5.2	0.7	4.5	66.4
Total	**401**	**709**	**69.9**	**232**	**39.0**	**1.6**	**24.6**	**3.0**	**13.5**	**2.1**	**5.5**	**0.7**	**4.4**	**0.6**	**5.3**	**56.5**

Qualifying notes for Mineral Resources are contained in Table 24.

In addition to Lower Reef horizon which has been the main zone mined in the past, the discovery of an Upper Reef in this area has allowed Appia to build resource blocks taking in both reefs and averaging 9.7 m in thickness. Compared to the historical average thickness of approximately 2.4 m, the potential for renewed mining on this scale allows for larger scaled equipment than was used in the past and the economies of scale that can be accomplished as a result.

The current Mineral Resources take in both uranium and rare earth element mineralization and is based on the six holes totalling 2,650.2 m (8,695 ft) completed by Appia in 2008-09 and the drilling completed during 2012 which totalled 8,130.2 m (26,673 ft), including 16 holes from surface, one wedged hole and one abandoned hole. The current resources do not take in some historical Conecho drill holes in the area that do show uranium mineralization, but fail to report grades in the Upper Reef (which was not sampled and analysed), and fail to report any assays for REEs.

The in-fill drilling to date indicates that it very likely that additional drilling will continue to enlarge the zone as well as converting most of the Inferred Resources to Indicated Resources. This is a reflection of the relatively uniform character of the stratabound mineralization.

The Elliot Lake uranium deposits can be a major co-producer of REEs, and the value of the REEs must be incorporated into any economic model used to evaluate the Teasdale deposit. The international market for rare earth metals has increased markedly in the last decade due to the advent of new technologies in which REEs play a small but vital role. China's position as the preeminent producer of REEs derives from a combination of official producers which account for 98% of global production and illegal miners (black-market producers) which account for an uncertain amount of production. China has acted to shut down illegal operators and curb the environmental impact of official producers while at the same time restricting exports. The net effect has been a significant decrease in China's supply of REEs to the global market which is probably on the order of 110,000 to 120,000 tonnes of REEs annually, and relatively stable over the past few years. What were initially strong increases in REE prices due to the fear of shortages has morphed into what appears to be a well-supplied market in the LREEs due to a return to production of US and Australia, and an under-supplied market for many of the HREEs. While it is unlikely that REE prices will re-gain the highs seen in recent years, the current prices in mid-2013 suggest a degree of supply-demand stability that should allow Appia an acceptable level of certainty for planning purposes.

The current and future global consumption of uranium by nuclear power reactors is measurable and predictable, and based on current data, a shortage of uranium for reactor fuel is forecast for the near future. The higher uranium prices which are a natural outcome of any mineral shortage projects like Appia's which have are well located in respect to infrastructure.

Without doubt, the Elliot Lake deposits offer the potential for a stable, long term supply of uranium oxide and rare earth metals. WGM believes that the world will not indefinitely ignore the presence of more than 200 million pounds of readily extractable uranium remaining in the Elliot Lake deposits, and many times that in terms of rare earth metal content.

21.2 MINERAL PROCESSING

It has been demonstrated that the uranium and REE mineralization are interrelated and can be mined without changes to the basic mining plan. The metallurgy of Elliot Lake uranium mineralization is well known in respect to its milling and leaching characteristics, including the application of bacteria leach technology to uranium. Evidence based on the historical recovery of yttrium proves that REE recovery can utilize much of the same processing technology as that used for uranium production.

Based on the preliminary metallurgical testing, the favoured flowsheet option includes a simple grind, a flotation process to recover a high grade concentrate, acid baking of the flotation concentrate and acid leaching of the flotation tailings with the total recovery levels expected from a 36-hour whole ore leach on finely ground ores as shown by the green line in Figure 29 together with the test results from other flowsheet trials. This approach can result in a high extraction of U (~90%) and most REEs (80-90%) from Appia's Teasdale U-REE deposit. It is believed that planned additional testwork and data analysis will substantiate these data and probably lead to reduced overall processing costs and improved recovery.

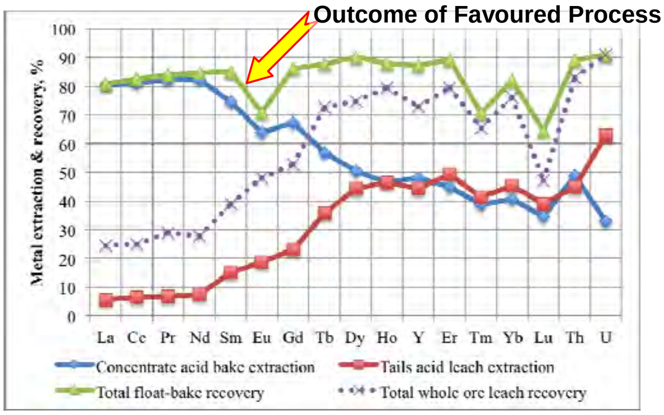

Figure 29: Green line illustrates recoveries for the favoured processing option compared to whole ore leaching

21.3 OTHER PROJECT FACTORS

Being located in a brownfields area that has been impacted by a long history of uranium and REE mining and production, and characterized as brownfields project, the Appia project should be more easily permitted than any similar greenfields project. The Appia project also has additional positive attributes such as:

1) the previous mining has demonstrated that uranium tailings can be stabilized and managed such that the impacts are successfully mitigated without serious adverse consequences for the local environment;
2) Appia's Teasdale Deposit is close to the Cameco uranium refinery situated west of Elliot Lake near the town of Blind River;
3) the mineralization is stratabound with excellent lateral grade and thickness continuity;
4) the associated REE production is another potential revenue generator that could exceed the value of uranium production;
5) mine rock conditions were favourable for underground development and their engineering properties are well understood;
6) the average thickness of the deposit is approximately 3 times greater than the average thickness mined in the past which should result in commensurate reductions in mining costs;
7) Appia bears no responsibility for potential environmental legacy issues in the future arising out of previous mining activities;
8) other companies delineating new uranium and REE resources in the area, such as Pele Mountain Resources, as well as companies such as Denison Mines and Rio Algom which continue to hold significant Mineral Resources, may present opportunities for joint mill ownership or toll milling;
9) proximity to the Panel Mine which offers the potential for shaft revitalization and underground openings for mine infrastructure as well as in-stope leaching and waste disposal;
10) the project is located in Ontario, Canada, in an area that has a long mining tradition and in an investor-friendly jurisdiction that supports flow-through financing;
11) a hard-working and knowledgeable local labour force lives in northern Ontario;
12) excellent year-round project access and close to major supply centres;
13) the property is large, extending over an area adjacent to and down-dip of former mine workings; and,
14) the potential to develop a large scale thorium resource (+REEs) from Elliot Lake tailings is a potential future consideration since India and some other countries are now working on the engineering challenges of using thorium fuel.

WGM previously determined though its discussions with government representatives in Sault Ste. Marie, there are no impediments in the mining and environmental statutes that would constitute fatal flaws to the Appia project. There are no land withdrawals in the Appia project area that would negatively impact Appia's exploration plans. However, it is clear that the new Ontario Mining Act imposes on Appia a duty to consult with local stakeholders prior to undertaking advanced exploration programs which would include diamond drilling. Recent changes in the Mining Act are intended to avoid future conflicts, and do not impose a need for Appia to give up any of its mineral rights. Local stakeholders are expected to discuss any concerns in good faith and not un-necessarily impede exploration.

In its previous discussions with local mining engineer, Bob MacGregor who has been active with Pele Mountain Resources, WGM understands that the residents of Elliot Lake and its Chamber of Commerce are intensely interested in the new jobs and tax revenue that renewed mining would bring to the town.

22. RECOMMENDATIONS

22.1 EXPLORATION

It is clear that the 2012 in-fill and step-out drilling on the Teasdale Deposit has enjoyed a high degree of success in confirming the resources where they were predicted by the geological model. Inferred resources were up-graded to Indicated Resources and additional resources were defined. Appia should continue its drilling of the deposit in accordance with the general provisions of the previous recommendations which envisioned a total of 39 diamond drill holes totalling 15,405 m. Of this, Appia has completed 16 holes totalling 8,130.2 m (26,673 feet), and effectively generated intersections at 14 of the points previously recommended. Appia's drilling included some areas not previously recommended, however with positive results, these have contributed to the mineral resources in areas that were originally thought to have lower priority.

WGM's original drilling program envisioned winter operations to facilitate drilling from ice platforms. Appia has recently used a barge to move the drill between sites and demonstrated the ability to produce representative and useful intersections from the same drill set-up. In this context, WGM has reconfigured its drilling proposal to account to the exploration results to date, to reflect the use of a barge for drill moves and to reduce the number of drill moves by using the same site to "cone-drill" drill multiple holes from the same set-up in much the same manner as underground drilling is designed. The relatively shallow dip of the mineralized horizons (reefs) allows a significant amount of flexibility regarding drilling angles and drill hole bearings. As an example, whereas drilling down dip is generally frowned upon as it can produce unrepresentative and misleadingly long intersections, drilling down dip at -70° on the Teasdale Zone produces an intersection angle of 50° which is within the range of what would be considered "acceptable".

WGM recommended drill sites and hole orientations have been confirmed through 3D computer modelling. WGM's proposal for the next drilling program follows in Table 34.

The locations for the drill holes proposed in the next (2014) drilling program are shown in Figure 30. The proposed drilling is predominantly intended to add Inferred Resources to the Appia project rather than to up-grade Inferred Resources to Indicated Resources. Some up-grading will occur as a natural consequence of the overlapping of spheres of drill hole influence. An increase in mineral resources will allow for better project planning insofar as a future NI 43-101 compliant preliminary economic assessment is concerned because the inclusion of Inferred Resources are allowed in such studies whereas higher level feasibilities are only permitted to use Indicated and Measured Resources. Appia understands that additional in-fill drilling will be required in the future to support feasibility-level studies.

A budget for the drilling is provided in the following section based on the experience gained during the 2012 drilling program.

Table 34
WGM Proposal for Appia Energy Drilling - Teasdale Lake Zone, 2014

DDH #	UTM Location		Elevation (m asl)	Dip	Bearing	Length (m)	Comments
	Easting	Northing					
AEC 14-18	383283	5149503	340	-45°	303°	320	Same location as AEC 12-03
AEC 14-19	383151	5149382	338	-45°	262°	410	Same location as Q-07-01
AEC 14-20	383112	5149040	361	-64°	047°	400	Same location as AEC 12-13
AEC 14-21	383112	5149040	360	-60°	135°	520	Same location as AEC 12-13
AEC 14-22	382440	5148785	338	-71°	313°	500	Same location as AEC 12-04
AEC 14-23	383060	5148110	350	-74°	159°	720	Same location as historical R-4
AEC 14-24	383060	5148110	350	-63	068°	670	Same location as AEC 14-23
AEC 14-25	383250	5147800	352	-60°	061°	750	New location
AEC 14-26	383250	5147800	352	-84°	165°	720	Same location as AEC 14-25
AEC 14-27	384350	5148530	365	-47°	333°	520	Previously proposed as Q-11-20
AEC 14-28	384350	5148530	365	-52°	260°	570	Same location as AEC 14-28
AEC 14-29	384350	5148530	365	-57°	106°	550	Same location as AEC 14-28
AEC 14-30	384350	5148530	365	-47°	041°	500	Same location as AEC 14-28
AEC 14-31	384350	5148530	365	-60°	180°	600	Same location as AEC 14-28
	Number of Holes = 14					**7,750 m**	

All drill core must be logged and analysed for U and REEs in accordance with established industry practices. Core recovery has been excellent to date at 95% or better. In respect to the additional cost of down-hole radiometric surveying, WGM does not see a significant technical advantage in down-hole spectrometer logging over the use of a hand-held spectrometer on the drill core, however down-hole surveying should be used if core recoveries are less than optimal. All drill core samples should be analysed for uranium using a solvent (acid) extraction process rather than by neutron activation analysis which measures total contained uranium rather than leachable uranium. All samples should be analysed for REEs using a conventional technique and for trace elements using an ICP-based multi-element technique.

APE REP / APE_18_DH_Proposed_Teasdale_Lake.cdr
Last revision date: Tuesday 30 July 2013

Watts, Griffis and McOuat

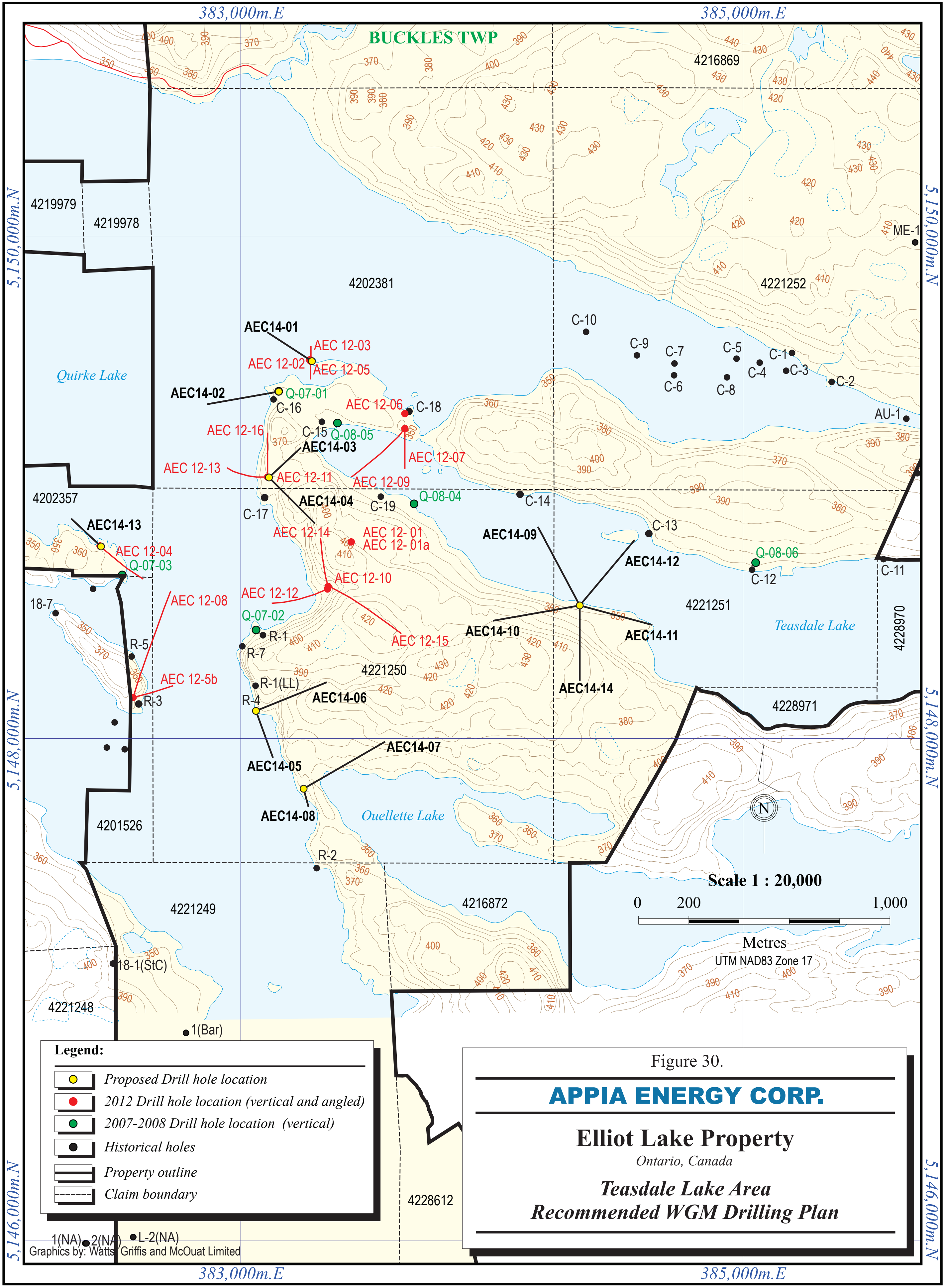

Figure 30.

APPIA ENERGY CORP.

Elliot Lake Property

Ontario, Canada

Teasdale Lake Area
Recommended WGM Drilling Plan

22.2 METALLURGICAL TESTING

In light of the encouraging results that have come from the most recent components of the metallurgical testing program carried out by SGS Lakefield, additional testing is recommended to focus on the preleach and acid bake testing of the flotation concentrate. Additional testing should address uranium and REE recovery to identify and design the most logical processing option to extract the metals from the leached slurry.

22.3 MINERAL ECONOMICS

Appia should undertake a preliminary economic assessment as soon as the metallurgical testing program establishes a well defined flowsheet for the processing of Teasdale ore.

The supply and demand fundamentals of the uranium market are dynamic, but subject to easily quantified measurements since the nuclear fuel demands of power reactors can be forecast based on electrical generating capacity. Like new uranium mines, reactors also require considerable time for planning and construction and this allows surpluses and deficits in uranium markets to be forecast with a high degree of certainty relative to other mineral commodities. Nevertheless, uranium deposits are becoming increasingly difficult to find, and the permitting of such deposits is requiring longer and longer lead times. If past experience is a measure, uranium fuel fabricating infrastructure is likely to lag mine output. Over the longer term, key factors will be substantially increased demand due to new reactor builds balanced against increasing production from Australia and Kazakhstan, and new production coming from countries such as Mongolia that had little or no output in the past. As a medium term goal, Appia should undertake a detailed review of the uranium industry to ensure its understands the market as it is foreseen to develop in the next two decades.

The recent findings of the World Nuclear Association, which meets every two years (most recently in September 2011), should be taken as a guide to overall plans, however in this period of great economic turmoil, the forecasts of most experts contain a wide area of uncertainty between high market and low market scenarios. WGM is uncertain whether the current findings are useful in this economic climate, and so Appia's economic study should be completed no sooner than the 2013 year end.

22.4 PUBLIC DIALOGUE

Appia must act proactively to ensure that its activities are consistent with government regulations and policies. Appia should open direct contact with the Elliot Lake Joint Review Commission and its constituent members, the Ontario Ministry of Northern Development and Mines (“**MNDM**”), the Ontario Ministry of Natural Resources, the Ontario Ministry of the Environment, the (Federal) Environment Canada and the Federal Department of Fisheries and Oceans.

The new Ontario Mining Act came into force in early 2013. The new law and its regulations do not directly affect Appia’s mineral rights or land tenure rights. However, as stated previously in Workman and Breede (2011), the new law includes sections mandating that Appia file an Exploration Plan (“**EP**”) with the MNDM and consult with First Nations (“**FN**”) representatives prior to executing its exploration plans. Upon receipt of the EP, the MNDM which will place Appia in touch with the relevant FN representative who will review the plan in respect to its impact on the FN. It is unclear how this will affect Appia’s exploration activities, however all diamond drilling work must be the subject of an EP and a 30-day review period. The law and its attendant discussion papers also state that FN do not have a right of refusal insofar as approving or disapproving the EP or by holding Appia hostage to negotiating an interest in the mineral rights held by the company. The intent of the “consultation” is to ensure that Appia and the FN are aware of each other’s concerns and that Appia has the opportunity to mitigate impacts. Many exploration companies have found a ready and willing workforce in the FN communities close to the exploration site. As Appia’s work is mainly drilling in brownfields areas that are not treaty land or land subject to a land claim, WGM understands that very little consultative work is actually required, however Appia’s management should familiarize itself with the new laws. First Nations (“**FN**”) communities are located in watersheds down-stream from the former mining operations. Early consultation is recommended with FN representatives to ensure an inclusive rather than exclusive dialogue. Marginalizing this community may spawn groups that are especially resistant to uranium mining. Appia should develop an open dialogue with Pele Mountain Resources which may be in the same situation in respect to its nearby project. Future development planning will require that Appia study the implications of new law and, if deemed necessary, engage suitable counsel to give it guidance relevant to its planning process.

As the Elliot Lake mine sites are under the administrative jurisdiction of the Canadian Nuclear Safety Commission (‘**NSC**”), Appia should also contact this federal body to ensure that its activities are known to those responsible for overseeing the regulation and monitoring of the Elliot Lake nuclear infrastructure, and for setting objectives for the Joint Review Commission.

22.5 OTHER RECOMMENDATIONS

WGM tenders the following recommendations which have been numbered for convenience.

1) The uranium and REE Mineral Resources of the Teasdale Deposit should be updated after the recommended drilling is completed.

2) An attempt should be made through Natural Resources Canada (NRCan) archives and other sources including library records (microfiche and digital records) to assemble a complete production and exploration history for the Elliot Lake camp. This should include purchasing copies of all published books, reports and other information on the history of Elliot Lake.

3) Additional work needs to be done to precisely determine the locations for new drill sites to test and enlarge the Banana Lake Mineral Resources. A budget for approximately 10-15,000 m of drilling is needed to adequately test the Banana Lake Zone. This drilling should be staged in accordance with a long-term plan, but Banana Lake drilling is considered to be a second priority task at this time with Teasdale Zone drilling considered to be a first priority. However, as Appia's drilling to date has essentially confirmed the viability of the historical estimate made by Rio Algom, there is clear potential for defining nearly 200 Mlbs of U_3O_8 in this zone.

4) Where practical in the Banana Lake Zone, the redrilling of the existing Kerr McGee holes and wedging from such holes is justified as a means of quickly and cost-effectively building a uranium resource base in some areas of the Appia property. During 2008, Appia has enjoyed some success in pursuing this approach. Wedging off-hole at a distance of 500 m above the Matinenda Formation should produce additional intersections at least 50 m away from the initial pierce point. By using multiple wedged holes in this way, the variability of mineralization can be tested and the resource potential assessed at a significantly lower cost that redrilling from surface.

5) The feasibility of using the existing abandoned Panel Mine workings for the disposal of waste rock, tailings or any other purpose largely hinges on whether they are assessable which in turn depends on the means of access and whether they are dry or flooded. The question needs to be answered as a necessary component of any future feasibility-level study.

6) A dialogue should be initiated with Pele Mountain Resources to explore the feasibility of constructing a central milling and processing facility for Elliot Lake ores as a means of improving the economic viability of individual projects. All discussions would necessarily be contingent on the discovery of a resource base of sufficient size and grade to justify a production decision.

7) A dialogue should be initiated with government authorities to determine how best Appia can carry forward its exploration on certain of its claims that now have restricted access due to on-going impact mitigation work, and are thus subject to restrictions on surface activities.

23. PROGRAM BUDGET

In the past, WGM has recommended a staged exploration program that, over time, has minimized risk by building slowly from the established facts concerning the historical work. This began with Appia's early drilling of the Teasdale and Banana Lake Zones, its follow-up work on the mineralogy of the uranium-bearing zones, its identification of associated REE mineralization, the re-estimation of the Teasdale mineral resources based on a combination of current and historical data, and finally, Appia's recent drilling on the Teasdale Zone which has substantially increased the Mineral Resources and increased the level of confidence overall in the project.

WGM proposes a budget of C $7,153,000 for a multi-year exploration project according to the following budget (Table 35). WGM believes that this exploration is justified based on the positive results of Appia's exploration programs completed to date, however, the recommended program is divided into first and second priority drilling programs: a high priority 14-hole (7,750 m) program on the Teasdale Zone and a lower priority multi-phase (17,600 m) program on the Banana Lake Zone comprising 8 deep holes from surface and 8 wedged holes. In carrying out this work, drilling on the Teasdale Lake Zone offers Appia the greatest potential for adding value to the project in the form of NI 43-101 compliant uranium and rare metal Mineral Resources. WGM has also included a secondary budget for additional public consultations, contracted metallurgical testing, data acquisition, a preliminary economic assessment based on an up-dated Mineral Resource estimate and a new technical report compliant with NI 43-101 (Table 36).

The proposed exploration work will substantially exceed Appia's needs insofar as exploration assessment requirements are concerned. All costs are in Canadian dollars. To place this budget in context, it represents an investment of less than 10 cents (Canadian $0.10) per pound of historical uranium oxide resources on the Appia exploration property if the previous estimates of Rio Algom and others can be shown to be correct. It represents an investment of 16 cents per pound (Canadian $0.16) per pound of uranium oxide resources currently outlined to NI 43-101 standards on the Appia Property.

Although it is of secondary interest at this time, the proposed Banana Lake drilling, comprising 8 deep drill holes and 8 wedged holes, is required to test the northerly, westerly and south-easterly extensions of mineralization originally discovered by Kerr McGee and recently confirmed by Appia. For planning purposes, three phases of drilling are proposed for budgetary and cash flow reasons. This program should be executed in a flexible manner that is responsive to actual results. Drill hole locations do not significantly influence length, but certainly may influence overall results in respect to uranium contents. Careful attention to the geology of the uranium-bearing zones (reefs) is required. In some areas, the hole locations may allow for slightly shallower uranium intersections as the zone is traced to the north and away from the centre of the basin, but drill site elevation (above sea level) will probably have a greater impact on hole length. A budget is also provided for wedging off the new holes to allow Appia to develop additional cuts through mineralization using the original hole as a pilot. These wedged holes can be used to demonstrate grade and thickness continuity.

Table 35
Appia Energy Corp. Budget for Recommended Diamond Drilling and Associated Work

Item		Description	Amount	Unit Cost	Unit Totals	Total
Exploration Drilling						
Teasdale Zone		14 Diamond Drill Holes to Define New U and REE Resources	7,750 m	$175 per metre	$ 1,350,000	
		Barge for Drill Transport		approx $26/m	200,000	
		Miscellaneous Boat, Transport and Delivery		approx $2/m	15,000	
		Project Management and Geological *	on 7,750 m	approx $30/m	232,500	
		Core Cutting *	on 7,750 m	approx $5.50/m	42,500	
		Sample Assaying *	1550 samples	@ $75 / sample	120,000	
		Metallurgical Testing – allowance for supervision			100,000	
		Office Rental, Room & Board *	on 7,750 m	approx $2.00/m	15,000	
		Consumables & Miscellaneous Costs *	on 7,750 m	approx $1.60/m	12,000	
		Contingency on Subtotal	~5% of costs above ($2,087,000)		105,000	
		Sub-Total Teasdale Zone Drilling			**$ 2,192,000**	**$ 2,192,000**
Banana Lake Zone	Phase 1	2 diamond drill holes to test SE & NW extensions of Zone	3,200 m	$175	$ 560,000	
		2 wedges (including rig and crew time)	2	$20,000	40,000	
		2 wedged holes from initial pilot holes	1,200 m	$200	240,000	
		Project Management and Geological	4,400 m	$30	132,000	
		Core Cutting	on 4,400 m	approx $5.50/m	25,000	
		Assaying Samples	880 samples	$75	66,000	
		Room & Board	4,400	$10	44,000	
		Consumables & Miscellaneous Costs	4,400	$5	22,000	
		Contingency on Subtotal	10% of costs above ($1,129,000)		110,000	
		Subtotal for Phase 1 Drilling			**$1,239,000**	
	Phase 2	4 diamond drill holes to test NW & SE extensions of Zone	6,400 m	$175	1,120,000	
		4 wedges (including rig and crew time)	4	$20,000	80,000	
		4 wedged holes from initial pilot holes	2,400 m	$200	480,000	
		Project Management and Geological	8,800 m	$30	264,000	
		Core Cutting	on 8,800 m	approx $5.50/m	50,000	
		Assaying Samples	1760 samples	$75	132,000	
		Room & Board	8,800 m	$10	88,000	
		Consumables & Miscellaneous Costs	8,800	$5	44,000	
		Contingency on Subtotal	10% of costs above ($2,258,000)		225,000	
		Subtotal for Phase 2 Drilling			**$ 2,483,000**	
	Phase 3	2 diamond drill holes to test SE & NW extensions of Zone	3,200 m	$175	$ 560,000	
		2 wedges (including rig and crew time)	2	$20,000	40,000	
		2 wedged holes from initial pilot holes	1,200 m	$200	240,000	
		Project Management and Geological	4,400 m	$30	132,000	
		Core Cutting	on 4,400 m	approx $5.50/m	25,000	
		Assaying Samples	880 samples	$75	66,000	
		Room & Board	4,400	$10	44,000	
		Consumables & Miscellaneous Costs	4,400	$5	22,000	
		Contingency on Subtotal	10% of costs above ($1,129,000)		110,000	
		Subtotal for Phase 3 Drilling			**$ 1,239,000**	
		Total for Banana Lake Zone Drilling				**$ 4,961,000**
		GRAND TOTALFOR TEASDALE LAKE AND BANANA LAKE ZONES				**$ 7,153,000**

* *all support costs are factored on a per metre basis*

Table 36
Appia Energy Corp. Budget for Recommended Supporting Work and Studies

Item	Description	Unit Cost
Mineral Economics Study	Review of uranium market, reactor construction plans, supply-demand criteria & delivery schedules.	$50,000
Public Dialogue	Proactive dialogue and consensus building with Elliot Lake and First Nations community leaders	$50,000
Data Acquisition	Search for complete historical information through library/university archives & private sources.	$30,000
Metallurgical Study	Based on proposal received from SGS-XRAL Analytical Services (Lakefield)	$300,000
Preliminary Assessment Study	Updated mineral resource estimate and economic evaluation of the Teasdale Deposit; mining/processing options.	$120,000
	Sub-Total	$550,000
Contingency	~10% of costs above	$55,000
	Total of Incidentals for Project Support	$605,000

In respect to drilling the Teasdale Lake Zone, WGM has previously recommended that Appia's exploration program be staged so that, over time, risk can be minimized by building slowly from the established facts concerning the historical work. A program of 14 diamond drill holes (7,750 m) is proposed on approximate 200-400 m spacings to enable delineation of additional Inferred Resources for uranium and REEs while allowing for some up-grading of Inferred Resources to the Indicated category[20]. All of the proposed holes are located on the shoreline of Quirke Lake, or inland. The locations of certain holes may be amended or eliminated as the drilling progresses, however, the overall amount of drilling should not vary significantly from that proposed herein.

The obliquely angled holes recommended in the table with shoreline set-ups are intended to replace holes that would otherwise have to be drilled from winter ice. Some of the proposed drilling could be accomplished during the winter using ice platforms as soon as the winter ice thickens to the point where it will can support a work crew. While this might provide for greater flexibility in drill hole locations, and provide for some cost savings in terms of reducing hole lengths, additional costs would be incurred for helicopter support and controlling the environmental impact of run-off from the drilling fluids which otherwise

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[20] It must also be noted that the confidence level of Mineral Resources, as defined under NI 43-101, is also dependent on market forces, especially the Term Market. WGM does not view Spot Market activity as relevant in the context of the Appia Project.

would drain directly into the lake. A budget of $120,000 would also be required for the labour involved in flooding and maintaining the ice.

On-going exploration should be directed at developing a separate budget for confirmation of uranium resources in other mineralized zones such as the Canuc Zone and in the Gemico Zones. In WGM's view, an initial budget for 2,000 to 3,000 m of drilling costing approximately $750,000 to $1.5 million would be appropriate for such purposes. Future drilling on Gemico Block 3 will require approval from the federal Nuclear Safety Commission to allow Denison to grant Appia the right to drill. As the area of interest is located near a major road, and is not in an area of tailings or other former mine infrastructure, WGM foresees no reason why such approval would be denied.

The project is ambitious and it requires favourable uranium market conditions, but it is prefaced on what WGM believes is an excellent opportunity to revitalize an area that has been long overlooked. If the outcome of the on-going drilling in the Teasdale Zone is positive, continued in-fill and step-out drilling down-dip will be required to up-grade the confidence level of the resources and to enlarge the resources.

24. SIGNATURE PAGE

This report titled "*Update Report on the Appia Energy Corp. Uranium-Rare Earth Property, Elliot Lake District, North-Central Ontario, Canada*" for Appia Energy Corp. and dated 30 July, 2013 was prepared and signed by the following authors:

Dated effective as of 30 July, 2013.

Al Workman, P.Geo.
Senior Geologist
Vice-President, Operations
Watts, Griffis and McOuat Ltd.

Kurt Breede, P.Eng
Senior Mineral Resources Engineer
Vice-President, Marketing
Watts, Griffis and McOuat Ltd.

John R. Goode, P.Eng.
Senior Associate Metallurgical Engineer
Watts, Griffis and McOuat Ltd.

25. REFERENCES

Agnew Lake Mines, 1980, **Agnew Lake Mines – Method of Reserve Calculation**, Appendix B, 18 January, 1980, 9 pages plus 8 figure pages.

Agnew Lake Mines, 1979, **Agnew Lake Mines – Reserve Summary, a block by block summary of reserves**, 31 December, 1979, 1 page.

Agnew Lake Mines, 1980, **Agnew Lake Mines – Method of Reserve Calculation**, unpublished letter, 3 pages.

Bain, G.W., 1956, **Discussion of Urano-Organic Ores**, in Economic Geology, Volume 52, Number 2, pp. 192-195.

Bain, G.W., 1960, **Patterns to Ores in Layered Rocks**, in Economic Geology, Volume 55, pp. 695-731.

Bernales, S., 2008, **2008 Fall-Winter Diamond Drilling Report, November 2007 to March, 2008, Elliot Lake Project, Elliot Lake District, Northwest Ontario, Canada – Townships of Buckles, Beange and Bouck (UTM Zone 17 – 46°23'N and 82°39'W) NTS 41/J7**, an independent consulting report for Appia Energy Corporation dated 20 April, 2008 and filed for assessment purposes with the Ministry of Northern Development and Mines, 59 pages plus 2 appendices.

Brotzen, O., 1958, **On the Occurrence of Uranium in Ancient Conglomerates**, in Economic Geology, Volume 53, pp. 489-491.

Canadian Nuclear Safety Commission, 2004, **Application to Revoke the Uranium Mine Decommissioning Licences for the Former Quirke, Panel and Stanleigh Mines and Incorporate the Associated Uranium Mine Tailings Sites into the Existing Waste Facility Operating Licence Issued to Rio Algom Limited**, a ruling by Marc A. Leblanc, Secretary, dated 3 June, 2004, 13 pages.

Carlson, G.J., 2006, **Induced Polarization / Resistivity Survey Consisting of Combined Dipole-Dipole and Schlumberger Arrays Over the Buckles Claim Block Near Elliot Lake, Ontario**, 9 pages plus figures.

China, 2009, **China's Nuclear Power Market Newsletter**, Issue 20, 16 November, 2009, 29 pages.

Ciuculescu, T, 2011, **Technical Report on the Eco Ridge Mine Project Uranium and Rare Earths Mineral Resources, Elliot Lake Area, Ontario, Canada**, an independent consulting report prepared for Pele Mountain Resources Inc. by Roscoe Postle Associates Inc., dated 5 April, 2011, 149 pages.

Cochrane, L.B. and Roscoe, W.E., 2007, **Technical Report on the Elliot Lake Project, Ontario, Canada**, an independent consulting report prepared for Pele Mountain Resources Inc. by Roscoe Postle Associates Inc., dated 15 January, 2007, 127 pages.

Cochrane, L.B., Hwozdyk, L.R. and Hayden, A.S., 2007, **Preliminary Assessment on the Elliot Lake Project, Ontario, Canada**, an independent consulting report prepared for Pele Mountain Resources Inc. by Roscoe Postle Associates Inc., dated 3 October, 2007, 293 pages.

Cox, J.J., Ciuculescu, T., Altman, K and Hwozdyk, L., 2012, **Technical Report on the eco Ridge Mine Project, Elliot Lake, Ontario, Canada**, an independent consultants report by Roscoe Postle Associates Inc., dated 20 June, 2012.

Davidson, C.F., 1957, **Occurrence of Uranium in Ancient Conglomerates**, in Economic Geology, Volume 52, pp. 668-693.

Denison Mines Limited, 1981, **Area 10 and 12 – Diamond Drill Hole Plan**, a 1"=500' map of the Denison Mine-Teasdale Lake area showing the location of underground workings, selected drill holes and higher grading U-bearing intervals in drill holes, sourced from the Ministry of Northern Development and Mines Assessment Files, Sault Ste. Marie, Denison File # G4, Drawing 12-0073.

Department of Energy (U.S.), 1995, **Uranium Industry Annual 1994**, an annual review by the US Department of Energy, document DOE/EIA-0478(94), July, 1995, 119 pages.

Dutkiewicz, A., 2001, **Hydrocarbon Fluid Inclusions and Pyrobitumen Nodules in the Uraniferous Matinenda Formation: Implications for Oil Migration During the Early Proterozoic**, in Faculdade de Ciencias do Porto, Departamento de Geologia Memoire No. 7, pp. 139-141.

Friedman, G.M., 1958, **On the Uranium-Thorium Ratio in the Blind River**, Ontario Uranium-Bearing Conglomerates, in Economic Geology, Volume 53, pp. 889-890.

Gray, E., 1982, **The Great Uranium Cartel**, 303 pages.

Health Canada, statements dated February 1993 and November, 1995, **Decommissioning of the Quirke/Panel & Stanrock/Denison Uranium Mine Tailings Management Areas in Elliot Lake, Ontario**, 4 pages.

Holgate, L., 2008, **Expanding Russian HEU Blend Down**, a paper delivered at the Institute of Nuclear Materials Management 49th Annual Meeting, Nashville, USA, 7 pages.

Ifill, R.O., Clark, A.H. and Cooper, W.C., 1989, **Mineralogical and Process Controls on the Oxidative Acid-Leaching of Radioactive Phases in Elliot Lake, Ontario, Uranium Ores, I -Uraninite, Uranothorite and Monazite**, Canadian Mineralogist Metallurgical Bulletin, Volume 82, Number 925 (May 1989), pp. 65-74.

International Atomic Energy Agency, 1987, **Uranium Deposits In Proterozoic Quartz-Pebble Conglomerates**, IAEA TECDOC 427, Report of the Working Group on Uranium Geology, Vienna, 453 pages.

International Atomic Energy Agency, 2002, **Non-Technical Factors Impacting on the Decision Making Processes in Environmental Remediation**, IAEA TECDOC 1279, April, 2002, 114 pages.

Lang, A.H., Griffith, J.W. and Steacy, H.R., 1962, **Canadian Deposits of Uranium and Thorium**, Geological Survey of Canada, Department of Mines and Technical Surveys Economic Geology Series No. 16 (2nd Edition), 324 pages and maps.

Lang, L. C. and Morrey, W. B., 1976, **Scientific Blasting for In-Situ Leaching Proves Successful at Agnew Lake Mines**, Engineering and Mining Journal, January 1976, pp. 100-103.

Lonn, G., 1966, **The Mine Finders**, 244 pages.

MacEachern, A.D., 2009, **Elliot Lake Uranium Exploration Project – Banana Lake Area**, an independent consulting report for Appia Energy Corp. dated 22 July, 2009 and filed for assessment purposes with the Ministry of Northern Development and Mines, 138 pages.

MacGregor, R.A., 1975, **Possible Recovery of Uranium from Solutions of Low Concentration**, an independent consulting report for Imperial Oil Limited dated 2 June, 1975, 10 pages.

MacLeod Dixon, 2008, **Uranium Mining in Kazakhstan and Pervasive State Participation: A Canadian Comparative Perspective**, a review by Crae Garrett and Zhibek Aidymbekova, 3 pages.

Marchbank, A., 1985 (?), **An Update on Bacterial Leaching of Uranium at Denison Mines**, an internal report by a Denison Mines taskforce to investigate bacterial leaching at the Denison Mine, Elliot Lake, Ontario, 9 pages.

Neff, T.L., 2004, **Insights into the Future: Uranium Prices and Price Information 1947-2004**, a presentation to the World Nuclear Association, September 2004, 19 pages.

Phelan, L. G., 1955, **Summary Report on New Turbois Mines Limited**, Agnew Lake Uranium Area Property, MNDM Assessment Files, unpublished report, 8 pages.

Pool, T.C., 1997, **Primary and Secondary Uranium Supplies: Different Cost Structures, Different Goals**, The Uranium Institute, Twenty-Second Annual Symposium, 3-5 September 1997: London, England, 7 pages.

Pretorius, D.A., 1981, **Gold and Uranium in Quartz-Pebble Conglomerates**, in Skinner, B.J., Economic Geology Seventy-Fifth Anniversary Volume, pp.117-138.

Rio Algom, 1979, **Gemico Properties, Elliot Lake Area**, a 1"=1 mile map of the Elliot Lake Basin showing exploration/mine property outlines, selected diamond drill holes and Rio Algom's estimated uranium resource for selected blocks, sourced from the Ministry of Northern Development and Mines Assessment Files, Sault Ste. Marie, Rio Algom File Reference – Drawing # 791.

Robertson, D.S., 1960, **On the Blind River Uranium Ores and Their Origin**, in Economic Geology, Volume 55, pp. 659-694.

Robertson, J.A., 1968, **The Uranium and Thorium Deposits of Northern Ontario**, Ontario Department of Mines, Mineral Resources Circular No. 9, 106 pages.

Robertson, J.A., 1976, **The Blind River Uranium Deposits: The Ores and Their Setting**, Ontario Division of Mines Miscellaneous Paper 65, 45 pages.

Robertson, J.A., 1981, **The Uranium Deposits of Ontario – Their Distribution and Classification**, Ministry of Natural Resources, Ontario Geological Survey Miscellaneous Paper 86, 37 pages.

Robinson, A.G. and Spooner, E.T.C., 1984, **Post-Depositional Modification of Uraninite-Bearing Quartz-Pebble Conglomerates from the Quirke Ore Zone, Elliot Lake, Ontario**, in Economic Geology; April 1984, Volume 79, Number 2, pp. 297-321.

Roscoe, S.M., 1959, **On the Uranium-Thorium Ratio in Conglomerates and Associated Rocks near Blind River, Ontario**, in Economic Geology, Volume 54, pp. 511-512.

Roscoe, S.M., 1996, **Paleoplacer Uranium, Gold**, in Geology of Canadian Mineral Deposit Types, editors O.R. Eckstrand, W.D. Sinclair and R.I. Thorpe, Geological Survey of Canada, Geology of Canada No. 8, pp. 10-23.

Ruzicka, V., 1971, **Geological Comparison Between East European and Canadian Uranium Deposits**, Geological Survey of Canada, Department of Energy, Mines and Resources, Paper 70-48, 196 pages and maps.

SGS Canada Inc., 2011, **An Investigation by High Definition Mineralogy into the Mineralogical Characteristics of Five Drill Core Samples from the Appia Energy (Project), Ontario, Canada**, prepared for Appia Energy Corporation, Project 13173-001, MI5049, 1 June, 2011, 98 pages.

SGS Canada Inc., 2013a, **An Investigation Into the Recovery of Uranium and Rare Earth Elements from the Teasdale Property**, prepared for Appia Energy Corp., Project 13849-001, Final Report, 19 July, 2013, 100 pages.

SGS Canada Inc., 2013b, **An Investigation Into Beneficiation Testwork to Assess the Recoverability of Uranium and Rare Earth Elements from the Teasdale Property**, prepared for Appia Energy Corp., Project 13849-002, Final Report, 5 June, 2013, 50 pages plus tables.

Sprague, D., undated, Report on Artison Gold Inc. Ground in Buckles Township, Ontario, an independent consultants report on the uranium resources in the Teasdale Lake Zone, 5 pages plus attachments.

Stephens, M., 1980, **Three Mile Island**, 245 pages.

Stevenson, J., Mancuso, J., Frizado, J., Truskoski, P. and Kneller, W., 1990, **Solid Pyrobitumen In Veins, Panel Mine, Elliot Lake District. Ontario**, in Canadian Mineralogist, Vol. 28, pp. 161-169.

Thomson, J.E., 1960, **Uranium and Thorium Deposits at the Base of the Huronian System in the District of Sudbury**, Ontario Department of Mines, Geological Report No. 1, 40 pages, 1 map.

Vance, R., 1999, **Uranium, Canadian Minerals Yearbook: 1998**, Natural Resources Canada, pp. 59-1 to 59-14.

Vance, R., 2004, **Uranium, Canadian Minerals Yearbook: 2003**, Natural Resources Canada, pp. 58-1 to 58-13.

Whillans, R.T., 1994 **Uranium, Canadian Minerals Yearbook: 1993**, Natural Resources Canada, pp. 59-1 to 59-17

Whillans, R.T., 1997, **Uranium, Canadian Minerals Yearbook: 1996**, Natural Resources Canada, pp. 59-1 to 59-16

Wilton, C. K. 1982, **Geology of the Agnew Lake Mine**, draft paper for CIMM, 16 pages.

Workman, A. and Vasak, P., 2008, **A Technical Review of the Appia Energy Corp. Elliot Lake Uranium Properties, Elliot Lake District, North-Western Ontario**, Canada, an independent consultants report dated 11 September, 2008, 194 pages.

Workman, A. and Breede, K., 2011, **A Technical Review of the Appia Energy Corp. Rare Earth Metal - Uranium Property, Elliot Lake District, North-Central Ontario, Canada**, an independent consultants report dated 18 July, 2011, 268 pages.

OTHER MATERIALS AVAILABLE FOR REVIEW

Card, K. D. 1965, **Hyman and Drury Townships**, Geology Report. No. 34, Ontario Department of Mines, 38 pages.

Carrington, J. K. and Wilton, C. K., 1969, **Economic Geology of Agnew Lake Mines Ltd.**, paper presentation, CIMM, April 1969, 21 pages.

Coode, A., 1977, **Classification of Uraniferous Blocks Used in Geologic Reserve Calculations**, an internal Agnew Lake Mines memo from A. Coode to J. Choi, 11 January, 1977, 1 page.

Goode, J.R., 2012, **Canadian Rare Earth and Thorium Recovery Operations - The First 30 Years**, a paper delivered at the Canadian Institute of Mining and Metallurgy Conference, Niagara Falls, September-October, 2012, 17 pages.

Goode, J.R., Drivas, T. and Workman, A., 2013, **Development of a High Recovery Process Flowsheet for Appia's Elliot Lake Uranium - Rare Earth Ore**, an independent paper published in Materials Science & Technology 2013 Conference Proceeding**s**, 14 pages.

Gulliver Archives, 1992, **Denison Mines Ltd., in The Gulliver File – Mines, People and Land: A Global Background**, internet source at http://www.sea-us.org.au/gulliver/denison.html, 8 pages.

De Voto, R.H., **Lecture Notes and References, Uranium Geology and Exploration**, a short course offered by the Colorado School of Mines, Golden Colorado, March, 1978, 396 pages.

Frondel, C., 1958, **Systematic Mineralogy of Uranium and Thorium**, Geological Survey Bulletin 1064, United States Atomic Energy Commission, 400 pages.

International Atomic Energy Agency (IAEA), 1974, **Formation of Uranium Ore Deposits**, Proceedings of a Symposium – Athens, 6-10 May, 1974, 748 pages.

U.S. Energy Information Administration (EIA), 2013, **2012 Uranium Marketing Annual Report**, 31 pages

U.S. Energy Information Administration (EIA), 2013, **2012 Domestic Uranium Production Report**, 21 pages.

CERTIFICATE

To Accompany the Report Entitled
"Update Report on the Appia Energy Corp. Uranium-Rare Earth Property,
Elliot Lake District, North-Central Ontario, Canada"
for Appia Energy Corp.
dated 30 July, 2013

I, Albert W. Workman, do hereby certify that:

1. I reside at 2285 Lakeshore Blvd. West, Suite 2413, Toronto, Ontario, Canada, M8V 3X9.

2. I graduated from Brock University in St. Catharines, Ontario, Canada, in 1975 with an Honours B.Sc. in Geological Sciences and I have practiced my profession continuously since then for a total of 36 years in Canada and internationally. I worked exclusively in uranium exploration during 1975 in the Collins Bay area Saskatchewan, and again exclusively in uranium on projects throughout Canada during 1977-1982. Since 1998, I have provided QP services on a variety of uranium projects in Saskatchewan, Ontario and Quebec as well as the United States, Madagascar, Tanzania and Guyana.

3. I am a Practicing Member of the Association of Professional Geoscientists of Ontario (Member #0170). I am also a Fellow of the Australian Institute of Mining and Metallurgy (AusIMM) as well as a Member of the Society of Economic Geologists (SEG), the Canadian Institute of Mining, Metallurgy and Petroleum (CIM), and the Prospectors and Developers Association of Canada (PDAC).

4. I am a Senior Geologist and Vice-President with Watts, Griffis and McOuat Limited, a firm of consulting geologists and engineers, which has been authorized to practice professional engineering by the Professional Engineers Ontario since 1969.

5. 1 have read the definition of "Qualified Person" in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, past relevant work experience and affiliation with a professional association, that I fulfill the requirements to be a Qualified Person for the purposes of NI 43-101.

6. I visited the Elliot Lake properties of Appia Energy Corp. on 15-16 May, 2007, during which I reviewed data and visited former exploration drilling sites. I visited the Ministry of Northern Development and Mines as well as the Ministry of the Environment, and met with local government officials concerning policy issues. I revisited the project sites during June, 2008 and again during July 2013 during which I confirmed and audited the most recent diamond drilling program, examined drill core and collected independent samples for the purpose of confirming Appia's reported analytical data.

7. I am responsible for all sections of this report, and jointly responsible for the Mineral Resource estimates in Section 17 with co-author Kurt Breede. I am also responsible for co-ordinating the in-put of other QPs who were used as sources of information for this technical report relating to the Appia Energy Corp. property in the Elliot Lake area of north-central Ontario, Canada.

8. Since 2008, I have acted in the capacity of Qualified Person in respect to the exploration property discussed in this report and authored or co-authored several previous reports on this property.

9. I am independent of the issuer applying all of the tests in Section 1.4 of the Canadian Securities Regulation NI 43-101.

a. Neither I, nor any affiliated entity of mine, is at present under or expects to be under an agreement, arrangement or understanding to become, an insider, associate, affiliated entity or employee of Appia Energy Corp., or any associated or affiliated entities.

b. Neither I nor any affiliated entity of mine, own, directly or indirectly, nor expect to receive any interest in the properties or securities of Appia Energy Corp., or any associated or affiliated companies.

c. Neither I nor any affiliated entity of mine, have earned the majority of our income during the preceding three years from Appia Energy Corp., or any associated or affiliated companies.

10. I have read NI 43-101 and Form 43-101F1 and have prepared the technical report in compliance with NI 43-101 and Form 43-101F1; and I have prepared the report in conformity with generally accepted Canadian mining industry practices.

11. As of the effective date of this technical report, and to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

PROFESSIONAL GEOSCIENTIST
A. W. WORKMAN
PRACTISING MEMBER
0170
ONTARIO

Al Workman, P.Geo.
30 July, 2013

CERTIFICATE

To Accompany the Report Entitled
"Update Report on the Appia Energy Corp. Uranium-Rare Earth Property,
Elliot Lake District, North-Central Ontario, Canada"
for Appia Energy Corp.
dated 30 July, 2013

I, Kurt Breede, do hereby certify that:

2. I reside at 76 Woodrow Avenue, Toronto, Ontario, M4C 1G7.

3. I graduated from the University of Toronto, Toronto, Ontario in 1996 with a B.A.Sc. in Geological and Mineral Engineering, and have been practicing my profession since 1996.

4. I am a Professional Engineer licensed by Professional Engineers Ontario (Registration Number 90501859) and The Association of Professional Engineers and Geoscientists of Saskatchewan (Registration Number 17014).

5. I am a Vice-President, Marketing and Technical Services with Watts, Griffis and McOuat Limited, a firm of consulting geologists and engineers, which has been authorized to practice professional engineering by Professional Engineers Ontario since 1969, and professional geoscience by the Association of Professional Geoscientists of Ontario.

6. I have read the definition of "Qualified Person" in National Instrument 43-101 ("NI 43-101") and certify that I am a Qualified Person for the purposes of NI 43-101 by virtue of my education and having experience with regard to a variety of mineral deposit types, with Mineral Reserve and Mineral Resource estimation parameters and procedures and with those involved in the preparation of technical studies.

7. I did not visit the Property.

8. I am jointly responsible for the current uranium-rare earth Mineral Resource estimate for the Teasdale Zone and the Mineral Resource estimate for the Banana Lake Zone in Section 17 with co-author Al Workman. I have had no involvement with the auditing of the Sprague historical estimate, or the 2008 uranium Mineral Resource estimate.

9. I am independent of the issuer applying all of the tests in Section 1.4 of the Canadian Securities Regulation NI 43-101.

    a. Neither I, nor any affiliated entity of mine, is at present under or expects to be under an agreement, arrangement or understanding to become, an insider, associate, affiliated entity or employee of Appia Energy Corp., or any associated or affiliated entities.

    b. Neither I nor any affiliated entity of mine, own, directly or indirectly, nor expect to receive any interest in the properties or securities of Appia Energy Corp., or any associated or affiliated companies.

    c. Neither I nor any affiliated entity of mine, have earned the majority of our income during the preceding three years from Appia Energy Corp., or any associated or affiliated companies.

10. I have read NI 43-101 and Form 43-101F1 and have prepared the technical report in compliance with NI 43-101 and Form 43-101F1; and I have prepared the report in conformity with generally accepted Canadian mining industry practices.

11. As of the effective date of this technical report, and to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Kurt Breede, P.Eng.
30 July, 2013

CERTIFICATE

To Accompany the Report Entitled
"Update Report on the Appia Energy Corp. Uranium-Rare Earth Property,
Elliot Lake District, North-Central Ontario, Canada"
for Appia Energy Corp.
dated 30 July, 2013

I, John R. Goode, do hereby certify that:

1. I am a metallurgist with J.R. Goode and Associates of Suite 1010, 65 Spring Garden Avenue, Ontario, Canada, M2N 6H9.
2. I am a graduate of the Royal School of Mines, London University, UK, in 1963 with a Bachelor of Science (Chemical Engineering in Metallurgy) degree.
3. I am registered as a Professional Engineer in the Province of Ontario (Reg.# 16561011).
4. I am a Senior Associate Metallurgical Engineer of Watts, Griffis and McOuat Limited, a firm of consulting geologists and engineers, which has been authorized to practice professional engineering by Professional Engineers Ontario since 1969, and professional geoscience by the Association of Professional Geoscientists of Ontario. I have worked as a Metallurgical Engineer for a total of 50 years since my graduation. My relevant experience for the purpose of the Technical Report is:
   - Worked in a plant that recovered rare earths and thorium as by-products of a Rio Algom uranium plant in the Elliot Lake mining camp
   - Researched, designed and commissioned a new rare earth recovery plant attached to the Denison uranium plant in the Elliot Lake camp
   - Performed several rare earth, niobium, and tantalum design and cost studies including those for Molycorp, Mountain Pass; Niocan, Oka Nb-Ta-rare earth project; AMR on-site analysis of Chinese separation plants; and Pacific Metal tantalum-niobium plant, Guangdong, China
   - Completed several dozen due diligence, pre-feasibility, and feasibility, design, and commissioning projects involving metallurgy of gold, uranium, rare metals, and base metals around the world.
   - Vice-President, Mining and Metallurgy, Kilborn Ltd., an engineering company now part of SNC Lavalin.
5. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
6. I did not visit the Appia Property.
7. I am responsible for the section of the report dealing with the Mineral Processing and Metallurgical Testing program carried out by Appia, namely Section 16. I have had no involvement with the other aspects of the project other than providing guidance with the assembly of the bulk sample for testing purposes and providing liaison and guidance to SGS Canada staff involved in the testing program.
8. I am independent of the issuer applying all of the tests in Section 1.4 of the Canadian Securities Regulation NI 43-101.
   a. Neither I, nor any affiliated entity of mine, is at present under or expects to be under an agreement, arrangement or understanding to become, an insider, associate, affiliated entity or employee of Appia Energy Corp., or any associated or affiliated entities.

    b. Neither I nor any affiliated entity of mine, own, directly or indirectly, nor expect to receive any interest in the properties or securities of Appia Energy Corp., or any associated or affiliated companies.

    c. Neither I nor any affiliated entity of mine, have earned the majority of our income during the preceding three years from Appia Energy Corp., or any associated or affiliated companies.

9. I have had no prior involvement with the property that is the subject of this Technical Report.
10. I have read NI 43-101 and Form 43-101F1 and have prepared that part of the technical report for which I am responsible in compliance with NI 43-101 and Form 43-101F1; and I have prepared the report in conformity with generally accepted Canadian mining industry practices.
11. As of the effective date of this technical report, and to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Dated this 30th day of July, 2013

JR Goode

John R. Goode, P.Eng.

ATTACHMENTS

Attachment 1:	**An Investigation Into Beneficiation Testwork to Assess the Recoverability of Uranium and Rare Earth Elements from the Teasdale Property**
Attachment 2:	**An Investigation Into the Recovery of Uranium and Rare Earth Elements from the Teasdale Property**

An Investigation into

BENEFICIATION TESTWORK TO ASSESS THE RECOVERABILITY OF URANIUM AND RARE EARTH ELEMENTS FROM THE TEASDALE PROPERTY

prepared for

APPIA ENERGY CORP.

Project 13849-002 Final Report
June 5, 2013

NOTE:
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Executive Summary

A scoping level beneficiation study was conducted on samples from the Teasdale property located in the Elliot Lake area, Ontario, at the request of Watt, Griffis and McOuat Limited on behalf of Appia Energy Corp. The purpose of the study was to identify a processing route that could recover the majority of the uranium, thorium and rare earth bearing minerals while rejecting a significant amount of the major gangue mineral quartz. The beneficiation study consisted of head sample chemical analysis, mineralogical characterization, gravity separation, magnetic separation and flotation testing.

The summary head assays for the four test composites are shown in Table 1.

Table 1: Head Assay Summary

Sample ID	UR Comp	IQ Comp	LR Comp	Master Comp
S, %	1.46	1.04	3.32	1.92
U, g/t	218	120	520	270
Th, g/t	445	218	350	354
TREE, g/t	2285	1153	1548	1702

A QEMSCAN^{TM} mineralogical examination indicated that the major phases present in these composites were quartz (63.8-67.4%), K-feldspars (12.2-16.8%), micas (9.6-12.6%), and pyrite (2.6-10%). Electron Microprobe Analysis (EMPA) revealed that the main Rare Earth Element (REE)-bearing minerals were monazite and allanite, and the U-Th minerals included uranothorite-thorite [$(Th,U)SiO_4$], altered uraninite (UO_2), and coffinite [$U(SiO_4)_{1-x}(OH)_{4x}$] . The monazite is generally over 60% free and liberated but the allanite has more complex particles at 100% passing 100 mesh (150 µm). The U-Th minerals are generally poorly liberated but they are moderately to well exposed.

A Bond ball mill grindablity test was performed at 100 mesh (150 µm) and showed that the Master Composite was moderately hard with a metric work index of 16.2 kWh/t.

Metallurgical testwork included gravity separation, magnetic separation and flotation. Gravity separation using a Mozley Shaking Table on a sample that was 80% passing 208 µm demonstrated U, Th and REE upgrading. The combined Mozely gravity concentrates recovered 52% U, 61% Th and 59% Total Rare Earth Element (TREE) in 7% mass. A significant amount of U, Th and TREE losses was found to deport in the -38 µm fraction of the gravity tailings. The combined Mozely gravity concentrate and -38 µm of the gravity tailings recovered 75% U, 81% Th and 79% TREE in 26% mass. A finer grind resulted in similar results.

Gravity separation using the Wilfley Table to evaluate the amenability of the Gekko IPJ was completed. A specific procedure, Gekko 296A, was provided by the client and followed. The results demonstrated possible gangue rejection at a coarse size. The combined Wilfley concentrate and the slime recovered 77% U, 84% Th and 78% TREE in 36% mass. The Gekko IPJ was not recommended as a pre-concentration method because the client reviewed the results and determined the recovery of U and REE was below their target.

Wet High Intensity Magnetic Separation (WHIMS) at 2,000 to 20,000 Gauss on a sample that was 80% passing 208 µm was performed. The cumulative magnetics recovered 61% U, 74% Th and 78% TREE in 9% mass. Further scavenging of the non-magnetics by flotation would likely improve recovery. A finer grind did not improve the WHIMS results.

Flotation flowsheet development was carried out on the Master composite. Variability testing on the Lower Reef (LR), Upper Reef (UR) and Intermediate Quartzite (IR) composites were completed using the conditions established with the Master composite. The best results on the Master composite were achieved in F6 using collector LR19. The F6 rougher concentrate recovered 79% U, 92% Th and 92% TREE in 20% mass. Results of variability flotation tests (F9, F16 and F17) and F6 are shown in Table 2.

Table 2: Variability Flotation Comparison

Sample ID	Test No.	Wt %	Grades: U, Th, REE-g/t; S %						% Distribution					
			U	Th	S	LREE	HREE	TREE	U	Th	S	LREE	HREE	TREE
LR Comp	F9	32.9	1270	1030	11.0	4330	382	4712	85.5	93.3	98.5	94.5	90.4	94.1
UR Comp	F16	33.3	493	1136	4.45	6006	409	6415	80.4	91.7	98.7	93.6	78.1	92.4
IQ Comp	F17	14.1	364	1053	7.41	6636	336	6972	48.7	72.4	96.8	86.0	48.3	82.9
Master Comp	F6	20.4	885	1471	8.99	7314	516	7831	78.6	91.9	98.5	93.6	74.7	92.1

Pyrite removal prior to U-Th-TREE flotation as well as on a leached residue was completed. Greater than 90% S was removed in either approach.

Recommendations for future work include:

- Investigate the effects of desliming;
- Investigate collector type – FA2 in place of LR19;
- A combination of gravity or magnetic separation and flotation.

Introduction

A proposal for mineralogical, beneficiation, and hydrometallurgical testwork on quartz pebble conglomerate samples from the Elliot Lake area was issued on October 16th, 2012 and approved the next day. The scope of beneficiation testwork consisted of gravity separation, magnetic separation and flotation. The testwork was initiated in December, 2012 and was carried out over the following three months with the objective of assessing the recoverability of uranium, thorium and rare earth elements. This report presents the beneficiation testwork results.

The test program was conducted under the direction of Mr. John Goode on behalf of Watts, Griffis and McOuat Limited and Appia Energy Corp. Results were provided to him for review and guidance as they became available.

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Testwork Summary

1. Sample Receipt and Preparation

A total of five shipments were received at the SGS Lakefield site from October to November 2012. The first and second shipments were assay rejects from Actlabs and the client decided not to use them for testwork after reviewing the inventory list. The 3rd to 5th shipments were ¼ drill core samples containing 30.8 kg, 55.3 kg and 78.4 kg of material with assigned internal receipt number 0059-Nov12, 0081-Nov12 and 0158-Nov12, respectively. Table 3 summarises the information of received samples.

Table 3: Inventory of As-Received Sample

Shipment No.	Samples	Mass kg	Assays, lb/st			
			LREO	HREO	U_3O_8	ThO_2
1st	1st Zone	79.4				
	Upper reef	4.3				
	Intermediate quartzite	3.4				
	Lower reef	0				
	LR: 1382316 -1382344	0				
	NB: 1382394 - 1382407	0				
	Miscellaneous	0.5				
	2nd Zone	0				
2nd	Upper reef	9.5				
	Intermediate quartzite	6.0				
	Lower reef	3.4				
3rd - 5th	Upper reef	83.7	4.945	0.368	0.488	0.927
	Intermediate quartzite	44.1	2.186	0.152	0.257	0.431
	Lower reef	44.1	3.364	0.314	0.969	0.717
	Total	171.9	3.796	0.297	0.562	0.741

The sample preparation flowsheet is presented in Figure 1. Three composites representing Upper reef (UR), Intermediate quartzite (IQ) and Lower reef (LR) were separately sorted out from the 3rd to 5th shipments. Each Composite was combined, homogenized and stage-crushed to 100% passing 6 mesh (3.35 mm) and 500 g was riffled out followed by further splitting for head assay, mineralogical examination and for bio-leaching amenability test. The Master Composite (MC) was composited from 31.4 kg UR, 16.5 kg IQ and 16.5 kg LR. It was combined, homogenized and rotary split into 2 kg test charges. The remaining 51.8 kg UR, 27.1 kg IQ and 27.1 kg LR Composites were reserved separately for variability testwork.

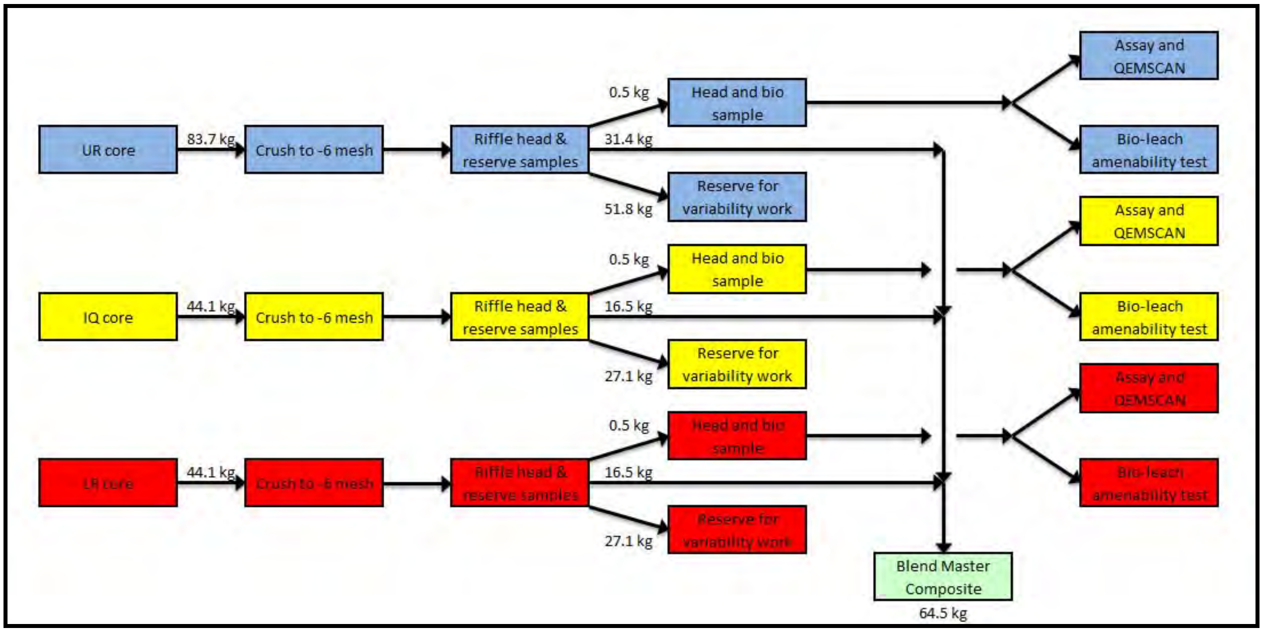


Figure 1: Sample Preparation Flowsheet

2. Sample Characterization

2.1. Head Assays

Samples representative of the four composites (UR, IQ, LR and MC) were submitted for detailed chemical analysis which included: whole rock analysis (SiO_2, Al_2O_3, Fe_2O_3, MgO, CaO, Na_2O, K_2O, TiO_2, P_2O_5, MnO, Cr_2O_3, V_2O_5 and loss on ignition (LOI)), REE-ICP Scan (Ce, Dy, Er, Eu, Gd, Ho, La, Lu, Nd, Pr, Sm, Tb, Th, Tm, Sc, U, Y, and Yb) S, $S^=$ and Au. The analytical results are shown in Table 4.

Table 4: Head Assays of Composites

Sample ID	UR Comp	IQ Comp	LR Comp	Master Comp
Whole Rock Analysis, %				
SiO_2	84.2	84.1	79.7	83.4
Al_2O_3	6.27	7.25	5.77	6.43
Fe_2O_3	2.28	1.68	4.61	2.75
MgO	0.12	0.12	0.07	0.10
CaO	0.32	0.19	0.58	0.37
Na_2O	0.10	0.06	0.10	0.11
K_2O	3.43	3.59	2.80	3.30
TiO_2	0.65	0.30	0.38	0.49
P_2O_5	0.14	0.07	0.09	0.11
MnO	< 0.01	< 0.01	< 0.01	0.01
Cr_2O_3	0.04	0.02	0.02	0.02
V_2O_5	< 0.01	< 0.01	< 0.01	< 0.01
LOI	1.72	1.53	2.98	1.95
Sum	99.2	98.9	97.1	99.0
REE - ICP Scan, g/t				
La	536	259	352	407
Ce	1000	488	681	767
Pr	105	51.7	73.5	81.0
Nd	328	162	235	254
Sm	54.3	26.2	41.9	42.6
Eu	2.90	1.50	2.70	1.90
Gd	34.0	15.7	28.5	26.1
Tb	4.70	1.80	4.20	3.30
Dy	20.3	8.60	19.2	16.5
Ho	3.70	1.40	3.40	2.60
Er	8.80	3.30	8.20	6.70
Tm	2.00	0.40	1.30	0.80
Yb	7.10	2.50	5.90	5.20
Lu	2.30	< 0.50	1.00	0.70
Y	76.0	31.0	65.0	62.0
Sc	< 25	< 25	< 25	< 25
U	218	120	520	270
Th	445	218	350	354
Element assays S, $S^=$, %; Au, g/t				
S	1.46	1.04	3.32	1.92
$S^=$	1.38	0.92	2.94	1.58
Au	< 0.02	< 0.02	0.03	< 0.02

2.2. Mineralogical Examination

The head samples of the three composites (UR Comp, IQ Comp and LR Comp) were submitted to the SGS Advanced Mineralogy Facility for mineralogical characterization using a combination of QEMSCAN^{TM} Particle Mineral Analysis (PMA) method, X-ray Diffraction (XRD), Electron Microprobe Analysis (EMPA) and Scanning Electron Microscopy (SEM) approaches. The objective was to determine the overall mineral assemblage with emphasis on the REE and U-Th-bearing minerals.

The head samples of three composites were stage-crushed to 100% passing 100 mesh (150 µm) and screened into three size fractions (+75, -75/+38 and -38 µm) for mineralogical study. The detailed QEMSCAN results are issued in a separate SGS report (CALR-13849-002-MI5007-Dec12) and only the main findings are highlighted here.

The modal analyses obtained for each composite are presented in Table 5 and graphically shown in Figure 2. All three samples (LR Comp, UR Comp and IQ Comp) are composed of quartz (63.8-67.4%), K-feldspars (12.2-16.8%), micas (9.6-12.6%), pyrite (2.6-10%), along with minor to trace amounts of rutile, carbonates, other oxides and other minerals. Monazite, bastnaesite, allanite and U-Th phases are present in trace amounts.

Table 5: Bulk Modals Analyses

Survey		**Appia Energy Corp.**																				
Project		**CALR-13849-002 / MI5056-JUL12**																				
Sample		**LR Comp**							**UR Comp**							**IQ Comp**						
Fraction		**Combined**	**+75um**		**-75/+38um**		**-38um**		**Combined**	**+75um**		**-75/+38um**		**-38um**		**Combined**	**+75um**		**-75/+38um**		**-38um**	
Mass Size Distribution (%)			38.9		16.5		44.6			39.3		18.5		42.2			40.3		18.2		41.6	
Calculated ESD Particle Size		19	64		31		11		19	67		31		11		19	64		32		11	
		Sample	**Sample**	**Fraction**	**Sample**	**Fraction**	**Sample**	**Fraction**	**Sample**	**Sample**	**Fraction**	**Sample**	**Fraction**	**Sample**	**Fraction**	**Sample**	**Sample**	**Fraction**	**Sample**	**Fraction**	**Sample**	**Fraction**
Mineral Mass (%)	Monazite	0.3	0.2	0.4	0.1	0.3	0.1	0.3	0.5	0.1	0.4	0.1	0.4	0.2	0.6	0.2	0.1	0.2	0.0	0.2	0.1	0.3
	Bastnaesite	0.1	0.0	0.0	0.0	0.1	0.0	0.1	0.1	0.0	0.1	0.0	0.0	0.0	0.1	0.0	0.0	0.0	0.0	0.0	0.0	0.1
	Synchysite	0.0	0.0	0.0	0.0	0.1	0.0	0.0	0.0	0.0	0.1	0.0	0.1	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
	Allanite	0.1	0.0	0.1	0.0	0.1	0.1	0.2	0.1	0.1	0.1	0.0	0.2	0.0	0.1	0.0	0.0	0.0	0.0	0.0	0.0	0.0
	U-Th Minerals	0.1	0.0	0.1	0.0	0.0	0.1	0.1	0.1	0.0	0.1	0.0	0.1	0.0	0.0	0.0	0.0	0.1	0.0	0.0	0.0	0.0
	Other REE	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
	Zircon	0.1	0.0	0.1	0.0	0.1	0.0	0.0	0.1	0.1	0.1	0.0	0.1	0.0	0.1	0.0	0.0	0.0	0.0	0.0	0.0	0.0
	Apatite	0.1	0.0	0.1	0.0	0.1	0.0	0.1	0.2	0.0	0.1	0.0	0.2	0.1	0.3	0.1	0.0	0.1	0.0	0.1	0.0	0.0
	Quartz	63.8	27.8	71.5	11.6	70.3	24.3	54.5	65.4	28.2	71.7	13.2	71.3	24.1	57.0	67.4	30.9	76.7	13.5	73.9	23.1	55.6
	K-Feldspar	12.2	4.2	10.8	2.0	11.8	6.1	13.7	16.8	6.4	16.3	3.3	18.0	7.0	16.7	15.7	5.9	14.6	3.1	17.0	6.7	16.2
	Micas	10.0	1.4	3.6	0.8	4.6	7.8	17.6	9.6	1.2	3.2	0.7	3.6	7.7	18.2	12.6	1.7	4.2	0.9	5.1	9.9	23.9
	Clays	0.0	0.0	0.0	0.0	0.1	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
	Other Silicates	1.0	0.3	0.8	0.2	1.0	0.5	1.1	0.2	0.1	0.2	0.0	0.2	0.2	0.4	0.2	0.0	0.1	0.0	0.1	0.1	0.3
	Carbonates	0.9	0.2	0.4	0.2	1.0	0.6	1.3	0.4	0.1	0.2	0.1	0.3	0.3	0.7	0.3	0.0	0.1	0.0	0.3	0.2	0.5
	Pyrite	10.0	4.2	10.9	1.5	9.2	4.2	9.4	4.1	2.2	5.6	0.6	3.5	1.2	2.8	2.6	1.3	3.2	0.4	2.4	0.9	2.1
	Rutile	0.7	0.3	0.7	0.1	0.7	0.3	0.7	1.5	0.6	1.6	0.2	1.1	0.7	1.7	0.5	0.2	0.6	0.1	0.4	0.2	0.5
	Other Oxides	0.3	0.1	0.2	0.0	0.2	0.2	0.5	0.2	0.0	0.0	0.1	0.3	0.1	0.3	0.1	0.0	0.0	0.0	0.1	0.1	0.2
	Other	0.2	0.0	0.1	0.1	0.3	0.1	0.3	0.6	0.1	0.2	0.1	0.5	0.4	1.0	0.1	0.0	0.1	0.0	0.1	0.1	0.2
	Total	100.0	38.9	100.0	16.5	100.0	44.6	100.0	100.0	39.3	100.0	18.5	100.0	42.2	100.0	100.0	40.3	100.0	18.2	100.0	41.6	100.0

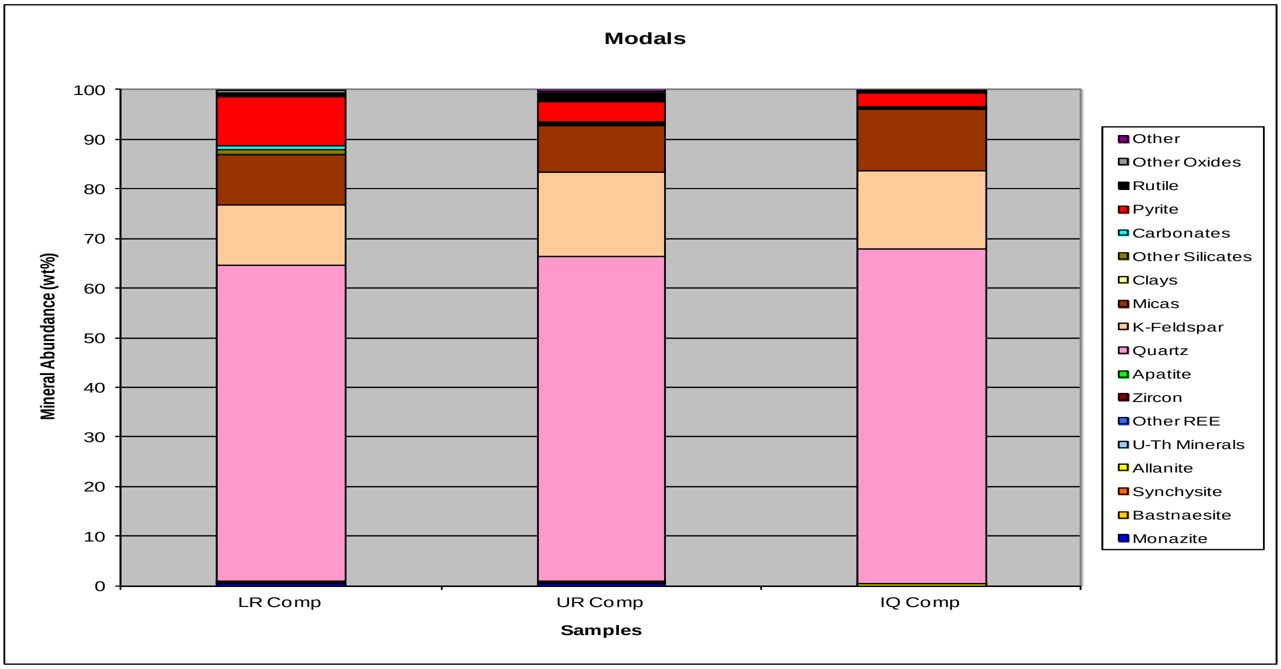


Figure 2: Mineral Abundance of Each Composite

Liberation and association data for U-Th minerals, Monazite, Allanite and Pyrite are present in Figures 3 to 6.

Major observations are as follows:

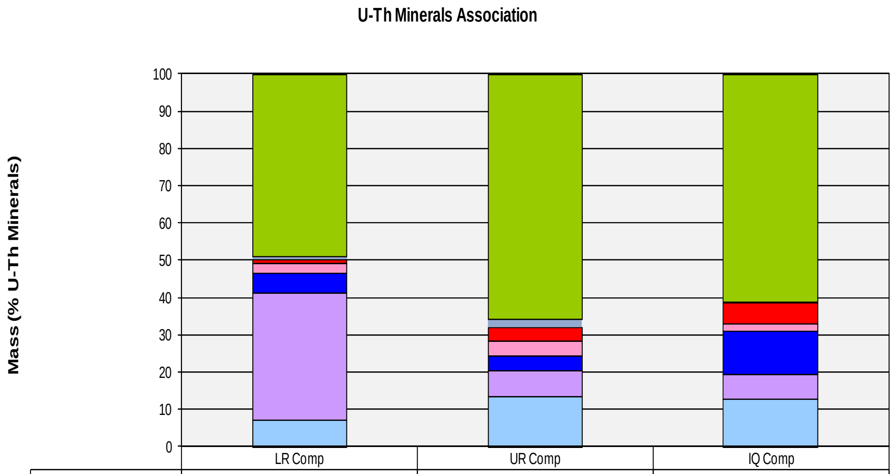
- Free and liberated U-Th minerals account for 41%, 21% and 19% for the LR, UR and IQ, respectively (Figure 3). Most of the U-Th minerals occur as complex particles (terniary and quaterniary middlings) (49%, 66% and 61%), minor middlings with monazite (5%, 4% and 12%), quartz/feldspars (<4%). Liberation generally increases significantly from the +75 to -38 um fraction.

- Mineral exposure data show that LR and UR have similar values with >30% exposed particles at 68% and 64%, respectively. U-Th minerals are better exposed at 86% in the IQ sample. Thus, the potential for leaching the U from these phases is better for IQ. Thus, although the U-Th minerals are poorly liberated, they are moderately to well exposed.

- Free (a mineral with ≥ 95% surface exposure) and liberated (a mineral with ≥ 80% but <95% surface exposure) monazite account for 59%, 68% and 62% for the LR, UR and IQ, respectively (Figure 4). The remainder occurs as complex particles (29%, 22% and 28%) and quartz/feldspars (<6%).

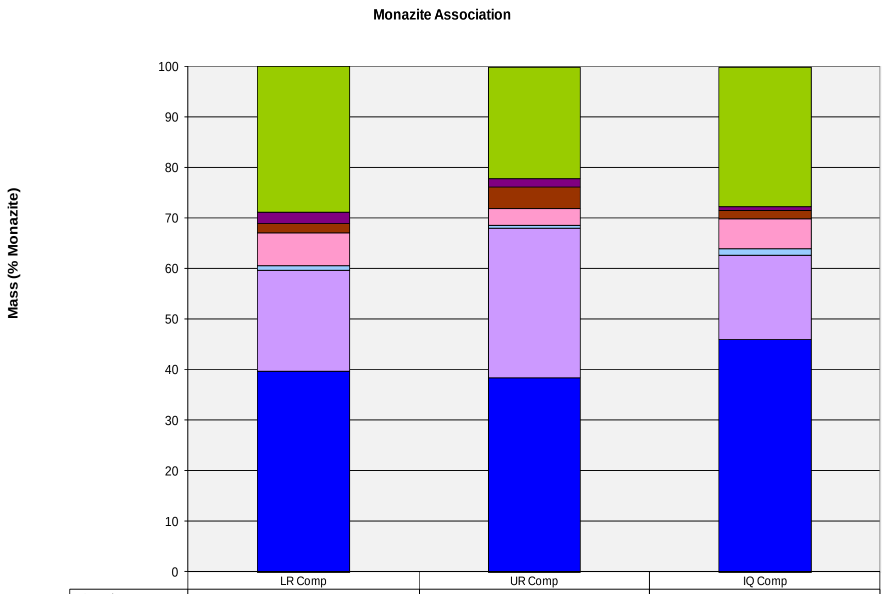
- Free and liberated allanite account for 35%, 63% and 54% for the LR, UR and IQ, respectively (Figure 5). The majority of the remainder occurs as complex particles (53%, 25% and 40%) and quartz/feldspars (6%, 9% and 4%).

- Free and liberated pyrite account for 95%, 94% and 97% for the LR, UR and IQ, respectively (Figure 6).



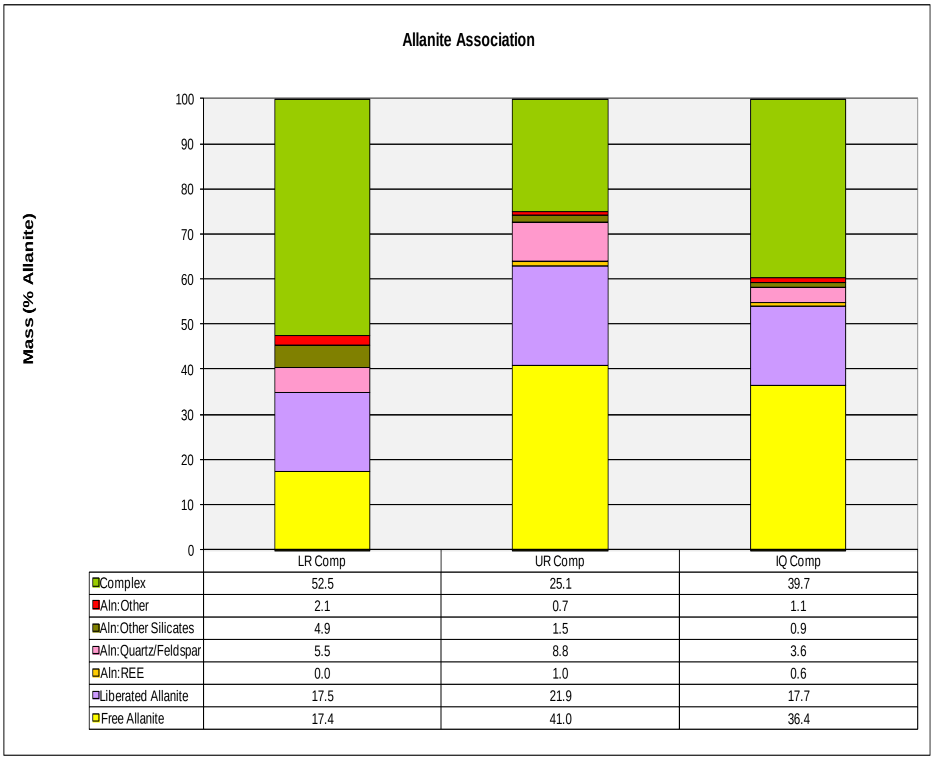
	LR Comp	UR Comp	IQ Comp
Complex	49.1	65.7	61.3
U-Th Min: Other	0.7	2.1	0.1
U-Th Min:Pyrite	1.1	3.8	5.8
U-Th Min:Quartz/Feldspar	2.5	3.8	1.8
U-Th Min:Monazite	5.2	4.0	11.8
Liberated U-Th Minerals	34.3	7.1	6.5
Free U-Th Minerals	7.1	13.4	12.8

Figure 3: Liberation and Association Profile of U-Th Minerals



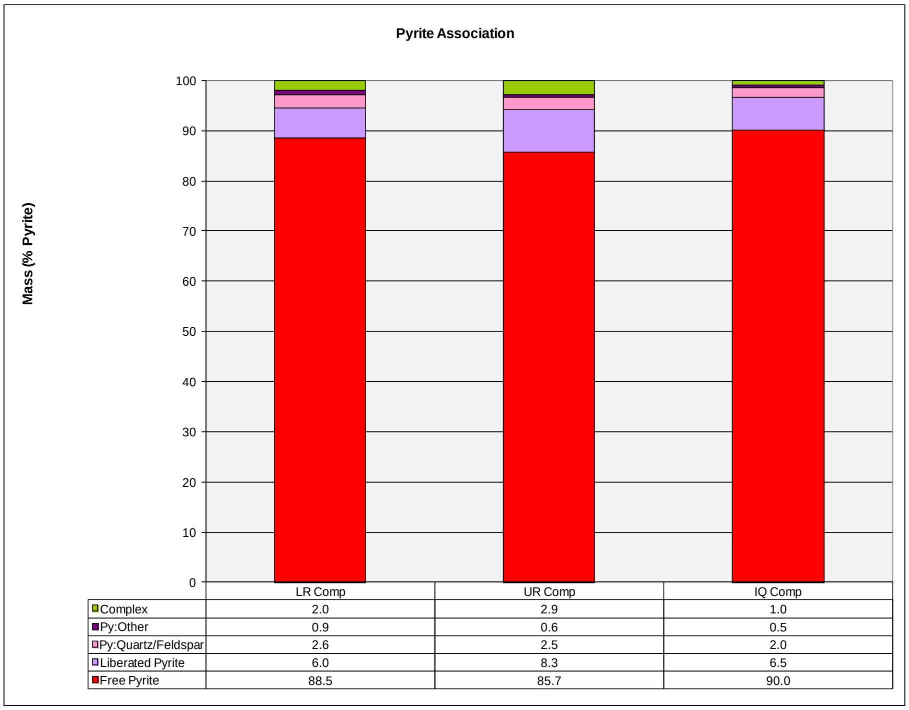
	LR Comp	UR Comp	IQ Comp
Complex	28.9	22.1	27.6
Mnz:Other	2.3	1.7	0.8
Mnz:Micas/Clays	1.9	4.2	1.7
Mnz:Quartz/Feldspar	6.3	3.3	5.8
Mnz:U-Th Minerals	1.1	0.5	1.4
Liberated Monazite	19.8	29.7	16.6
Free Monazite	39.7	38.3	45.9

Figure 4: Liberation and Association Profile of Monazite



	LR Comp	UR Comp	IQ Comp
■ Complex	52.5	25.1	39.7
■ Aln:Other	2.1	0.7	1.1
■ Aln:Other Silicates	4.9	1.5	0.9
■ Aln:Quartz/Feldspar	5.5	8.8	3.6
■ Aln:REE	0.0	1.0	0.6
■ Liberated Allanite	17.5	21.9	17.7
■ Free Allanite	17.4	41.0	36.4

Figure 5: Liberation and Association Profile of Allanite



	LR Comp	UR Comp	IQ Comp
■ Complex	2.0	2.9	1.0
■ Py:Other	0.9	0.6	0.5
■ Py:Quartz/Feldspar	2.6	2.5	2.0
■ Liberated Pyrite	6.0	8.3	6.5
■ Free Pyrite	88.5	85.7	90.0

Figure 6: Liberation and Association Profile of Pyrite

EMPA of the U-Th phase reveals that the UR and IQ composites contain mainly uranothorite-thorite [(Th,U)SiO_4]. The LR Comp contains altered uraninite (UO_2), coffinite [$U(SiO_4)_{1-x}(OH)_{4x}$] and uranothorite-thorite [(Th,U)SiO_4]. Brannerite was tentatively identified during the SEM examination of the samples as micrometric inclusions in various minerals.

EMPA was carried out on monazite to determine the chemistry of the mineral in the three composites. The average values of the main REE (Ce, La and Nd) vary slightly among monazite in the three composites. Monazite contains trace amounts of Y (0.14%, 0.49% and 0.11% in LR, UR and IQ, respectively). Th was found to range from 2.00%, 3.44% and 1.27% in LR, UR and IQ, respectively. It is critical to note that monazite does not carry U.

The grade-recovery curves represent the ideal and potential maximum recovery for physical separation (gravity, flotation) of the specific mineral and not chemical recovery (leaching).

At a K_{80} of 75 µm mineralogy indicates U-Th minerals are capable of the following grades and recoveries:

- grades between 94% and 76% for recoveries of 41% to 71%, respectively, for the *LR*;
- grades between 95% and 65% for recoveries of 21% to 73%, respectively, for *UR*;
- grades between 98% and 70% for mineral recoveries of 19% to 89%, respectively, for *IQ*.

The grade-recovery curve representing each composite indicates monazite:

- grades between 96% and 77% for recoveries of 59% to 89%, respectively, for the *LR*;
- grades between 96% and 82% for recoveries of 68% to 93%, respectively, for *UR*;
- grades between 97% and 80% for mineral recoveries of 62% to 92%, respectively, for *IQ*.

2.3. Bond Ball Mill Grindability Testing

The Bond ball mill grindability test was performed on the Master Composite using a 100 mesh (150 µm) closing screen. The results are summarized in Table 6 and compared to the SGS database in Figure 7. The Master Composite can be categorized as moderately hard. The test details are presented in Appendix A.

Table 6: Bond Ball Mill Grindability Test Summary

Sample Name	Mesh of Grind	F_{80} (µm)	P_{80} (µm)	Gram per Revolution	Work Index (kWh/t)	Hardness Percentile
Master Comp	100	2,002	124	1.53	16.2	70

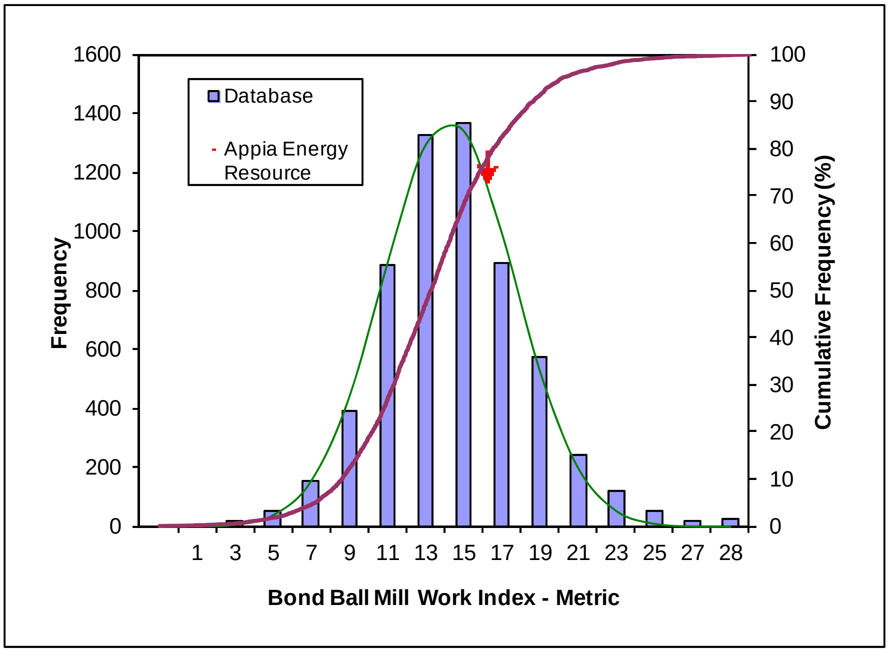


Figure 7: SGS Bond Ball Mill Work Index Database

3. Beneficiation Testing

3.1. Gravity Separation – Mozley Shaking Table

Two gravity separation tests at different sizes were performed on the Master Composite using a Mozley Shaking Table. Results are presented in Table 7. The gravity results were similar for grind sizes of 80% passing 208 µm in G1 and 108 µm in G2. Over half of the U, Th and TREE was recovered in the gravity concentrate in approximately 7% mass. The G1 cumulative gravity concentrate 1-2 recovered 52% U, 61% Th, 84% S and 59% TREE in 7% mass. The middlings were less than head grade and would likely be combined with the tailings. The tailings for each test was further screened into three fractions (+75, -75/+38 and -38 µm) to investigate the potential of preferential U, Th and TREE deportment by size. The U, Th and TREE were found to deport into the -38 µm fraction. The combined gravity concentrate and the -38 µm fraction of the gravity tailings recovered 75% U, 81% Th and 79% TREE in 26% mass. Further upgrading of the -38 µm fraction of the gravity tailings may be possible. The losses in the middlings and the coarser fraction (+38 µm) of the gravity tailings could be due to poor liberation.

Table 7: Gravity Separation Results - Mozley Shaking Table

Test No.	Conditions	Products	Weight	Grades: U, Th, REE-g/t; S%						% Distribution					
			%	U	Th	S	LREE	HREE	TREE	U	Th	S	LREE	HREE	TREE
G1	100% passing 48mesh, K80 = 208 μm	Conc1	2.4	1670	3600	44.1	18530	1116	19646	13.9	22.8	44.4	24.4	19.7	24.1
		Conc1-2	6.6	2282	3556	30.9	16477	1173	17650	51.5	61.1	84.3	58.7	56.1	58.6
		Conc+Midds	36.2	535	769	5.85	3641	261	3902	66.2	72.4	87.3	71.1	68.3	70.9
		Conc 1-2 + Tails -38 μm	25.7	850	1216	8.93	5671	418	6089	74.7	81.2	94.7	78.6	77.7	78.6
		Conc+Midds+Tails -38μm	55.4	473	644	4.29	3050	225	3275	89.4	92.5	97.7	91.0	89.9	90.9
		Tails (+38μm)	44.6	69.8	64.8	0.12	376	31.3	407	10.6	7.5	2.3	9.0	10.1	9.1
		Tails	63.8	155	167	0.48	842	68.8	911	33.8	27.6	12.7	28.9	31.7	29.1
		Head (Calc.)	100.0	293	385	2.43	1856	138	1995	100.0	100.0	100.0	100.0	100.0	100.0
G2	100% passing 100mesh, K80 = 108 μm	Conc1	2.2	1320	3340	44.5	18813	1016	19830	10.8	17.2	36.0	20.9	15.2	20.5
		Conc1-2	5.8	2025	4277	37.1	19363	1315	20678	44.8	59.3	81.0	58.0	53.1	57.7
		Conc+Midds	22.9	641	1234	9.81	5612	388	6000	55.6	67.1	83.9	65.9	61.4	65.6
		Conc1-2 + Tails -38μm	37.8	524	930	6.69	4237	304	4541	75.2	83.6	94.5	82.3	79.5	82.1
		Conc+Midds+Tails -38μm	54.8	413	701	4.75	3202	232	3433	86.0	91.3	97.4	90.2	87.9	90.0
		Tails (+38μm)	45.2	81.8	80.7	0.16	424	38.9	462	14.0	8.7	2.6	9.8	12.1	10.0
		Tails	77.1	151	179	0.56	860	72.3	932	44.4	32.9	16.1	34.1	38.6	34.4
		Head (Calc.)	100.0	263	421	2.67	1947	145	2091	100.0	100.0	100.0	100.0	100.0	100.0

3.2. Gravity Separation – Gekko 296A Procedure

A large scale gravity test was carried out on ~15 kg of the Master Composite with a Wilfley Table following the Gekko 296A procedure. The objective was to assess the amenability of the samples to the Gekko IPJ. The tabling procedure replicates the expected recovery that could be achieved by the IPJ. The feed was first stage-ground to 100% passing 20 mesh (850 µm). The ground sample was then fed across a Wilfley Table to collect a first concentrate. The combined middling and tailings was screen at 35 mesh (500 µm) with the oversize stage-ground to 100% pasing 500 µm size. These materials were tabled again to collect a second concentrate. The tailings from the second Wilfley Table pass were stage-ground to 100% passing 106 µm and passed through a cyclone for desliming. The underflow of the cyclone was tabled and a third concentrate was collected. A total of three concentrates, a gravity tailing and one slime product were collected and submitted for assay. The results of the test are summarized in Table 8.

Table 8: Gravity Separation Results – Wilfley Table at Progressive Grind

P_{80} (μm)	Products	Weight	Assays: U, Th, REE g/t, S %						% Distribution					
		%	S	U	Th	LREE	HREE	TREE	S	U	Th	LREE	HREE	TREE
1st Pass=850	W Conc-1	19.9	8.32	608.0	1010.0	4421.8	333.1	4754.9	82.9	53.0	59.1	54.8	48.7	54.3
2nd pass=500	W Conc 1-2	29.0	6.25	540.8	887.1	3892.0	294.8	4186.8	90.6	68.6	75.5	70.2	62.7	69.6
3rd pass=106	W Conc 1-3	33.0	5.64	508.2	832.8	3691.1	279.7	3970.8	93.3	73.4	80.7	75.8	67.8	75.2
	W Conc1-3 + Slime	35.5	5.30	497.1	800.9	3553.9	271.7	3825.5	94.2	77.2	83.5	78.5	70.8	77.9
	W Tails	64.5	0.18	80.9	87.3	536.7	61.7	598.4	5.8	22.8	16.5	21.5	29.2	22.1
	Head (Calc.)	100.0	2.00	228.6	340.5	1607.1	136.2	1743.3	100.0	100.0	100.0	100.0	100.0	100.0
	Head (Direct)		1.92	270.0	354.0	1578.5	123.9	1702.4						

Gravity separation using a Wilfley Table achieved similar results as with a Mozley Table. The combined gravity concentrate 1-3 recovered 73% U, 81% Th and 75% TREE in 33% mass. Excessive losses to the tailings could be due to liberation or in fines not recovered by gravity. Further screening of the tailings and upgrading using alternative beneficiation such as flotation could result in further recovery increases or upgrading.

3.3. Magnetic Separation-WHIMS

The magnetic separation tests were carried out with two ~ 100g charges of the Master Composite which were ground to a P_{80} of 208 µm in M1 and 108 µm in M2. The ground sample was pulped, well mixed, and fed through a wet high intensity magnetic separation (WHIMS) at intensities of 2A (~2,000 Gauss), 5A (5,000 Gauss), 10A (10,000 Gauss) and 20A (17,000 Gauss), respectively. Results are presented in Table 9. The 20A non-magnetics of each test was screened into three fractions (+75, -75/+38 and -38µm) to further investigate the U and REE deportment by size.

Table 9: Magnetic Separation Results

Test No	Conditions	Cumulative Products	Mass	Grades: U, Th, REE-g/t; S						% Distribution					
			%	U	Th	S	LREE	HREE	TREE	U	Th	S	LREE	HREE	TREE
M1	100% passing 48mesh P80=208 µm	0-2A WHIMS Mag	2.2	1540	2140	6.08	9705	725	10430	9.1	8.5	7.2	8.1	8.9	8.1
		0-5A WHIMS Mag	4.5	1950	3233	6.02	17397	1078	18475	23.7	26.3	14.5	29.7	27.0	29.6
		0-10A WHIMS Mag	6.6	2496	4670	6.25	24352	1473	25825	44.3	55.4	22.0	60.8	53.9	60.4
		0-20A WHIMS Mag	8.6	2619	4752	6.21	23804	1498	25302	61.2	74.2	28.8	78.2	72.2	77.8
		0-20A Mag+ Non-Mag -38µm	28.9	958	1606	2.99	7813	512	8325	74.8	83.8	46.3	85.7	82.4	85.5
		20A WHIMS Non-Mag +38µm	71.1	131	126	1.41	529	44.5	573	25.2	16.2	53.7	14.3	17.6	14.5
		Non-Mag -38µm	20.3	249	554	1.61	984	90.5	1074	13.6	9.6	17.5	7.6	10.2	7.7
		Head (Calc.)	100.0	370	554	1.86	2634	180	2814	100.0	100.0	100.0	100.0	100.0	100.0
M2	100% passing 100mesh, P80= 108µm	0-2A WHIMS Mag	2.5	890	1370	4.00	6117	500	6617	8.9	8.8	5.5	8.4	9.2	8.4
		0-5A WHIMS Mag	4.9	1457	2833	5.19	15903	1005	16908	28.2	35.3	13.8	42.2	35.7	41.7
		0-10A WHIMS Mag	4.9	1457	2833	5.19	15903	1005	16908	28.2	35.3	13.8	42.2	35.7	41.7
		0-20A WHIMS Mag	7.6	1763	3364	5.12	17556	1147	18703	52.6	64.6	20.9	71.6	62.7	71.0
		0-20A Mag+ Non-Mag -38µm	38.5	495	834	2.15	4122	289	4411	74.9	81.2	44.5	85.3	80.1	84.9
		20A WHIMS Non-Mag +38µm	61.5	104	121	1.68	447	44.9	492	25.1	18.8	55.5	14.7	19.9	15.1
		Non-Mag -38µm	30.9	184	213	1.42	821	78.1	900	22.3	16.6	23.6	13.6	17.4	13.9
		Head (Calc.)	100.0	255	396	1.86	1863	139	2002	100.0	100.0	100.0	100.0	100.0	100.0

The majority of mass (>90%) reported to the non-magnetics. Results between the two grind sizes were similar. The M1 combined magnetics recovered 61% U, 74% Th and 78% TREE in 9% mass. Magnetic separation resulted in greater U and Th recoveries than gravity separation. The U and Th bearing minerals are more susceptible to magnetic separation than gravity separation. Magnetic separation could be considered as part of the flowsheet but further scavenging of the non-magnetics by using an alternative process, such as flotation, would be required.

3.4. Flotation

A total of 20 batch flotation tests were performed on the Master, LR, UR and IQ composites and one flotation test was conducted on an acid leach residue (F18). Collector LR19 was evaluated in tests F1, F2 and F4 at different pH values as well as with and without pre-floating pyrite. In test F3, Cupferron and FS-2 as the collectors were compared against Salicycylhydroxamate (SHD), a Chinese hydroxamate reagent effective on other REE ores, in F5, but these tests showed inferior performance for U, Th and TREE. Subsequent testwork used LR19 to evaluate pulp pH, primary grind size, sulphide pre-float, desliming, modifiers and depressants, collector addition rate and the pulp density. Tests F9 and F10 were carried out on the LR composite in a 28L flotation cell. Cleaning was performed on the F10 rougher concentrate. Tests F16 and F17 were conducted on the separate UR and IQ comp in a 28L flotation cell. Tests F19, F20 and F21 were performed on the Master composite with different conditioning time and air flow rate. A

10% mass pull was targeted to generate concentrate for subsequent hydrometallurgical tests. Rougher conditions are presented in Table 10.

Table 10: Summary of Rougher Flotation Conditions

Comp	Test No.	Solid	Fd K80	pH	Collector (g/t)						Depressant (g/t)			g/t	g/t	g/t	Target Mass
		%	µm		KAX	LR19	FS-2	Cupferron	V4085	SHD	Metso	Na_2SiF_6	Oxalic	H_2SO_4	Na_2CO_3	DF250	%
MC	F1	35	111	8.0-10.2		750			75		500					18	
MC	F2	35	111	7.6-8.4		500					500			140			
MC	F3	35	111	6.5-7.2	50		495	2000						980		5	
MC	F4	35	111	6.5-6.5	50	450					500			900		5	
MC	F5	35	111	7.8-10.2	50	150				3000	500			150		2.5	
MC	F6	35	135	7.8-10.2		1425					500					26	
MC	F7	35	80	7.8-9.8		1125					500					18	
MC	F8	35	102	7.8-10.2	50	500					500					4	
MC	F13	18	135	6.3-7.7		1200					500		900				
MC	F15	18	135	7.0-7.8		900					500		800				
MC	F19	35	135	7.4-9.3		300					500				250		10
MC	F20	35	135	7.2-9.4		300					500				250		10
MC	F21	35	135	9.3-9.5		300					500				250		10
LR	F9	35	131	9.3-9.7		1100					500						
LR	F10	35	100	9.6-9.7		900					500						
LR	F11	35	135	6.5-10.5		1200					500		900				
LR	F12	35	135	6.3-10.5		1200					500	900					
LR	F14	18	135	6.5-7.4		1200					500		900				
UR	F16	35	117	9.4-9.4		1200					500						
IQ	F17	35	122	9.6-9.6		750					500						

Figure 8 to Figure 10 compare the mass-recovery relationships for U, Th, and TREE of the best test results achieved on the Master composite. Note that tests F6, F7 and F15 are cleaner tests and F13 is a rougher kinetics test. The rougher and the cleaner flotation results are presented in Table 11 and Table 12, respectively.

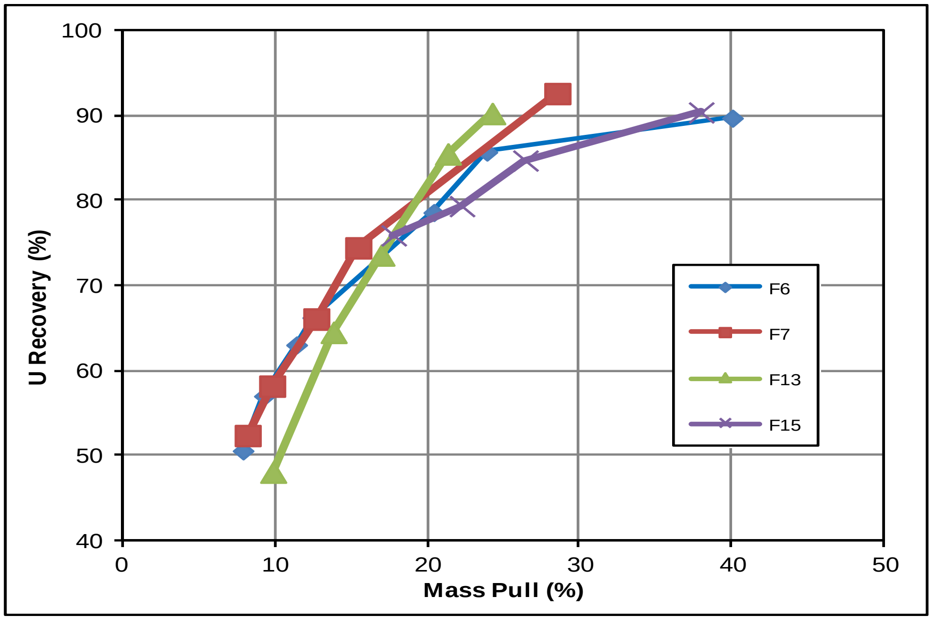


Figure 8: U Recovery vs. Mass Pull for F6, F7, F13 and F15

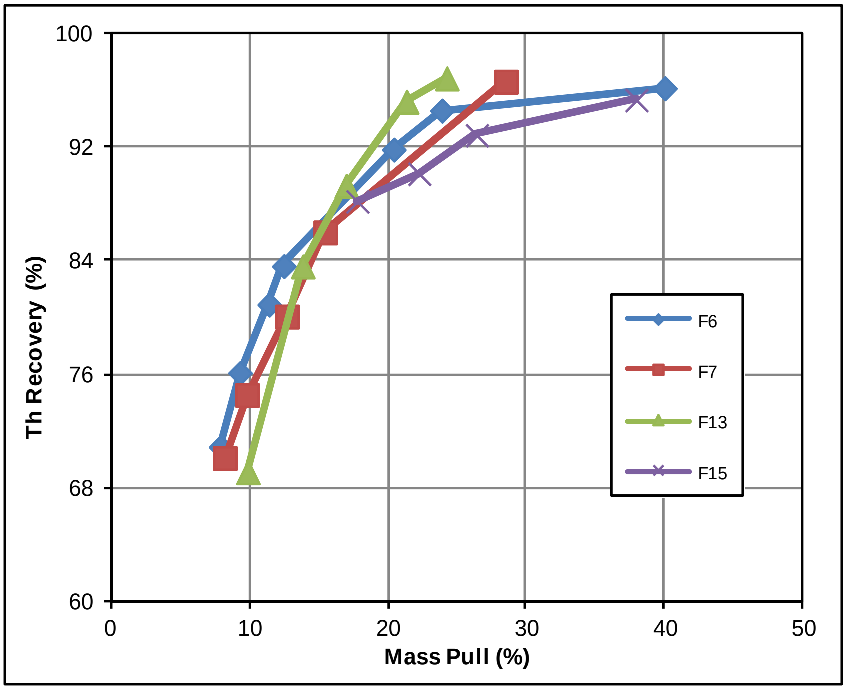


Figure 9: Th Recovery vs. Mass Pull for F6, F7, F13 and F15

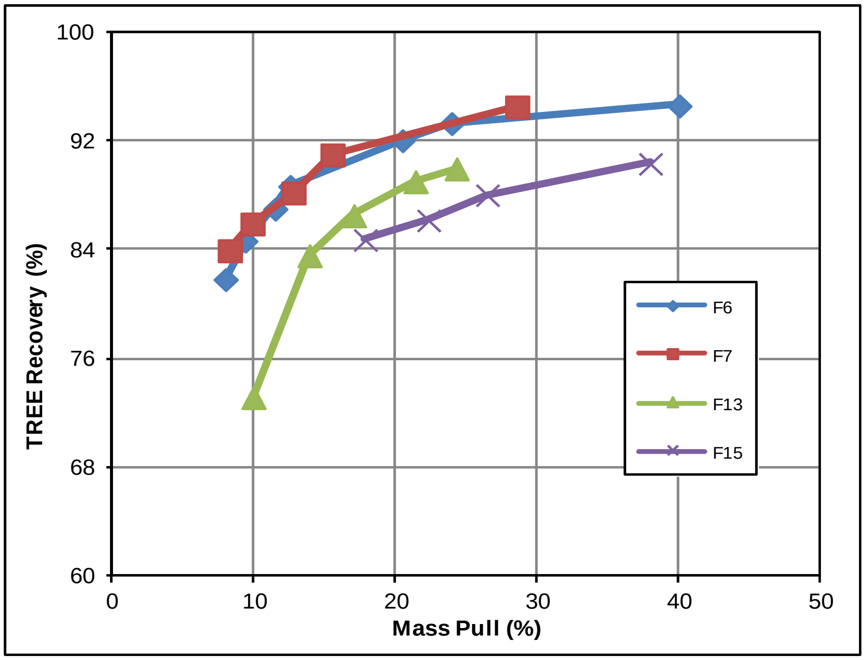


Figure 10: TREE Recovery vs. Mass Pull for F6, F7, F13 and F15

Table 11: Results of Rougher Flotation Tests

Test No.	Conditions	Reagent	Cumulative Products	Mass	Grades: U, Th, REE-g/t; S %						% Distribution					
		g/t		%	U	Th	S	LREE	HREE	TREE	U	Th	S	LREE	HREE	TREE
F1	Master Comp	Metso= 500	Ro Conc 1	7.8	760	1930	19.4	14261	704	14965	23.5	43.3	80.1	70.7	43.1	68.7
	Fd: K80~111µm	LR19=750	Ro Conc 1-2	14.7	780	1562	12.3	9445	555	10000	45.5	66.3	95.9	88.7	64.3	86.8
	Pulp density: 35%	DF250=18	Ro Conc 1-3	19.4	777	1400	9.54	7518	491	8008	59.8	78.3	98.2	92.9	74.9	91.6
	pH=8.0-10.2	V4085=75	Ro Conc 1-4	23.6	789	1305	7.92	6321	451	6771	73.8	88.7	99.1	95.0	83.7	94.2
			Ro Conc 1-5	28.6	751	1145	6.56	5277	398	5675	85.2	94.5	99.6	96.3	89.7	95.8
			Ro Conc 1-6	**30.1**	**739**	**1100**	**6.25**	**5037**	**384**	**5422**	**88.1**	**95.4**	**99.6**	**96.5**	**91.0**	**96.1**
			Ro Tls	69.9	43.1	22.7	0.01	77.8	16.4	94.2	11.9	4.6	0.4	3.5	9.0	3.9
			Head (Calc.)	100.0	252.4	347.0	1.89	1570.0	127.1	1697.1	100.0	100.0	100.0	100.0	100.0	100.0
F2	Master Comp	Metso= 500	Ro Conc 1	15.0	867	1660	97.6	9366	578	9944	49.5	71.5	97.6	87.4	66.8	85.9
	Fd: K80~111µm	LR19=900	Ro Conc 1-2	21.4	784	1347	1.37	6917	467	7385	63.9	82.7	98.9	92.1	77.1	91.0
	Pulp density: 35%	H_2SO_4=140	Ro Conc 1-3	28.0	722	1143	0.49	5466	396	5862	76.8	91.6	99.4	95.0	85.3	94.2
	pH=7.6-8.4		Ro Conc 1-4	32.7	692	1018	0.17	4755	357	5112	85.9	95.2	99.6	96.5	89.7	95.9
			Ro Conc 1-5	**36.2**	**672**	**940**	**0.07**	**4328**	**332**	**4661**	**92.4**	**97.3**	**99.7**	**97.2**	**92.5**	**96.9**
			Ro Tls	63.8	31.5	14.5	0.34	70.3	15.2	85.5	7.6	2.7	0.3	2.8	7.5	3.1
			Head (Calc.)	100.0	263.1	349.1	1.88	1610	130	1740	100.0	100.0	100.0	100.0	100.0	100.0
F3	Master Comp	Cupferron=2000	S Ro Conc 1	4.9	765	737	34.4	2474	284	2758	14.9	11.2	91.7	7.4	10.3	7.6
	Fd: K80~111µm	KAX=50	Ro Conc 1-2	14.2	856	1078	12.5	4967	409	5376	47.9	46.9	95.5	42.7	42.7	42.7
	Pulp density: 35%	FS-2=495	Ro Conc 1-3	38.0	466	615	4.75	2956	236	3192	69.9	71.8	97.4	68.3	66.0	68.1
	pH=6.5-7.2	DF250=5	Ro Conc 1-4	**73.3**	**291**	**386**	**2.49**	**1908**	**156**	**2065**	**84.3**	**86.9**	**98.6**	**85.0**	**84.5**	**84.9**
	Pre-float Sulphide	H_2SO_4=980	Ro Tls	26.7	149	160	0.10	926	79	1005	15.7	13.1	1.4	15.0	15.5	15.1
			Head (Calc.)	100.0	253	325	1.85	1646	136	1782	100.0	100.0	100.0	100.0	100.0	100.0
F4	Master Comp	Metso= 500	S Ro Conc 1	5.7	684	692	30.7	2414	256	2670	15.9	12.4	91.8	8.5	11.0	8.6
	Fd: K80~111µm	LR19=450	Ro Conc 1-2	18.6	676	1052	9.92	5735	368	6103	51.5	61.6	97.2	65.8	52.0	64.8
	Pulp density: 35%	KAX=50	Ro Conc 1-3	29.5	551	809	6.34	4211	285	4497	66.6	75.2	98.4	76.6	63.9	75.6
	pH=6.0-6.5	DF250=5	Ro Conc 1-4	**48.0**	**391**	**556**	**3.91**	**2837**	**203**	**3040**	**76.8**	**84.0**	**98.9**	**83.9**	**74.0**	**83.2**
	Pre-float Sulphide	H_2SO_4=900	Ro Tls	52.0	109	98	0.04	502	66	568	23.2	16.0	1.1	16.1	26.0	16.8
			Head (Calc.)	100.0	244	318	1.90	1623	132	1755	100.0	100.0	100.0	100.0	100.0	100.0
F5	Master Comp	Metso= 500	So Ro Conc1	5.8	773	878	30.0	3121	338	3459	19.3	17.5	92.2	11.3	16.4	11.6
	Fd: K80~111µm	SHD=3000	Ro Conc 1-2	9.1	630	749	19.37	2909	287	3197	24.7	23.4	93.2	16.5	21.8	16.8
	Pulp density: 35%	LR19=150	Ro Conc 1-3	19.1	431	515	9.38	2292	203	2495	35.3	33.7	94.5	27.2	32.2	27.5
	pH=7.8-10.2	KAX=50	Ro Conc 1-4	**24.0**	**653**	**958**	**7.70**	**5273**	**373**	**5646**	**67.4**	**78.8**	**97.6**	**78.7**	**74.4**	**78.4**
		DF250=2.5	Ro Tls	76.0	99.9	81.4	0.06	452.1	41	493	32.6	21.2	2.4	21.3	25.6	21.6
		H_2SO_4=150	Head (Calc.)	100.0	233	292	1.89	1610.3	120.3	1730.6	100.0	100.0	100.0	100.0	100.0	100.0
		NaOH=150	Head (Direct)		270	354	1.92	1578.5	123.9	1702.4						
F9	LR Comp	Metso= 500	REO Ro Conc	**32.9**	**1270**	**1030**	**11.0**	**4330.2**	**382.2**	**4712.4**	**85.5**	**93.3**	**98.5**	**94.5**	**90.4**	**94.1**
	Fd: K80~131µm	LR19=1100	REO Ro Tail	67.1	106	36.3	0.08	124.8	20.0	144.8	14.5	6.7	1.5	5.5	9.6	5.9
	Pulp density: 35%		Head (Calc.)	100.0	489.3	363.6	3.68	1506.5	139.3	1645.7	100.0	100.0	100.0	100.0	100.0	100.0
	Ro: pH=9.3-9.7															
F10	LR Comp	Metso= 500 in Ro	REO Clnr Conc	**22.7**	**1100**	**1020**	**16.1**	**5252.5**	**380.1**	**5632.6**	**49.4**	**69.6**	**95.3**	**85.7**	**64.2**	**83.9**
	Fd: K80~100µm	Metso= 700 in Cl	REO Clnr Tail	40.4	405	161	0.20	281.5	74.5	356.0	32.3	19.5	2.1	8.2	22.4	9.4
	Pulp density: 35%	LR19=900 in Ro	REO Ro Tail	36.8	252	98.5	0.27	230.0	48.8	278.8	18.3	10.9	2.6	6.1	13.4	6.7
	Ro: pH=9.6-9.7	LR19=350 in Cl	Head (Calc.)	100.0	506.7	333.3	3.84	1392.8	134.5	1527.3	100.0	100.0	100.0	100.0	100.0	100.0
F11	LR Comp	Metso= 500	Ro Conc 1	14.2	1270	1330	22.7	8246	518	8764	39.9	61.4	96.7	80.2	47.1	77.0
	Fd: K80~135µm	LR19=1200	Ro Conc 1-2	20.4	1371	1196	16.0	6136	467	6602	61.9	79.4	98.2	85.9	61.0	83.5
	Pulp density: 35%	Oxalic acid=900	Ro Conc 1-3	26.4	1332	1029	12.5	4935	411	5346	77.8	88.3	98.9	89.3	69.5	87.4
	Ro: pH=6.5-8.7		Ro Conc 1-4	31.6	1245	903	10.5	4220	369	4589	87.2	92.9	99.3	91.5	74.7	89.8
			Ro Conc 1-5	**34.4**	**1186**	**843**	**9.62**	**3911**	**347**	**4257**	**90.4**	**94.4**	**99.4**	**92.3**	**76.5**	**90.8**
			REO Ro Tls	65.6	66	26	0.03	171	56	227	9.6	5.6	0.6	7.7	23.5	9.2
			Head (Calc.)	100.0	451.5	307.2	3.33	1458.1	156.1	1614.2	100.0	100.0	100	100.0	100.0	100.0
F12	LR Comp	Metso= 500	Ro Conc 1	17.7	1330	957	18.8	3916	383	4299	52.3	55.2	96.9	47.6	45.0	47.4
	Fd: K80~135µm	LR19=1200	Ro Conc 1-2	23.3	1294	948	14.5	4238	379	4617	67.0	71.9	98.4	67.9	58.6	67.0
	Pulp density: 35%	Na_2SiF_6 = 900	Ro Conc 1-3	26.2	1256	907	12.94	4074	365	4439	73.2	77.4	98.9	73.4	63.5	72.5
	Ro: pH=6.3-6.6		Ro Conc 1-4	34.5	1119	804	9.87	3704	324	4028	85.9	90.4	99.3	87.9	74.3	86.6
			Ro Conc 1-5	**38.9**	**1032**	**729**	**8.75**	**3363**	**297**	**3660**	**89.4**	**92.6**	**99.5**	**90.1**	**76.9**	**88.8**
			REO Ro Tls	61.1	78	37	0.03	236	57	293	10.6	7.4	0.5	9.9	23.1	11.2
			Head (Calc.)	100.0	449.5	306.6	3.43	1453.3	150.5	1603.8	100.0	100.0	100.0	100.0	100.0	100.0
F13	Master Comp	Metso= 500	Ro Conc 1	9.9	1100	2200	18.6	12493	775	13268	48.1	69.1	95.6	75.6	48.2	73.2
	Fd: K80~135µm	LR19=1200	Ro Conc 1-2	13.8	1053	1900	13.5	10149	672	10822	64.5	83.6	97.4	86.0	58.5	83.6
	Pulp density: 18%	Oxalic acid=900	Ro Conc 1-3	17.0	980	1654	11.1	8545	588	9133	73.6	89.3	97.9	88.8	62.8	86.5
	Ro: pH=6.3-7.7		Ro Conc 1-4	21.3	905	1403	8.86	6974	505	7478	85.5	95.2	98.3	91.1	67.7	89.0
			Ro Conc 1-5	**24.2**	**841**	**1257**	**7.81**	**6197**	**456**	**6653**	**90.3**	**96.9**	**98.4**	**92.0**	**69.5**	**90.0**
			REO Ro Tls	75.8	29	13	0.04	173	64	237	9.7	3.1	1.6	8.0	30.5	10.0
			Head (Calc.)	100.0	225.9	314.4	1.92	1633.0	159.0	1792.0	100.0	100.0	100.0	100.0	100.0	100.0
F16	UR Comp	Metso = 500	REO Ro Conc 1-4	**18.5**	**620**	**1680**	**7.88**	**10335**	**606**	**10941**	**56.3**	**75.5**	**97.2**	**89.6**	**64.5**	**87.7**
	Fd: K80~117µm	LR19 = 1200	REO Ro Conc 1-5	24.7	604	1477	5.98	7989	522	8511	73.0	88.5	98.3	92.4	74.0	91.0
	Pulp density: 35%		REO Ro Conc 1-6	33.3	493	1136	4.45	6006	409	6415	80.4	91.7	98.7	93.6	78.1	92.4
	pH=9.4		REO Ro Tail	66.7	60	51	0.03	205	57	262	19.6	8.3	1.3	6.4	21.9	7.6
			Head (Calc.)	100.0	204.0	411.9	1.50	2134.3	174.0	2308.2	100.0	100.0	100.0	100.0	100.0	100.0
F17	IQ Comp	Metso = 500	REO Ro Conc 1-3	**11.0**	**343**	**1070**	**9.36**	**8267**	**365**	**8632**	**35.7**	**57.2**	**95.2**	**83.4**	**40.9**	**79.9**
	Fd: K80~122µm	LR19 = 750	REO Ro Conc 1-4	14.1	364	1053	7.41	6636	336	6972	48.7	72.4	96.8	86.0	48.3	82.9
	Pulp density: 35%		REO Ro Tail	85.9	63	66	0.04	177	59	236	51.3	27.6	3.2	14.0	51.7	17.1
	pH=9.6		Head (Calc.)	100.0	105.4	205.3	1.08	1088.9	98.1	1187.0	100.0	100.0	100.0	100.0	100.0	100.0
F19	Master Comp	Metso= 500	Ro Conc 1	8.1	845	2140	20.8	15850	880	16730	28.6	51.8	88.7	77.8	48.1	75.4
	Fd: K80~135µm	LR19=300	Ro Conc 1-2	9.1	800	1973	19.0	14434	810	15244	30.2	53.3	90.2	79.0	49.4	76.6
	No desliming	Ro: pH=7.4-9.5	Ro Conc 1-3	**10.5**	**734**	**1749**	**16.5**	**12577**	**717**	**13294**	**32.1**	**54.7**	**91.1**	**79.8**	**50.6**	**77.4**
	5 mins cond		REO Ro Tls	89.5	182	170	0.19	374	82	456	67.9	45.3	8.9	20.2	49.4	22.6
	4 mins float;		Head (Calc.)	100.0	240	335.8	1.91	1655.1	148.7	1803.7	100.0	100.0	100.0	100.0	100.0	100.0
F20	Master Comp	Metso= 500	Ro Conc 1	8.9	741	1870	19.7	13640	745	14385	28.6	52.1	89.9	77.7	47.0	75.1
	Fd: K80~135µm	LR19=300	Ro Conc 1-2	10.4	692	1673	17.3	11976	667	12643	31.1	54.1	91.6	79.2	48.9	76.7
	No desliming	Ro: pH=7.2-9.4	Ro Conc 1-3	**11.5**	**653**	**1538**	**15.7**	**10854**	**613**	**11467**	**32.6**	**55.3**	**92.3**	**79.9**	**50.0**	**77.4**
	10 mins cond		REO Ro Tls	88.5	176	162	0.17	357	80	437	67.4	44.7	7.7	20.1	50.0	22.6
	4 mins float		Head (Calc.)	100.0	231	320.8	1.96	1568.5	141.5	1710.0	100.0	100.0	100.0	100.0	100.0	100.0
F21	Master Comp	Metso= 500	Ro Conc 1	7.6	788	2050	22.6	16210	860	17070	24.2	46.5	89.0	75.6	45.3	73.1
	Fd: K80~135µm	LR19=300	Ro Conc 1-2	8.9	731	1821	19.6	14065	762	14827	26.3	48.4	90.5	76.9	47.1	74.5
	No desliming	Ro: pH=9.3-9.5	Ro Conc 1-3	**9.9**	**694**	**1685**	**17.9**	**12842**	**705**	**13547**	**27.6**	**49.5**	**91.1**	**77.6**	**48.1**	**75.2**
	10 mins cond		REO Ro Tls	90.1	199	188	0.19	406	83	489	72.4	50.5	8.9	22.4	51.9	24.8
	8 mins float; half air		Head (Calc.)	100.0	248	335.6	1.93	1631.9	144.3	1776.1	100.0	100.0	100.0	100.0	100.0	100.0

Table 12: Results of Cleaner Flotation Tests

Test No.	Conditions	Reagent g/t	Cumulative Products	Mass	Grades: U, Th, REE-g/t; S %						% Distribution					
				%	U	Th	S	LREE	HREE	TREE	U	Th	S	LREE	HREE	TREE
F6	Master Comp	Metso= 500 in Ro	REO 3rd Cl Conc	7.9	1470	2930	22.3	16932	1028	17960	50.6	70.9	94.7	84.0	57.6	81.8
	Fd: K80~135µm	LR19=1425 in Ro	REO 2nd Cl Conc	9.3	1408	2673	19.17	14857	937	15793	57.1	76.1	95.8	86.7	61.8	84.7
	Pulp density: 35%	DF250=26 in Ro	REO 1st Cl Conc	11.4	1269	2317	15.77	12422	811	13233	63.1	80.9	96.7	88.9	65.6	87.0
	Ro: pH=7.8-10.2	Metso= 500 in Cl	REO 1st Cl Conc +1st Cl Sc Conc	12.5	1220	2191	14.53	11572	766	12338	66.3	83.7	97.4	90.5	67.8	88.7
	Cl: pH=9.3-9.8	LR19=180 in Cl	**REO Ro Conc**	**20.4**	**885**	**1471**	**8.99**	**7314**	**516**	**7831**	**78.6**	**91.9**	**98.5**	**93.6**	**74.7**	**92.1**
			REO Ro Conc + Ro Sc Conc	23.9	825	1294	7.70	6322	460	6782	85.7	94.6	98.8	94.7	77.9	93.3
			REO Ro Conc + Ro Sc Conc + Ro Sc Tail -38 um	40.0	516	786	--	3817	291	4108	89.7	96.2	--	95.7	82.6	94.6
			Ro Scav Tail +38um	60.0	39.3	20.8	--	114.8	41.0	155.8	10.3	3.8	--	4.3	17.4	5.4
			Head (Calc.)	100.0	229.8	326.8	1.9	1595.0	141.1	1736.1	100.0	100.0	100	100.0	100.0	100.0
F7	Master Comp	Metso= 500 in Ro	REO 3rd Cl Conc	8.2	1410	2930	21.6	16885	999	17884	52.4	70.1	95.1	86.4	57.0	84.0
	Fd: K80~80µm	LR19=1125 in Ro	REO 2nd Cl Conc	9.8	1310	2606	18.22	14424	888	15312	58.2	74.6	95.9	88.2	60.6	85.9
	Pulp density: 35%	DF250=18 in Ro	REO 1st Cl Conc	12.7	1149	2163	14.21	11406	738	12144	66.1	80.1	96.8	90.3	65.2	88.2
	Ro: pH=7.8-9.8	Metso= 500 in Cl	REO 1st Cl Conc +1st Cl Sc Conc	15.5	1064	1908	11.82	9644	652	10296	74.5	86.0	98.0	92.9	70.1	91.0
	Cl: pH=9.3-9.8	LR19=180 in Cl	**REO Ro Conc**	**28.5**	**717**	**1163**	**6.49**	**5398**	**402**	**5800**	**92.6**	**96.6**	**99.2**	**95.9**	**79.6**	**94.5**
			REO Ro Conc + Ro Tail -38 um	51.1	416	660	--	3051	242	3293	96.4	98.4	--	97.2	86.1	96.3
			Ro Ro Tail +38um	48.9	16.2	11.5	--	92.7	41.0	133.7	3.6	1.6	--	2.8	13.9	3.7
			Head (Calc.)	100.0	220.9	343.0	1.9	1605.3	143.8	1749.2	100.0	100.0	100	100.0	100.0	100.0
F8	Master Comp	S Ro: KAX=50; DF250=4	REO 3rd Cl Conc	6.3	1820	3210	1.55	18940	1247	20187	48.8	65.6	5.0	74.5	58.4	73.2
	Fd: K80~µm	Ro: Metso= 500	REO 2nd Cl Conc	8.2	1515	2568	1.30	14854	1002	15856	53.5	69.1	5.5	76.9	61.8	75.8
	Pulp density: 35%	LR19=500 (one stage)	REO 1st Cl Conc	11.0	1211	1989	1.05	11304	779	12083	57.3	71.7	6.0	78.4	64.3	77.3
	S Ro: pH=8.4	Clnr: Metso=325	REO 1st Cl Conc +1st Cl Sc Conc	12.5	1108	1779	0.95	10028	700	10728	59.4	72.7	6.1	78.9	65.5	77.8
	Ro: pH=7.8-10.2	LR19=175	**REO Ro Conc**	**17.7**	**820**	**1283**	**0.70**	**7151**	**507**	**7659**	**62.2**	**74.2**	**6.4**	**79.6**	**67.2**	**78.6**
	Cl: pH=7.7-7.9	H_2SO_4=110	REO Ro Conc + Slime	24.6	677	1027	0.66	5677	409	6086	71.3	82.4	8.4	87.7	75.2	86.7
			REO Ro Conc + Slime + S Cl Tail	26.2	674	1018	1.06	5515	404	5919	75.7	87.1	14.4	90.8	79.2	89.9
			REO Ro Conc + Slime + S	29.8	669.5	975.1	6.32	5108.4	387.2	5495.6	85.6	95.0	97.5	95.7	86.4	95.0
			Head (Calc.)	100.0	233.3	306.2	1.93	1591.4	133.7	1725.1	100.0	100.0	100.0	100.0	100.0	100.0
F14	LR Comp	Metso= 500 in Ro	REO 3rd Cl Conc	16.7	1280	1350	19.4	6885	506	7391	51.5	71.6	96.9	78.9	57.6	76.9
	Fd: K80~135µm	LR19=1200 in Ro	REO 2nd Cl Conc	20.9	1166	1164	15.73	5779	443	6221	58.4	76.9	97.8	82.4	62.7	80.6
	Pulp density: 18%	Oxalic acid=900 in Ro	REO 1st Cl Conc +1st Cl Sc Conc	30.9	935	864	10.73	4138	337	4475	69.3	84.4	98.8	87.4	70.8	85.9
	Ro: pH=6.5-7.4	Metso= 500 in Cl	REO 1st Cl Sc Tail	27.6	229	103	0.07	316	67	383	15.2	9.0	0.6	6.0	12.6	6.6
	Cl: pH=7.2-7.9	LR19=135 in Cl	**REO Ro Conc**	**58.5**	**601**	**504**	**5.70**	**2333**	**210**	**2543**	**84.5**	**93.4**	**99.4**	**93.4**	**83.4**	**92.5**
		Oxalic acid=325 in Cl	REO Ro Tail	41.5	156	50	0.05	233	59	292	15.5	6.6	0.6	6.6	16.6	7.5
			Head (Calc.)	100.0	417	316	3.35	1461	147	1609	100.0	100.0	100.0	100.0	100.0	100.0
F15	Master Comp	Metso= 500 in Ro	REO 2nd Cl Conc	17.8	1050	1520	10.1	7839	562	8401	75.9	88.2	97.7	87.1	61.2	84.7
	Fd: K80~135µm	LR19=900 in Ro	REO 1st Cl Conc	22.3	877	1241	8.09	6360	466	6826	79.4	90.1	98.0	88.5	63.6	86.2
	Pulp density: 18%	Oxalic acid=800 in Ro	REO 1st Cl Conc +1st Cl Sc Conc	26.4	789	1077	6.84	5466	410	5876	84.7	92.9	98.3	90.3	66.4	88.0
	Ro: pH=7.0-7.8	Metso= 375 in Cl	**REO Ro Conc**	**37.9**	**586**	**770**	**4.78**	**3890**	**309**	**4199**	**90.4**	**95.3**	**98.6**	**92.2**	**71.9**	**90.4**
	Cl: pH=7.6-8.2	LR19=270 in Cl	REO Ro Tail	62.1	38	23	0.04	200	74	274	9.6	4.7	1.4	7.8	28.1	9.6
		Oxalic acid=200 in Cl	Head (Calc.)	100.0	246	306	1.84	1600	163	1763	100.0	100.0	100.0	100.0	100.0	100.0

The following observations and conclusions can be drawn based on the flotation tests:

- The U, Th and TREE grade of the slime in F8 was higher than the feed. The minus 38 µm fraction of the rougher tailings was grade higher than the +38 µm fraction. This indicates that the U, Th and TREE-bearing minerals were preferentially distributed in the finer fraction;

- Flotation using LR19 (62% FA2, 27% Aero 855, 9% Kerosene and 2% MIBC) at a pH range of 6.0 to 6.5 in F4 test resulted in reduced selectivity;

- Pyrite was readily removed by using Potassium Amyl Xanthate (PAX). The pyrite concentrate recovered ~91-92% S in ~5% mass. The U upgraded ~ 3 times and the TREE upgraded ~ 1.5-2.0 times in the sulphide concentrate indicating that PAX recovers U and REO since mineralogy indicates mineral association of U and TREE with sulphide;

- Cupferron and Flotinor FS-2, evaluated in test F3, showed no selectivity;

- Salicyclhydromate(SHD), evaluated in F5, resulted in low recoveries;

- Additional flotation time and collection dosage are required to achieve U rougher recovery greater than 90%. Subsequent cleaning rejects a significant amount of U;

- Primary grind size at 80% passing 135 µm in F6 versus 80 µm in F7 indicate no effect on metallurgy;

- Oxalic acid in F11 and F13 tests and sodium fluorosilicate in F12 did not improve selectivity;

- Flotation results on LR, UR and IQ Composites (tests F9, F16 and F17) were comparable to the Master Composite, and the results of variability flotation tests are summarized in Table 2;

- The change of conditioning time and air low rate in F19-F21 tests did not improve metallurgy;

- The best rougher flotation results on Master composite were achieved in test F6. The feed was ground to a P_{80} of 135 µm and it used 500 g/t sodium silicate and stage-added 1425 g/t LR19 and 26 g/t DF250 for floating 29 minutes. The rougher concentrate recovered 79% U, 92% Th and 92% TREE in 20% mass;

- Flotation only is the recommended processing route as it achieved better results than gravity or magnetic separation;

- Test F18 was performed on acid leach residues with the purpose of recovering pyrite. The sulphide concentrate grading 34.6% S with 91% recoveries in 26% mass was achieved. Pyrite removal via flotation on leach residue is possible. This can be an alternative option to pyrite flotation prior to U-Th-TREE flotation

Conclusions and Recommendations

Four composites (LR, UR, IR and Master Composite) were prepared using samples from the Elliot Lake deposit. Bond ball mill grindability, gravity separation, magnetic separation and flotation testwork were completed.

Mineralogical examination indicated that the major phases present in these composites are quartz (63.8-67.4%), K-feldspars (12.2-16.8%), micas (9.6-12.6%), pyrite (2.6-10%). Monazite, bastnaesite, allanite and U-Th phases are present in trace amounts. EMPA reveals that the main REE-bearing minerals are monazite and allanite, and the U-Th minerals include uranothorite-thorite [$(Th,U)SiO_4$], altered uraninite (UO_2), and coffinite [$U(SiO_4)_{1-x}(OH)_{4x}$]. Monazite is over 60% free and liberated at 100% passing 100 mesh (150 µm). The U-Th minerals are generally poorly liberated..

The Bond ball mill grindability test performed at 100 mesh (150 µm) identified the Master Composite as moderately hard with a metric work index of 16.2 kWh/t.

Gravity separation using a Mozley Table and a Wilfley Table, and magnetic separation with WHIMS achieved limited upgrading for U, Th and REE-bearing minerals, and would need a scavenging stage to further process the tailings for better recoveries. Gravity and magnetic separation were successful in recovering greater than 50% U, Th and TREE in a small amount of mass (10%). Additional scavenging by flotation is required.

The rougher flotation with LR19 and sodium silicate at a P_{80} of 135 µm achieved reasonable metallurgical results for the Master Composite. The F6 rougher concentrate recovered 79% U, 92% Th and 92% TREE in 20% mass.

Recommendations for future work include

- Investigate the effects of desliming;
- Consider the combination of gravity or magnetic separation and flotation;
- Optimize the LR19 dosage;
- Evaluated collector FA2 in place of LR19.

Appendix A – Detailed of Grindability Tests

SGS Minerals Services

Standard Bond Ball Mill Grindability Test

Project No.: 13849-002 Date: 4-Apr-13
Sample: Master Comp

Purpose: To determine the ball mill grindability of the sample in terms of a Bond work index number.

Procedure: The equipment and procedure duplicate the Bond method for determining ball mill work indices.

Test Conditions:
Feed 100% Passing 6 mesh
Mesh of grind: 100 mesh
Test feed weight (700 mL): 1,219 grams
Equivalent to : 1,741 kg/m³ at Minus 6 mesh
Weight % of the undersize material in the ball mill feed: 12.1%
Weight of undersize product for 250% circulating load: 348 grams

Results:
Gram per Rev Average for the Last Three Stages = **1.53 g**
Circulation load = **246%**

CALCULATION OF A BOND WORK INDEX

$$\boxed{BWI = \frac{44.5}{P1^{0.23} \times Grp^{0.82} \times \left\{ \frac{10}{\sqrt{P}} - \frac{10}{\sqrt{F}} \right\}}}$$

P1 = 100% passing size of the product	150	microns
Grp = Grams per revolution	1.53	grams
P_{80} = 80% passing size of product	124	microns
F_{80} = 80% passing size of the feed	2,002	microns

BWI = **14.7** kWh/t (imperial)

BWI = **16.2** kWh/t (metric)

Comments:

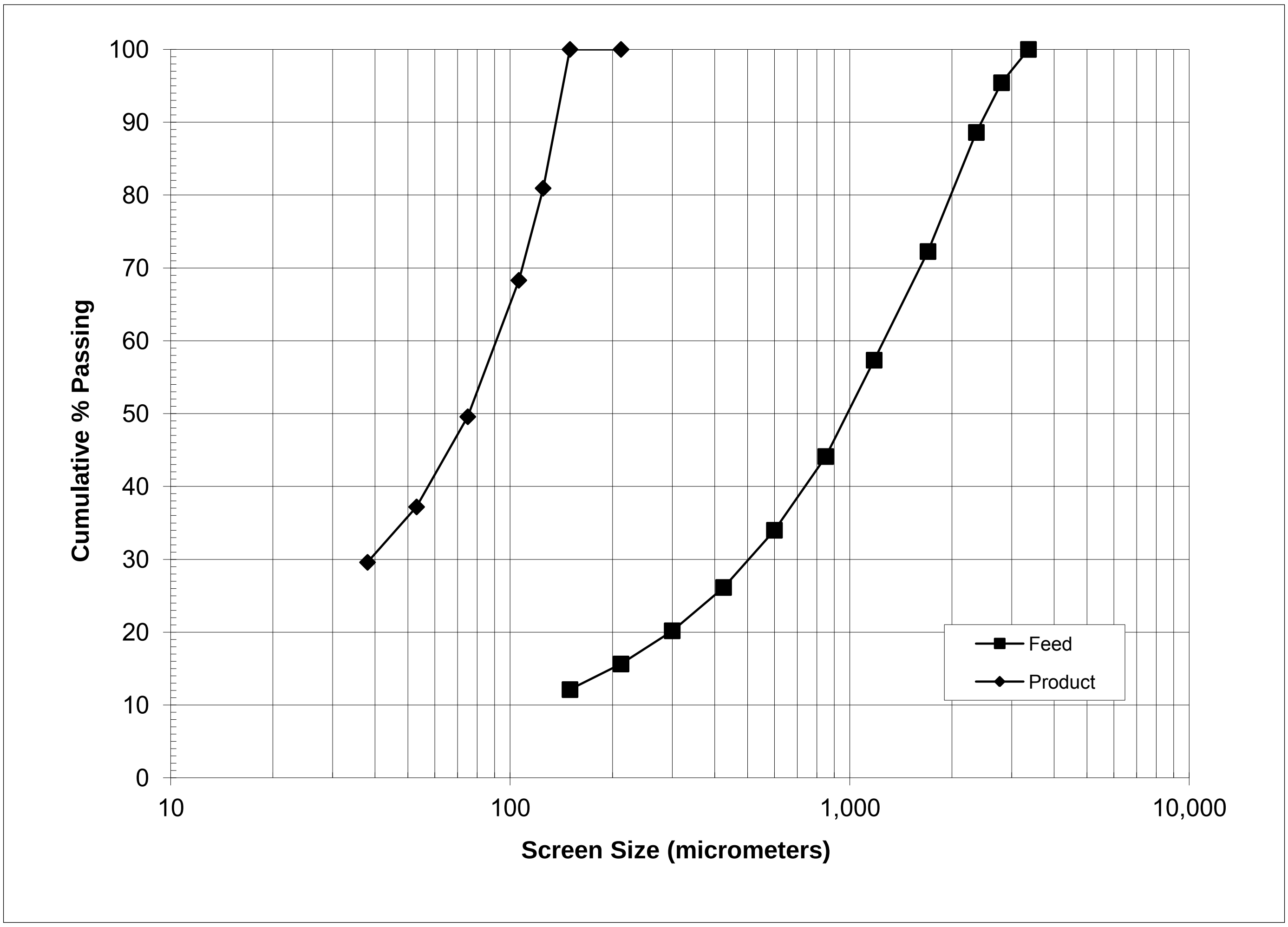
Stage No.	# of Revs	New Feed (grams)	Product in Feed (grams)	Material to Be Ground (grams)	Material Passing 100 mesh in Product (grams)	Net Ground Material (grams)	Material Ground Per Mill Rev (grams)
1	100	1,219	148	201	266	118	1.18
2	267	266	32	316	377	345	1.29
3	234	377	46	303	377	331	1.42
4	214	377	46	303	364	318	1.49
5	205	364	44	304	352	308	1.50
6	204	352	43	306	357	314	1.54
7	198	357	43	305	348	305	1.54
		Average for Last Three Stages =			352 g		1.53 g

SGS Minerals Services

Standard Bond Ball Mill Grindability Test

Project No.:	13849-002	Date:	4-Apr-13
Sample:	Master Comp		

		Feed Particle Size Analysis				Product Particle Size Analysis			
Size		Weight	% Retained		% Passing	Weight	% Retained		% Passing
Mesh	µm	grams	Individual	Cumulative	Cumulative	grams	Individual	Cumulative	Cumulative
6	3,360	0.00	0.00	0.00	100.0				
7	2,800	23.0	4.57	4.57	95.4				
8	2,360	34.3	6.82	11.4	88.6				
10	1,700	82.2	16.3	27.7	72.3				
14	1,180	75.0	14.9	42.7	57.3				
20	850	66.5	13.2	55.9	44.1				
28	600	50.9	10.1	66.0	34.0				
35	425	39.5	7.86	73.9	26.1				
48	300	29.9	5.95	79.8	20.2				
65	212	22.9	4.55	84.4	15.6	0.00	0.00	0.00	100.0
100	150	17.7	3.52	87.9	12.1	0.00	0.00	0.00	100.0
115	125					28.8	19.1	19.1	80.9
150	106					19.1	12.6	31.7	68.3
200	75					28.3	18.7	50.4	49.6
270	53					18.7	12.4	62.8	37.2
400	38					11.5	7.61	70.4	29.6
Pan	-	60.9	12.1	100.0	-	44.7	29.6	100.0	-
Total	**-**	**502.8**	**100.0**	**F_{80}:**	**2,002**	**151.1**	**100.0**	**P_{80}:**	**124**



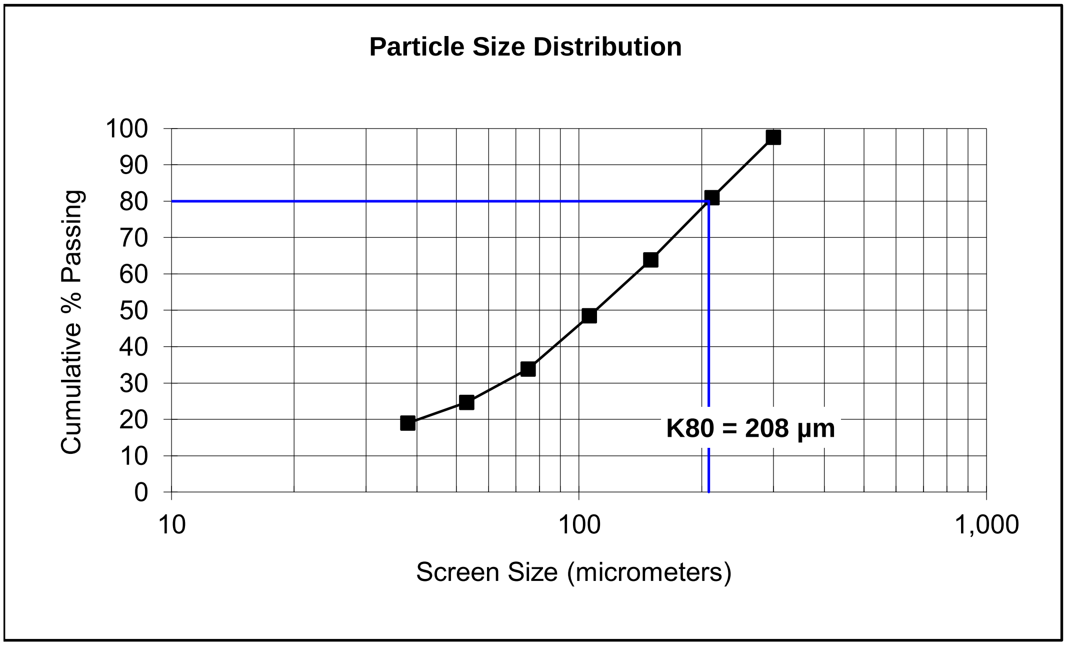
Appendix B – Detailed of Mozley Shaking Table Tests

SGS Minerals Services
Size Distribution Analysis

Project No.
13849-002

Sample: **minus 48 mesh master com** Test No.: **G1**

Size		Weight	% Retained		% Passing
Mesh	µm	grams	Individual	Cumulative	Cumulative
48	300	3.8	2.5	2.5	97.5
65	212	25.7	16.6	19.0	81.0
100	150	26.5	17.1	36.2	63.8
150	106	23.8	15.4	51.5	48.5
200	75	22.7	14.7	66.2	33.8
270	53	14.2	9.2	75.3	24.7
400	38	8.8	5.7	81.0	19.0
Pan	-38	29.4	19.0	100.0	0.0
Total	**-**	**154.9**	100.0	**-**	**-**
K80	**208**				

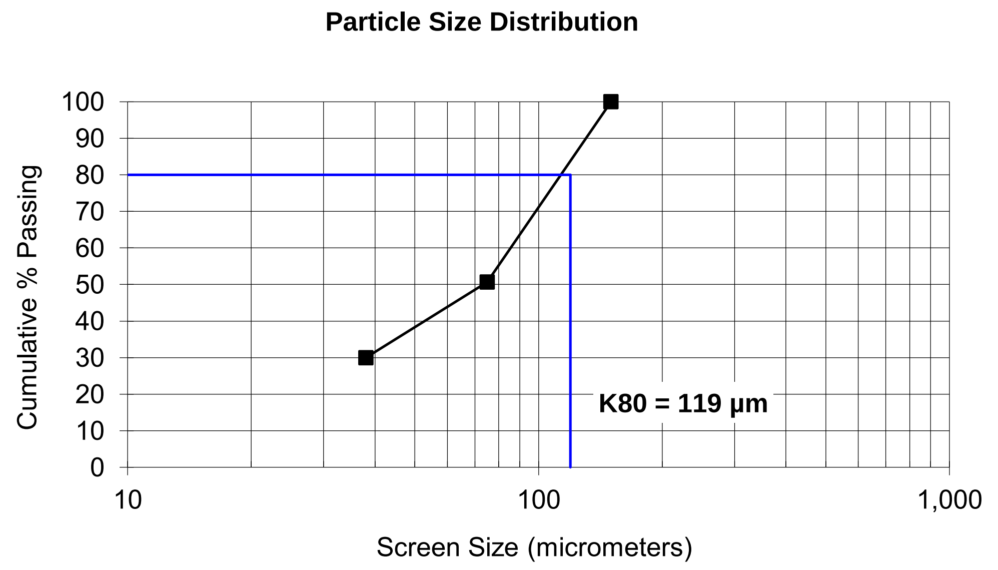


SGS Minerals Services
Size Distribution Analysis

Project No.
13849-002

Sample: **Mozley Tail** Test No.: **G-1**

Size		Weight	% Retained		% Passing
Mesh	µm	grams	Individual	Cumulative	Cumulative
100	150	0.0	0.0	0.0	100.0
200	75	76.0	49.3	49.3	50.7
400	38	31.9	20.7	70.0	30.0
Pan	-38	46.2	30.0	100.0	0.0
Total	**-**	**154.1**	100.0	**-**	**-**
K80	**119**				



Test No. : G1 **Project No:** 13849-002 **Operator**: Dan Lang **Date**: ???

Purpose: To examine gravity separation by Mozley table on 100% minus 48 mesh split sample

Procedure:

1) Grind 1 Kg Maste composite to 100% passing 48 mesh
2) Split into 250 grams and pulped and mixed test feed
3) Recorded the % solids of feed.
4) Passed feed through Mozley table and collected Conc, Middling and Tail
5) Carried out sieve analysis on the Tail with 200 mesh and 400 mesh screens
6) Submitted Conc, Middling, Tail, +75µm, -75-+38µm, -38µm for S, Au and REE ICP Scan.

Feed: 250 g of -300 µm feed sample.

Mozley Feed K_{80} = 208 µm

Definitions:

LREE=La,Ce,Pr,Nd,Sm,Eu,Sc
HREE=Gd,Tb,Dy,Ho,Er,Tm,Yb,Lu,Y
TREE=LREE+HREE

Metallurgical Balance

Products	Weight		Assays, g/t																				
	g	%	S, %	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
Conc1	6.2	2.4	44.1	4920	9060	938	3080	489	18.0	295	29.4	141	19.8	48.9	6.4	35.7	5.2	535	<25	1670	3600	18530.0	1116.4	19646.4
Conc2	10.6	4.2	23.2	4050	7520	763	2490	412	16.5	271	31.3	152	24.2	58.9	8.0	44.4	6.4	610	<25	2640	3530	15276.5	1206.2	16482.7
Middlings	75.2	29.6	0.25	201	371	35.3	120	19.7	1.0	11.5	0.5	6.4	<0.3	1.7	0.6	1.6	0.5	34	<25	145	147	773.0	57.1	830.1
Tail +75um	79.8	31.4	0.11	78.6	137	12.1	49.9	6.1	0.8	3.4	<0.5	2.0	<0.3	<0.5	0.4	<0.5	<0.5	15	<25	53.7	53.4	309.5	23.1	332.6
Tail -75 +38um	33.5	13.2	0.16	133	249	24.4	86.9	14.9	1.3	12	0.8	8.0	<0.3	2.30	0.5	1.5	<0.5	25	<25	108	91.8	534.5	50.9	585.4
Tail -38um	48.5	19.1	1.32	500	936	91.6	326	49.8	2.4	33.6	2.9	19.2	1.9	6.20	1.2	5.4	1.1	85	<25	355	406	1930.8	156.5	2087.3
Head (Calc.)	253.9	100.0	2.43	486.6	900.0	89.8	304.2	48.4	2.31	31.0	3.0	17.0	2.08	5.80	1.1	4.6	1.0	72.9	<25	293.0	385.3	1856.2	138.4	1994.6
Head (Direct)			1.92	407.0	767.0	81.0	254.0	42.6	1.90	26.1	3.3	16.5	2.60	6.70	0.80	5.20	0.70	62.0	< 25	270.0	354.0	1578.5	123.9	1702.4

Products	Weight %	% Distribution																					
		S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
Conc1	2.4	44.4	24.7	24.6	25.5	24.7	24.7	19.1	23.2	24.0	20.2	23.2	20.6	14.3	19.0	13.0	17.9	2.4	13.9	22.8	24.4	19.7	24.1
Conc2	4.2	39.9	34.7	34.9	35.5	34.2	35.6	29.9	36.5	43.7	37.2	48.6	42.4	30.7	40.4	27.4	34.9	4.2	37.6	38.2	34.4	36.4	34.5
Midds	29.6	3.05	12.2	12.2	11.6	11.7	12.1	12.8	11.0	5.0	11.1	4.3	8.7	16.3	10.3	15.2	13.8	29.6	14.7	11.3	12.3	12.2	12.3
Tails +75um	31.4	1.43	5.1	4.8	4.2	5.2	4.0	10.9	3.4	5.3	3.7	4.5	2.7	11.5	3.4	16.1	6.5	31.4	5.8	4.4	5.2	5.2	5.2
Tails -75 +38um	13.2	0.87	3.6	3.7	3.6	3.8	4.1	7.4	5.1	3.5	6.2	1.9	5.2	6.1	4.3	6.8	4.5	13.2	4.9	3.1	3.8	4.9	3.9
Tails -38um	19.1	10.4	19.6	19.9	19.5	20.5	19.7	19.9	20.7	18.5	21.5	17.5	20.4	21.1	22.5	21.6	22.3	19.1	23.2	20.1	19.9	21.6	20.0
Head (Calc.)	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0

Cumulative Products

Products	Weight %	% Distribution																					
		S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
Conc1	2.4	44.1	4920.0	9060.0	938.0	3080.0	489.0	18.0	295.0	29.4	141.0	19.8	48.9	6.4	35.7	5.2	535.0	25.0	1670.0	3600.0	18530.0	1116.4	19646.4
Conc1-2	6.6	30.9	4371.1	8088.3	827.6	2707.7	440.4	17.1	279.9	30.6	147.9	22.6	55.2	7.4	41.2	6.0	582.3	25.0	2282.0	3555.8	16477.2	1173.1	17650.3
Conc+Midds	36.2	5.85	962.5	1780.3	180.0	592.5	96.5	3.9	60.5	6.0	32.2	4.4	11.5	1.8	8.8	1.5	134.1	25.0	535.2	769.5	3640.7	260.9	3901.6
Conc+Midds+Tails -38µm	55.4	4.29	802.8	1488.7	149.5	500.5	80.4	3.4	51.2	4.9	27.7	3.5	9.7	1.6	7.6	1.4	117.2	25.0	473.0	643.9	3050.2	224.8	3275.0
Tails (+38µm)	44.6	0.12	94.7	170.1	15.7	60.8	8.7	0.9	5.9	0.6	3.8	0.3	1.0	0.4	0.8	0.5	18.0	25.0	69.8	64.8	376.0	31.3	407.3
Tails	63.8	0.48	216.2	399.7	38.5	140.3	21.0	1.4	14.2	1.3	8.4	0.8	2.6	0.7	2.2	0.7	38.1	25.0	155.3	167.1	842.2	68.8	911.0

Products	% Distribution																					
	S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
Conc1	44.4	24.7	24.6	25.5	24.7	24.7	19.1	23.2	24.0	20.2	23.2	20.6	14.3	19.0	13.0	17.9	2.4	13.9	22.8	24.4	19.7	24.1
Conc1-2	84.3	59.4	59.5	61.0	58.9	60.2	48.9	59.7	67.7	57.4	71.8	63.0	45.0	59.4	40.4	52.9	6.6	51.5	61.1	58.7	56.1	58.6
Conc+Midds	87.3	71.7	71.7	72.7	70.6	72.3	61.8	70.7	72.7	68.6	76.1	71.6	61.3	69.7	55.6	66.7	36.2	66.2	72.4	71.1	68.3	70.9
Conc+Midds+Tails -38µm	97.7	91.3	91.6	92.2	91.1	92.0	81.7	91.4	91.2	90.1	93.6	92.1	82.4	92.3	77.1	89.0	55.4	89.4	92.5	91.0	89.9	90.9
Tails (+38µm)	2.3	8.7	8.4	7.8	8.9	8.0	18.3	8.6	8.8	9.9	6.4	7.9	17.6	7.7	22.9	11.0	44.6	10.6	7.5	9.0	10.1	9.1
Tails	12.7	28.3	28.3	27.3	29.4	27.7	38.2	29.3	27.3	31.4	23.9	28.4	38.7	30.3	44.4	33.3	63.8	33.8	27.6	28.9	31.7	29.1

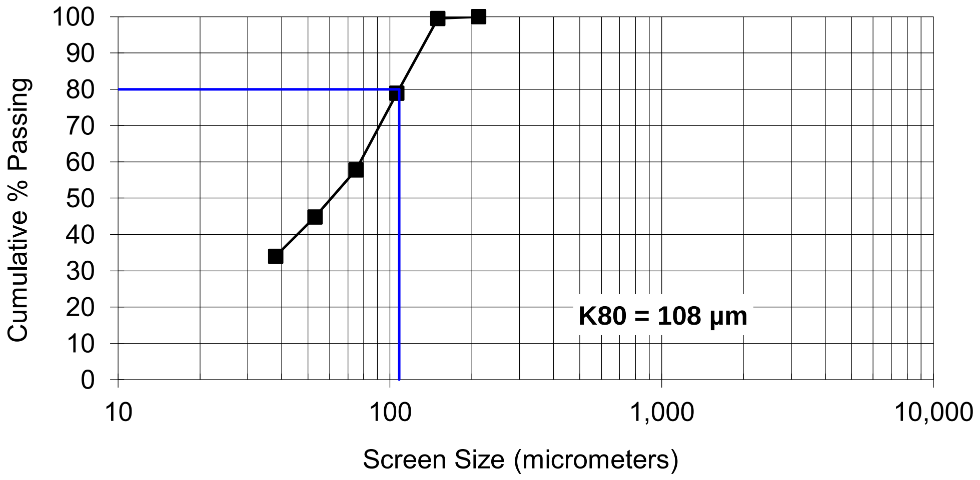
SGS Minerals Services
Size Distribution Analysis

Project No.
13849-002

Sample: **minus 100 master MC** Test No.: **G2**

Size		Weight	% Retained		% Passing
Mesh	µm	grams	Individual	Cumulative	Cumulative
65	212	0.0	0.0	0.0	100.0
100	150	0.8	0.5	0.5	99.5
150	106	31.9	20.6	21.1	78.9
200	75	32.8	21.1	42.2	57.8
270	53	20.1	13.0	55.2	44.8
400	38	16.8	10.8	66.0	34.0
Pan	-38	52.7	34.0	100.0	0.0
Total	**-**	**155.1**	100.0	**-**	**-**
K80	**108**				

Particle Size Distribution

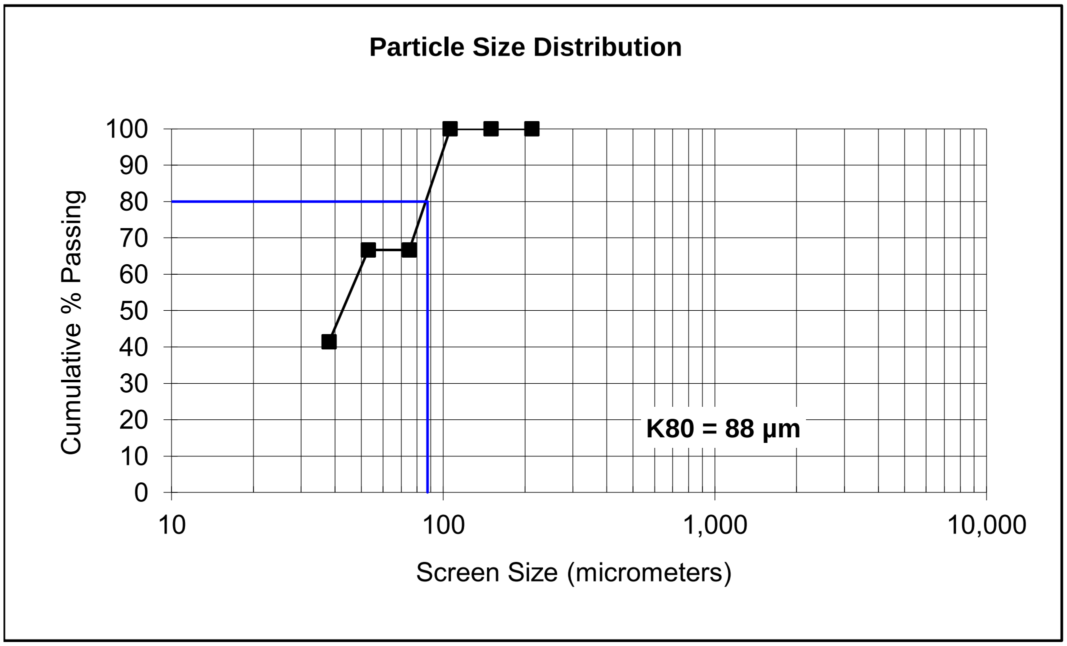


SGS Minerals Services
Size Distribution Analysis

Project No.
13849-002

Sample: **Mozley Tail** Test No.: **G-2**

Size		Weight	% Retained		% Passing
Mesh	µm	grams	Individual	Cumulative	Cumulative
65	212	0.0	0.0	0.0	100.0
100	150	0.0	0.0	0.0	100.0
150	106	0.0	0.0	0.0	100.0
200	75	64.0	33.3	33.3	66.7
270	53	0.0	0.0	33.3	66.7
400	38	48.5	25.2	58.6	41.4
Pan	-38	79.6	41.4	100.0	0.0
Total	**-**	**192.1**	100.0	**-**	**-**
K80	**88**				



Test No. : G2 **Project No:** 13849-002 **Operator**: Dan Lang **Date**: ???

Purpose: To examine gravity separation by Mozley table on 100% minus 100 mesh split sample

Procedure:

1) Grind 1 Kg Maste composite to 100% passing 100 mesh
2) Split into 250 grams and pulped and mixed test feed
3) Recorded the % solids of feed.
4) Passed feed through Mozley table and collected Conc, Middling and Tail
5) Carried out sieve analysis on the Tail with 200 mesh and 400 mesh screens
6) Submitted Conc, Middling, Tail, +75μm, -75-+38μm, -38μm for S, Au and REE ICP Scan.

Feed: 250 g of -150 μm feed sample.

Mozley Feed K_{80} = 108 μm

Definitions:

LREE=La,Ce,Pr,Nd,Sm,Eu,Sc
HREE=Gd,Tb,Dy,Ho,Er,Tm,Yb,Lu,Y
TREE=LREE+HREE

Metallurgical Balance

Products	Weight		Assays, g/t																				
	g	%	S, %	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
Conc1	5.6	2.2	44.5	4900	9250	911	3230	456	16.3	280	29.5	122	18.6	44.4	5.6	31.9	4.4	480	<50	1320	3340	18813.3	1016.4	19829.7
Conc2	9.5	3.7	32.8	5180	9740	906	3290	501	19.7	332	40.0	187	29.4	72.8	9.5	53.4	7.4	760	<50	2440	4830	19686.7	1491.5	21178.2
Middling	44.1	17.0	0.45	230	423	42.5	135	22.1	1.1	15.6	1.8	10.5	1.6	3.7	0.5	3.4	<0.5	33	<50	167	192	903.7	70.6	974.3
Tails +75um	66.5	25.7	0.10	88	160	16.0	54	10.5	0.6	6.5	0.7	4.5	0.7	1.9	<0.3	1.5	<0.5	17	<50	68	66	379.1	33.6	412.7
Tails -75 +38um	50.4	19.5	0.23	116	213	20.7	69	13.0	0.6	8.4	0.9	6.1	1.0	2.6	0.3	2.0	<0.5	24	<50	99.9	99.5	482.3	45.8	528.1
Tails -38um	82.7	32.0	1.13	395	697	69.8	222	41.3	2.0	25.5	3.3	16.7	2.5	6.3	0.8	4.8	0.7	59	<50	250	319	1477.1	119.6	1596.7
Head (Calc.)	258.9	100.0	2.67	506.7	934.9	90.6	311.9	50.4	2.17	32.4	3.82	19.0	2.93	7.27	0.95	5.54	0.90	71.8	<50	263.4	420.6	1946.7	144.5	2091.2
Head (Direct)			1.92	407.0	767.0	81.0	254.0	42.6	1.90	26.1	3.3	16.5	2.60	6.70	0.80	5.20	0.70	62.0	< 25	270.0	354.0	1578.5	123.9	1702.4

Products	Weight %	% Distribution																				
		S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
Conc1	2.2	36.0	20.9	21.4	21.7	22.4	19.6	16.2	18.7	16.7	13.9	13.7	13.2	12.8	12.5	10.6	14.5	2.2	10.8	17.2	20.9	15.2	20.5
Conc2	3.7	45.0	37.5	38.2	36.7	38.7	36.4	33.3	37.7	38.4	36.2	36.9	36.7	36.8	35.4	30.1	38.8	3.7	34.0	42.1	37.1	37.9	37.2
Middling	17.0	2.9	7.7	7.7	8.0	7.4	7.5	8.6	8.2	8.0	9.4	9.3	8.7	9.0	10.5	9.4	7.8	17.0	10.8	7.8	7.9	8.3	7.9
Tails +75um	25.7	1.0	4.5	4.4	4.5	4.4	5.3	7.1	5.2	4.7	6.1	6.1	6.7	8.1	7.0	14.3	6.1	25.7	6.6	4.1	5.0	6.0	5.1
Tails -75 +38um	19.5	1.7	4.5	4.4	4.4	4.3	5.0	5.4	5.1	4.6	6.3	6.7	7.0	6.2	7.0	10.8	6.5	19.5	7.4	4.6	4.8	6.2	4.9
Tails -38um	32.0	13.5	24.9	23.8	24.6	22.8	26.2	29.4	25.2	27.6	28.1	27.3	27.7	27.0	27.7	24.8	26.3	32.0	30.3	24.2	24.3	26.5	24.4
Head (Calc.)	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0

Cumulative Products

Products	Weight %	% Distribution																				
		S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
Conc1	2.2	44.5	4900.0	9250.0	911.0	3230.0	456.0	16.3	280.0	29.5	122.0	18.6	44.4	5.6	31.9	4.4	480.0	50.0	1320.0	3340.0	18813.3	1016.4	19829.7
Conc1-2	5.8	37.1	5076.2	9558.3	907.9	3267.7	484.3	18.4	312.7	36.1	162.9	25.4	62.3	8.1	45.4	6.3	656.2	50.0	2024.6	4277.4	19362.8	1315.3	20678.1
Conc+Midds	22.9	9.81	1466.1	2753.1	263.2	934.1	140.0	5.5	91.4	10.6	49.4	7.7	18.6	2.4	14.1	2.0	191.9	50.0	640.8	1234.1	5612.0	388.1	6000.1
Conc+Midds+Tails -38μm	54.8	4.75	841.7	1554.5	150.5	519.0	82.5	3.5	53.0	6.3	30.3	4.7	11.4	1.5	8.7	1.2	114.4	50.0	413.0	700.6	3201.6	231.6	3433.1
Tails (+38μm)	45.2	0.16	100.1	182.8	18.0	60.5	11.6	0.6	7.3	0.8	5.2	0.8	2.2	0.3	1.7	0.5	20.0	50.0	81.8	80.7	423.6	38.9	462.5
Tails	77.1	0.56	222.3	395.9	39.5	127.4	23.9	1.2	14.9	1.8	10.0	1.5	3.9	0.5	3.0	0.6	36.2	50.0	151.5	179.4	860.1	72.3	932.4

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Products	% Distribution																					
	S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
Conc1	36.0	20.9	21.4	21.7	22.4	19.6	16.2	18.7	16.7	13.9	13.7	13.2	12.8	12.5	10.6	14.5	2.2	10.8	17.2	20.9	15.2	20.5
Conc1-2	81.0	58.4	59.6	58.4	61.1	56.0	49.5	56.4	55.1	50.1	50.6	50.0	49.6	47.8	40.7	53.3	5.8	44.8	59.3	58.0	53.1	57.7
Conc+Midds	83.9	66.2	67.3	66.4	68.5	63.5	58.1	64.6	63.1	59.5	59.9	58.6	58.6	58.3	50.1	61.1	22.9	55.6	67.1	65.9	61.4	65.6
Conc+Midds+Tails -38μm	97.4	91.1	91.2	91.0	91.2	89.6	87.5	89.8	90.7	87.6	87.2	86.3	85.7	86.0	74.9	87.4	54.8	86.0	91.3	90.2	87.9	90.0
Tails (+38μm)	2.6	8.9	8.8	9.0	8.8	10.4	12.5	10.2	9.3	12.4	12.8	13.7	14.3	14.0	25.1	12.6	45.2	14.0	8.7	9.8	12.1	10.0
Tails	16.1	33.8	32.7	33.6	31.5	36.5	41.9	35.4	36.9	40.5	40.1	41.4	41.4	41.7	49.9	38.9	77.1	44.4	32.9	34.1	38.6	34.4

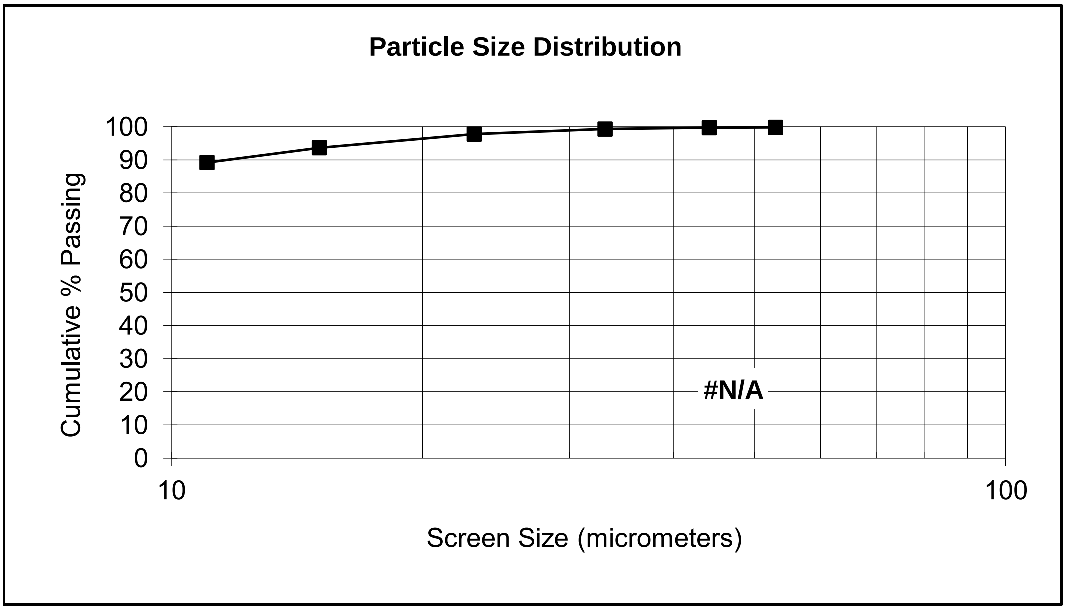
Appendix C – Detailed of Wilfley Table Tests

SGS Minerals Services
Size Distribution Analysis

Project No.
13849-002

Sample: **Slime 1** Test No.: **W1**

Dry Solids S.G.=		**2.85**	Water Temperature =		**9.00 Cº**
Size		Weight	% Retained		% Passing
Mesh	µm	grams	Individual	Cumulative	Cumulative
270	53	0.1	0.2	0.2	99.8
	44	0.1	0.1	0.3	99.7
	33	0.2	0.4	0.7	99.3
	23	0.8	1.5	2.2	97.8
	15	2.1	4.1	6.3	93.7
	11	2.2	4.5	10.8	89.2
	-11	44.7	89.2	100.0	0.0
Total	**-**	**50.1**	100.0	**-**	**-**
K80	**#N/A**				



Gekko 296A **Project No:** 13849-002 **Operator**: Dan Lang **Date**: Dec06,2012

Purpose: To examine gravity separation by Shaking Table on progressive grind sample

Procedure: As outlined in part A of Gekko 296 Procedure

Feed: 10Kg of -6Mesh Master Composite sample

Grind: Grinding feed to 100% passing 20mesh, 1st tails to 35mesh, 2nd tails to 140mesh

Definitions: LREE=La,Ce,Pr,Nd,Sm,Eu,Sc
HREE=Gd,Tb,Dy,Ho,Er,Tm,Yb,Lu,Y
TREE=LREE+HREE

3rd Table Fd K_{80} = µm

Metallurgical Balance

Products	**Weight**		**Assays, g/t**																				
	g	**%**	**S, %**	**La**	**Ce**	**Pr**	**Nd**	**Sm**	**Eu**	**Gd**	**Tb**	**Dy**	**Ho**	**Er**	**Tm**	**Yb**	**Lu**	**Y**	**Sc**	**U**	**Th**	**LREE**	**HREE**	**TREE**
W Conc-1	3005.4	19.9	8.32	1190	2160	211	685	121	4.8	72.5	9.6	44.4	6.9	17.1	2.1	12.7	1.8	166	<50	608	1010	4421.8	333.1	4754.9
W Conc-2	1367.7	9.1	1.69	745	1320	135	429	70.8	3.1	44.7	6	27.1	4.3	10.5	1.3	8.4	1.2	107	<25	393	617	2727.9	210.5	2938.4
W Conc-3	608.1	4.0	1.31	564	1060	102	358	59.3	3.2	37.9	4.6	20.8	3.6	7.4	1.3	7.6	1.1	87	<100	274	442	2246.5	171.3	2417.8
W Slime	373.0	2.5	0.75	466	826	84	271	46	2.5	30	4.5	20.9	3.4	8.5	1.1	7.1	0.9	88	<25	349	375	1720.5	164.4	1884.9
W Tails	9737.6	64.5	0.18	121	217	20.4	66	11.3	1	8.2	1.4	5.6	1.1	2.2	0.5	2.2	<0.5	<40	<100	80.9	87.3	536.7	61.7	598.4
Head (Calc.)	15091.8	100.0	2.00	416.8	752.9	73.6	239.0	41.3	2.07	26.0	3.66	16.3	2.70	6.28	0.94	5.19	0.86	74.2	81.4	228.6	340.5	1607.1	136.2	1743.3
Head (Direct)			1.92	407.0	767.0	81.0	254.0	42.6	1.90	26.1	3.3	16.5	2.60	6.70	0.80	5.20	0.70	62.0	< 25	270.0	354.0	1578.5	123.9	1702.4

Products	**Weight**	**% Distribution**																				
	%	**S**	**La**	**Ce**	**Pr**	**Nd**	**Sm**	**Eu**	**Gd**	**Tb**	**Dy**	**Ho**	**Er**	**Tm**	**Yb**	**Lu**	**Y**	**Sc**	**U**	**Th**	**LREE**	**HREE**	**TREE**
W Conc-1	19.9	82.9	56.9	57.1	57.1	57.1	58.3	46.1	55.4	52.3	54.4	50.8	54.2	44.6	48.7	41.9	44.5	12.2	53.0	59.1	54.8	48.7	54.3
W Conc-2	9.1	7.7	16.2	15.9	16.6	16.3	15.5	13.6	15.6	14.9	15.1	14.4	15.1	12.6	14.7	12.7	13.1	2.8	15.6	16.4	15.4	14.0	15.3
W Conc-3	4.0	2.6	5.5	5.7	5.6	6.0	5.8	6.2	5.9	5.1	5.2	5.4	4.7	5.6	5.9	5.2	4.7	5.0	4.8	5.2	5.6	5.1	5.6
W Slime	2.5	0.9	2.8	2.7	2.8	2.8	2.8	3.0	2.8	3.0	3.2	3.1	3.3	2.9	3.4	2.6	2.9	0.8	3.8	2.7	2.6	3.0	2.7
W Tails	64.5	5.8	18.7	18.6	17.9	17.8	17.6	31.1	20.3	24.7	22.2	26.3	22.6	34.4	27.3	37.7	34.8	79.3	22.8	16.5	21.5	29.2	22.1
Head (Calc.)	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0

Cumulative Products

Products	**Weight**	**% Distribution**																				
	%	**S**	**La**	**Ce**	**Pr**	**Nd**	**Sm**	**Eu**	**Gd**	**Tb**	**Dy**	**Ho**	**Er**	**Tm**	**Yb**	**Lu**	**Y**	**Sc**	**U**	**Th**	**LREE**	**HREE**	**TREE**
W Conc-1	19.9	8.32	1190.0	2160.0	211.0	685.0	121.0	4.8	72.5	9.6	44.4	6.9	17.1	2.1	12.7	1.8	166.0	50.0	608.0	1010.0	4421.8	333.1	4754.9
W Conc 1-2	29.0	6.25	1050.8	1897.3	187.2	604.9	105.3	4.3	63.8	8.5	39.0	6.1	15.0	1.8	11.4	1.6	147.5	42.2	540.8	887.1	3892.0	294.8	4186.8
W Conc 1-3	33.0	5.64	991.4	1795.1	176.8	574.8	99.7	4.1	60.6	8.0	36.8	5.8	14.1	1.8	10.9	1.5	140.2	49.2	508.2	832.8	3691.1	279.7	3970.8
W Conc1-3 + Slime	35.5	5.30	954.8	1727.6	170.4	553.6	95.9	4.0	58.5	7.8	35.7	5.6	13.7	1.7	10.6	1.5	136.5	47.6	497.1	800.9	3553.9	271.7	3825.5
W Tails	64.5	0.18	121.0	217.0	20.4	66.0	11.3	1.0	8.2	1.4	5.6	1.1	2.2	0.5	2.2	0.5	40.0	100.0	80.9	87.3	536.7	61.7	598.4
Head (Calc.)	100.0	2.00	416.8	752.9	73.6	239.0	41.3	2.1	26.0	3.7	16.3	2.7	6.3	0.9	5.2	0.9	74.2	81.4	228.6	340.5	1607.1	136.2	1743.3

Products	**% Distribution**																				
	S	**La**	**Ce**	**Pr**	**Nd**	**Sm**	**Eu**	**Gd**	**Tb**	**Dy**	**Ho**	**Er**	**Tm**	**Yb**	**Lu**	**Y**	**Sc**	**U**	**Th**	**LREE**	**HREE**	**TREE**
W Conc-1	82.9	56.9	57.1	57.1	57.1	58.3	46.1	55.4	52.3	54.4	50.8	54.2	44.6	48.7	41.9	44.5	12.2	53.0	59.1	54.8	48.7	54.3
W Conc 1-2	90.6	73.1	73.0	73.7	73.3	73.8	59.7	71.0	67.2	69.5	65.3	69.3	57.1	63.4	54.6	57.6	15.0	68.6	75.5	70.2	62.7	69.6
W Conc 1-3	93.3	78.5	78.7	79.3	79.4	79.6	65.9	76.8	72.2	74.6	70.6	74.1	62.7	69.3	59.7	62.3	20.0	73.4	80.7	75.8	67.8	75.2
W Conc1-3 + Slime	94.2	81.3	81.4	82.1	82.2	82.4	68.9	79.7	75.3	77.8	73.7	77.4	65.6	72.7	62.3	65.2	20.7	77.2	83.5	78.5	70.8	77.9
W Tails	5.8	18.7	18.6	17.9	17.8	17.6	31.1	20.3	24.7	22.2	26.3	22.6	34.4	27.3	37.7	34.8	79.3	22.8	16.5	21.5	29.2	22.1
Head (Calc.)	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0

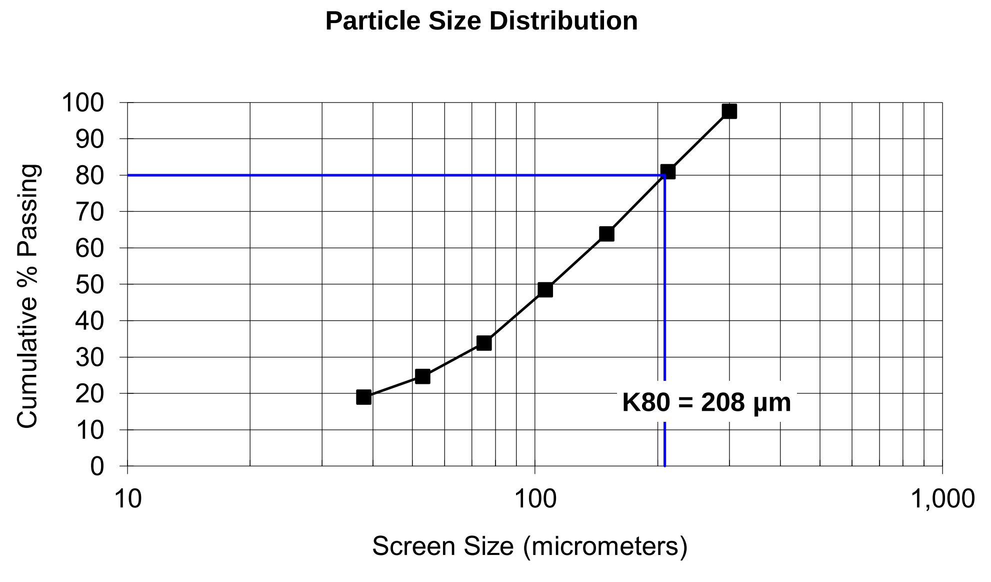
Appendix D – Detailed of WHIMS Tests

SGS Minerals Services
Size Distribution Analysis

Project No.
13849-002

Sample: **minus 48 mesh master com** Test No.: **M-1**

Size		Weight	% Retained		% Passing
Mesh	µm	grams	Individual	Cumulative	Cumulative
48	300	3.8	2.5	2.5	97.5
65	212	25.7	16.6	19.0	81.0
100	150	26.5	17.1	36.2	63.8
150	106	23.8	15.4	51.5	48.5
200	75	22.7	14.7	66.2	33.8
270	53	14.2	9.2	75.3	24.7
400	38	8.8	5.7	81.0	19.0
Pan	-38	29.4	19.0	100.0	0.0
Total	**-**	**154.9**	100.0	**-**	**-**
K80	**208**				



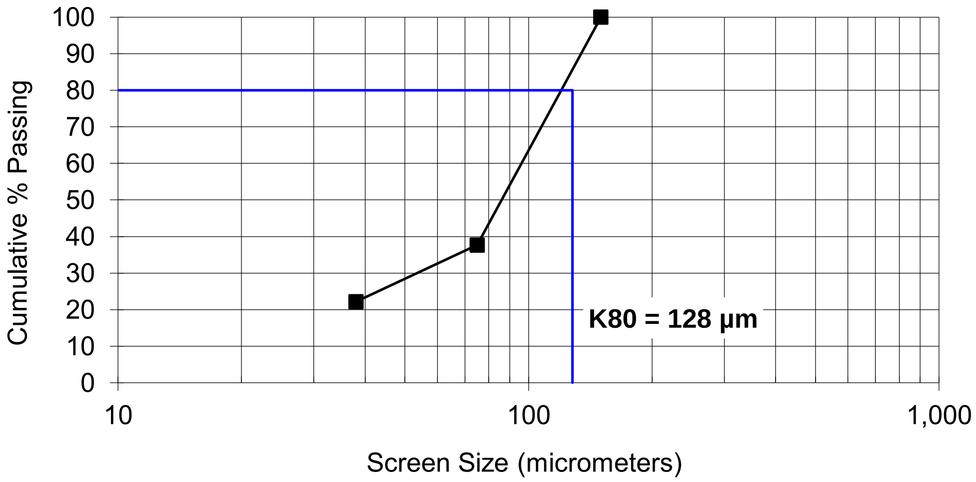
SGS Minerals Services
Size Distribution Analysis

Project No.
13849-002

Sample: **20A Non Mag** Test No.: **M-1**

Size		Weight	% Retained		% Passing
Mesh	µm	grams	Individual	Cumulative	Cumulative
100	150	0.0	0.0	0.0	100.0
200	75	67.2	62.3	62.3	37.7
400	38	16.7	15.5	77.8	22.2
Pan	-38	23.9	22.2	100.0	0.0
Total	**-**	**107.8**	100.0	**-**	**-**
K80	**128**				

Particle Size Distribution



Test No.: M-1 **Project No:** 13849-002 **Operator:** Dan Lang **Date:** Dec05,2012

Purpose: To examine magnetic separation by WHIMS on 100% minus 48 mesh split sample

Procedure:

1) Grind 1 Kg Maste composite to 100% passing 48 mesh
2) Split into 250 grams and pulped and mixed test feed
3) Recorded the % solids of feed.
4) Passed feed through WHIMS at increasing intensities of 2, 5, 10, 20 A
5) A total of 5 products: 2A mag, 5A mag, 10A mag, 20A mag, and 20A non-mag were generated.
6) Submitted all samples for S and REE ICP Scan.

Feed: 250 g of -300 µm feed sample. Pulped and mixed ~250g of -300 µm feed sample. % Solids =

Grind: Stage-grinding to 100% passing 48mesh WHIMS Fd K_{80} = 208 µm

Definitions:
LREE=La,Ce,Pr,Nd,Sm,Eu,Sc
HREE=Gd,Tb,Dy,Ho,Er,Tm,Yb,Lu,Y
TREE=LREE+HREE

Metallurgical Balance

Products	Weight		Assays, g/t																				
	g	%	S, %	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
0-2A WHIMS Mag	5.6	2.2	6.08	2620	4730	483	1570	267	9.9	165	18.4	95.2	14.1	35.3	4.7	25.1	3.5	364	<25	1540	2140	9705	725	10430
2-5A WHIMS Mag	5.9	2.3	5.96	6560	12200	1240	4030	621	22.5	370	38.6	181	26.7	63.0	8.7	44.0	6.0	675	<25	2340	4270	24699	1413	26112
5-10A WHIMS Mag	5.3	2.1	6.74	10500	19500	1970	6400	1010	37.4	611	62.2	299	45.1	106	13.3	72.3	10.0	1110	<25	3680	7790	39442	2329	41771
10-20A WHIMS Mag	5.3	2.1	6.10	5830	10900	1100	3600	589	21.4	377	42.9	200	31.1	75.0	9.8	52.7	7.2	782	<25	3010	5010	22065	1578	23643
Non-Mag +75µm	145.5	56.9	1.31	144	263	24.8	91.1	14.7	<0.3	8.7	<0.5	5.3	<0.3	1.3	0.5	1.1	<0.5	27	<25	132	128	562.9	45.2	608.1
Non-Mag -75+38µm	36.2	14.2	1.79	97.4	181	16.9	61.4	8.8	<0.3	6.8	<0.5	5.3	<0.3	0.9	0.4	1.0	<0.5	26	<25	129	118	390.8	41.7	432.5
Non-Mag -38µm	51.7	20.3	1.61	255	478	46.0	153	26.6	<0.3	17.3	1.2	10.6	0.9	3.8	<0.3	2.8	0.6	53	<25	249	263	983.9	90.5	1074.4
Head (Calc.)	255.5	100.0	1.86	695.1	1288.2	128.7	426.5	68.4	2.2	42.1	4.1	22.5	2.9	7.6	1.2	5.5	1.0	92.6	25.0	370.4	553.9	2634.1	179.5	2813.6
Head (Direct)			1.92	407.0	767.0	81.0	254.0	42.6	1.9	26.1	3.3	16.5	2.6	6.7	0.8	5.2	0.7	62.0	< 25	270.0	354.0	1578.5	123.9	1702.4

Products	Weight %	% Distribution																					
		S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
0-2A WHIMS Mag	2.2	7.2	8.3	8.0	8.2	8.1	8.6	9.7	8.6	9.9	9.3	10.6	10.2	8.7	10.0	7.3	8.6	2.2	9.1	8.5	8.1	8.9	8.1
2-5A WHIMS Mag	2.3	7.4	21.8	21.9	22.2	21.8	21.0	23.3	20.3	21.9	18.5	21.2	19.1	17.0	18.5	13.2	16.8	2.3	14.6	17.8	21.7	18.2	21.4
5-10A WHIMS Mag	2.1	7.5	31.3	31.4	31.7	31.1	30.6	34.8	30.1	31.7	27.5	32.2	28.9	23.3	27.3	19.8	24.9	2.1	20.6	29.2	31.1	26.9	30.8
10-20A WHIMS Mag	2.1	6.8	17.4	17.6	17.7	17.5	17.9	19.9	18.6	21.8	18.4	22.2	20.4	17.2	19.9	14.2	17.5	2.1	16.9	18.8	17.4	18.2	17.4
Non-Mag +75µm	56.9	40.1	11.8	11.6	11.0	12.2	12.2	7.7	11.8	7.0	13.4	5.9	9.7	24.0	11.4	27.1	16.6	56.9	20.3	13.2	12.2	14.3	12.3
Non-Mag -75+38µm	14.2	13.6	2.0	2.0	1.9	2.0	1.8	1.9	2.3	1.7	3.3	1.5	1.7	4.8	2.6	6.7	4.0	14.2	4.9	3.0	2.1	3.3	2.2
Non-Mag -38µm	20.3	17.5	7.4	7.5	7.2	7.3	7.9	2.7	8.3	6.0	9.5	6.3	10.1	5.1	10.3	11.6	11.6	20.3	13.6	9.6	7.6	10.2	7.7
Head (Calc.)	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0

Cumulative Products

Products	Weight %	Assays, g/t																					
		S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
0-2A WHIMS Mag	2.2	6.08	2620	4730	483	1570	267	9.9	165.0	18.4	95.2	14.1	35.3	4.70	25.1	3.50	364.0	<25	1540	2140	9705	725	10430
0-5A WHIMS Mag	4.5	6.02	4641	8562	871	2832	449	16.4	270.2	28.8	139.2	20.6	49.5	6.75	34.8	4.78	523.6	<25	1950	3233	17397	1078	18475
0-10A WHIMS Mag	6.6	6.25	6490	12013	1218	3958	626	23.0	377.7	39.3	189.6	28.3	67.3	8.82	46.6	6.43	708.6	<25	2496	4670	24352	1473	25825
0-20A WHIMS Mag	8.6	6.21	6331	11746	1190	3872	617	22.6	377.5	40.2	192.1	29.0	69.2	9.05	48.1	6.61	726.2	<25	2619	4752	23804	1498	25302
0-20A Mag+ Non-Mag -38µm	28.9	2.99	2073	3850	388	1266	203	7.0	125.1	12.9	64.9	9.3	23.4	2.92	16.4	2.40	254.5	<25	958	1606	7813	512	8325
20A WHIMS Non-Mag +38µm	71.1	1.41	135	247	23	85	14	0.3	8.3	0.5	5.3	0.3	1.2	0.48	1.1	0.50	26.8	<25	131	126	529	45	573
Head (Calc.)	100.0	1.86	695.1	1288.2	128.7	426.5	68.4	2.2	42.1	4.1	22.5	2.9	7.6	1.2	5.5	1.0	92.6	<25	370.4	553.9	2634.1	179.5	2813.6

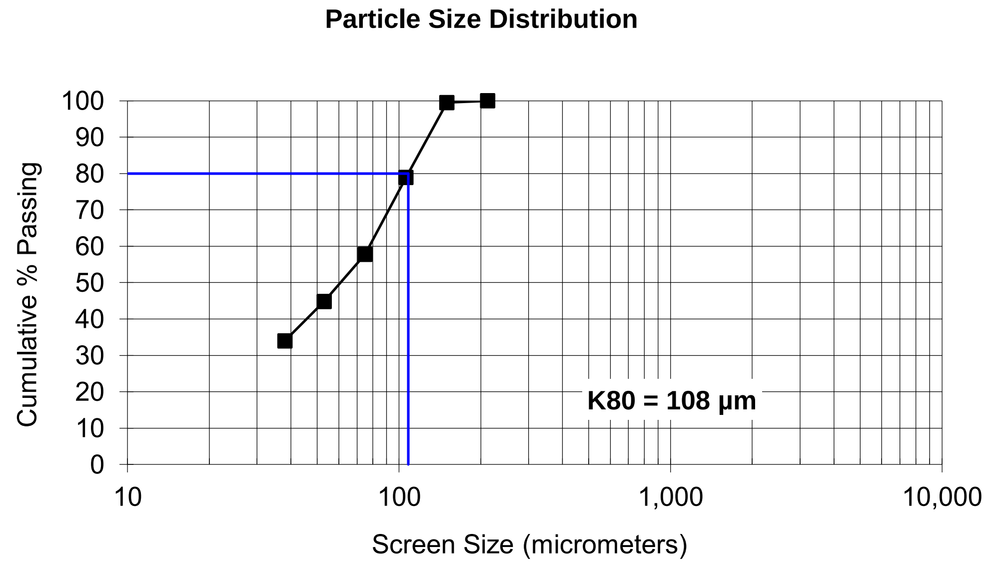
Products	% Distribution																					
	S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
0-2A WHIMS Mag	7.2	8.3	8.0	8.2	8.1	8.6	9.7	8.6	9.9	9.3	10.6	10.2	8.7	10.0	7.3	8.6	2.2	9.1	8.5	8.1	8.9	8.1
0-5A WHIMS Mag	14.5	30.1	29.9	30.5	29.9	29.5	33.0	28.9	31.8	27.8	31.9	29.2	25.6	28.5	20.5	25.4	4.5	23.7	26.3	29.7	27.0	29.6
0-10A WHIMS Mag	22.0	61.4	61.3	62.2	61.0	60.2	67.8	59.0	63.5	55.3	64.1	58.1	48.9	55.8	40.3	50.3	6.6	44.3	55.4	60.8	53.9	60.4
0-20A WHIMS Mag	28.8	78.8	78.9	79.9	78.5	78.1	87.7	77.6	85.3	73.7	86.4	78.5	66.1	75.7	54.5	67.8	8.6	61.2	74.2	78.2	72.2	77.8
0-20A Mag+ Non-Mag -38µm	46.3	86.2	86.4	87.2	85.8	85.9	90.4	85.9	91.3	83.3	92.6	88.6	71.2	86.0	66.1	79.4	28.9	74.8	83.8	85.7	82.4	85.5
20A WHIMS Non-Mag +38µm	53.7	13.8	13.6	12.8	14.2	14.1	9.6	14.1	8.7	16.7	7.4	11.4	28.8	14.0	33.9	20.6	71.1	25.2	16.2	14.3	17.6	14.5
Head (Calc.)	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0

SGS Minerals Services
Size Distribution Analysis

Project No.
13849-002

Sample: **minus 100 master comp** Test No.: **M-2**

Size		Weight	% Retained		% Passing
Mesh	µm	grams	Individual	Cumulative	Cumulative
65	212	0.0	0.0	0.0	100.0
100	150	0.8	0.5	0.5	99.5
150	106	31.9	20.6	21.1	78.9
200	75	32.8	21.1	42.2	57.8
270	53	20.1	13.0	55.2	44.8
400	38	16.8	10.8	66.0	34.0
Pan	-38	52.7	34.0	100.0	0.0
Total	**-**	**155.1**	100.0	**-**	**-**
K80	**108**				

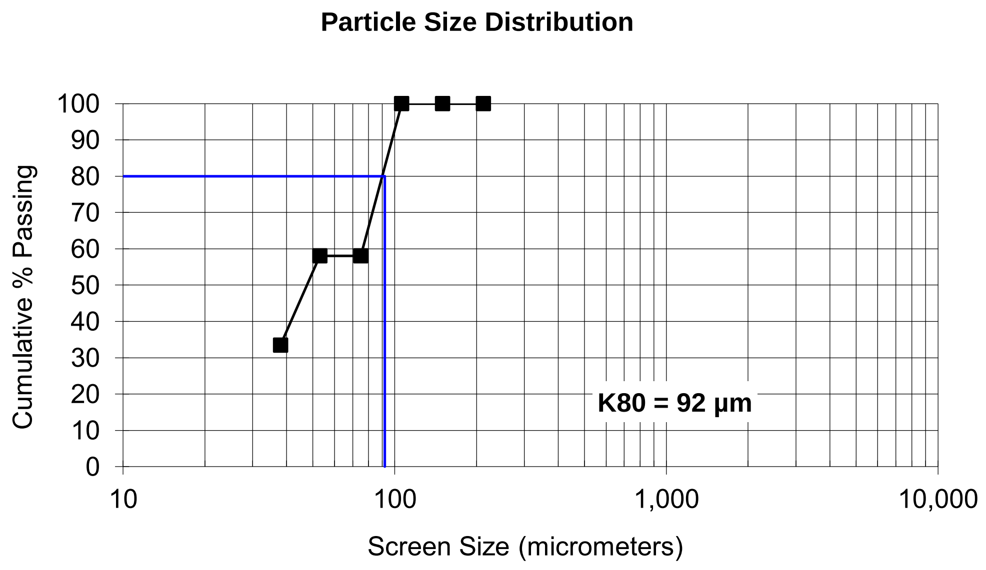


SGS Minerals Services
Size Distribution Analysis

Project No.
13849-002

Sample: **20A Non Mags** Test No.: **M-2**

Size		Weight	% Retained		% Passing
Mesh	µm	grams	Individual	Cumulative	Cumulative
65	212	0.0	0.0	0.0	100.0
100	150	0.0	0.0	0.0	100.0
150	106	0.0	0.0	0.0	100.0
200	75	76.0	41.9	41.9	58.1
270	53	0.0	0.0	41.9	58.1
400	38	44.6	24.6	66.5	33.5
Pan	-38	60.7	33.5	100.0	0.0
Total	**-**	**181.3**	100.0	**-**	**-**
K80	**92**				



Test No.: M-2 **Project No:** 13849-002 **Operator**: Dan Lang **Date**: 06-Dec-12

Purpose: To examine magnetic separation by WHIMS on 100% minus 100 mesh split sample

Procedure:
1) Grind 1 Kg Maste composite to 100% passing 100 mesh
2) Split into 250 grams and pulped and mixed test feed
3) Recorded the % solids of feed.
4) Passed feed through WHIMS at increasing intensities of 2, 5, 10, 20 A
5) A total of 5 products: 2A mag, 5A mag, 10A mag, 20A mag, and 20A non-mag were generated.
6) Submitted all samples for S, and REE ICP Scan.

Feed: 250 g of -150 μm feed sample. Pulped and mixed ~250g of -150 μm feed sample. % Solids =

Grind: Stage-grinding to 100% passing 100mesh WHIMS Fd K_{80} = 108 μm

Definitions:
LREE=La,Ce,Pr,Nd,Sm,Eu,Sc
HREE=Gd,Tb,Dy,Ho,Er,Tm,Yb,Lu,Y
TREE=LREE+HREE

Metallurgical Balance

Products	Weight		Assays, g/t																				
	g	%	S, %	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
0-2A WHIMS Mag	6.6	2.5	4.00	1610	3050	284	944	172	7.1	103	13.8	66.6	10.6	25.5	3.2	19.6	2.5	255	<50	890	1370	6117	500	6617
2-5A WHIMS Mag	6.2	2.4	6.46	7030	13000	1180	4400	637	23.8	382	43.0	191	28.8	71.2	8.7	48.6	6.5	762	<50	2060	4390	26321	1542	27863
5-10A WHIMS Mag	0.0	0.0	0.00	0	0	0	0	0	0.0	0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0	0	0	0	0	0	0
10-20A WHIMS Mag	6.9	2.7	4.98	5510	10200	908	3430	504	20.3	318	37.5	175	26.6	68.1	8.0	46.9	6.2	724	<50	2330	4350	20622	1410	22033
Non-Mag +75µm	100.4	38.7	1.63	120	213	20.5	69	14.0	0.7	9.3	1.1	7.1	1.1	2.9	0.3	2.5	<0.5	22	<50	106	126	487	47	534
Non-Mag -75+38µm	58.9	22.7	1.77	90	157	16.3	54	9.9	0.9	8.3	1.3	6.1	1.3	2.7	0.7	2.6	0.6	18	<50	101	113	378	42	420
Non-Mag -38µm	80.2	30.9	1.42	208	378	38.4	123	22.8	1.2	14.9	2.1	10.9	1.8	4.4	0.6	3.9	0.5	39	<50	184	213	821	78	900
Head (Calc.)	259.2	100.0	1.86	487.1	895.3	83.2	297.6	47.8	2.14	30.3	3.7	18.4	2.95	7.26	0.96	5.68	0.87	68.7	50.0	254.9	396.1	1863.1	138.9	2002.0
Head (Direct)			1.92	407.0	767.0	81.0	254.0	42.6	1.90	26.1	3.3	16.5	2.60	6.70	0.80	5.20	0.70	62.0	< 25	270.0	354.0	1578.5	123.9	1702.4

Products	Weight	% Distribution																					
	%	S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
0-2A WHIMS Mag	2.5	5.5	8.4	8.7	8.7	8.1	9.2	8.5	8.6	9.4	9.2	9.2	8.9	8.5	8.8	7.3	9.5	2.5	8.9	8.8	8.4	9.2	8.4
2-5A WHIMS Mag	2.4	8.3	34.5	34.7	33.9	35.4	31.9	26.6	30.1	27.4	24.8	23.4	23.4	21.6	20.5	17.9	26.5	2.4	19.3	26.5	33.8	26.6	33.3
5-10A WHIMS Mag	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
10-20A WHIMS Mag	2.7	7.1	30.1	30.3	29.1	30.7	28.1	25.3	27.9	26.6	25.3	24.0	25.0	22.1	22.0	19.0	28.1	2.7	24.3	29.2	29.5	27.0	29.3
Non-Mag +75µm	38.7	33.9	9.5	9.2	9.5	9.0	11.4	12.7	11.9	11.4	14.9	14.5	15.5	12.1	17.1	22.3	12.4	38.7	16.1	12.3	10.1	13.1	10.3
Non-Mag -75+38µm	22.7	21.6	4.2	4.0	4.5	4.1	4.7	9.6	6.2	7.9	7.5	10.0	8.4	16.5	10.4	15.7	6.0	22.7	9.0	6.5	4.6	6.8	4.8
Non-Mag -38µm	30.9	23.6	13.2	13.1	14.3	12.8	14.8	17.4	15.2	17.3	18.3	18.9	18.7	19.3	21.3	17.8	17.6	30.9	22.3	16.6	13.6	17.4	13.9
Head (Calc.)	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0

Cumulative Products

Products	Weight	Assays, g/t																					
	%	S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
0-2A WHIMS Mag	2.5	4.00	1610	3050	284	944	172.0	7.1	103.0	13.8	66.6	10.60	25.50	3.20	19.60	2.50	255.0	50.0	890.0	1370.0	6117.1	499.8	6616.9
0-5A WHIMS Mag	4.9	5.19	4235	7870	718	2618	397.2	15.2	238.1	27.9	126.9	19.42	47.64	5.86	33.65	4.44	500.6	50.0	1456.7	2832.8	15903.3	1004.5	16907.8
0-10A WHIMS Mag	4.9	5.19	4235	7870	718	2618	397.2	15.2	238.1	27.9	126.9	19.42	47.64	5.86	33.65	4.44	500.6	50.0	1456.7	2832.8	15903.3	1004.5	16907.8
0-20A WHIMS Mag	7.6	5.12	4682	8686	785	2902	434.6	17.0	266.1	31.3	143.7	21.93	54.80	6.61	38.29	5.05	578.8	50.0	1762.6	3364.2	17556.1	1146.6	18702.8
0-20A Mag+ Non-Mag -38µm	38.5	2.15	1090	2017	186	671	104.0	4.3	64.4	7.9	37.1	5.77	14.34	1.79	10.68	1.40	145.5	50.0	495.3	834.5	4121.9	288.8	4410.8
20A WHIMS Non-Mag +38µm	61.5	1.68	109	192	19	63	12.5	0.8	8.9	1.2	6.7	1.17	2.83	0.45	2.54	0.54	20.5	50.0	104.2	121.2	446.9	44.9	491.7
Head (Calc.)	100.0	1.86	487.1	895.3	83.2	297.6	47.8	2.14	30.3	3.7	18.4	2.9	7.3	1.0	5.7	0.9	68.7	<50	254.9	396.1	1863.1	138.9	2002.0

Products	% Distribution																					
	S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
0-2A WHIMS Mag	5.5	8.4	8.7	8.7	8.1	9.2	8.5	8.6	9.4	9.2	9.2	8.9	8.5	8.8	7.3	9.5	2.5	8.9	8.8	8.4	9.2	8.4
0-5A WHIMS Mag	13.8	42.9	43.4	42.6	43.4	41.1	35.1	38.8	36.8	34.0	32.6	32.4	30.1	29.3	25.2	36.0	4.9	28.2	35.3	42.2	35.7	41.7
0-10A WHIMS Mag	13.8	42.9	43.4	42.6	43.4	41.1	35.1	38.8	36.8	34.0	32.6	32.4	30.1	29.3	25.2	36.0	4.9	28.2	35.3	42.2	35.7	41.7
0-20A WHIMS Mag	20.9	73.0	73.7	71.7	74.1	69.2	60.4	66.7	63.4	59.3	56.6	57.3	52.2	51.3	44.2	64.1	7.6	52.6	64.6	71.6	62.7	71.0
0-20A Mag+ Non-Mag -38µm	44.5	86.3	86.8	86.0	86.9	83.9	77.7	81.9	80.8	77.6	75.5	76.1	71.4	72.5	62.0	81.6	38.5	74.9	81.2	85.3	80.1	84.9
20A WHIMS Non-Mag +38µm	55.5	13.7	13.2	14.0	13.1	16.1	22.3	18.1	19.2	22.4	24.5	23.9	28.6	27.5	38.0	18.4	61.5	25.1	18.8	14.7	19.9	15.1
Head (Calc.)	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0

Appendix E – Detailed of Flotation Tests

Test No. : F1 **Project No:** 13849-002 **Operator**: Dan Lang **Date**: Dec 10/2012

Purpose: To examine effect of collector LR19 at natural pH

Procedure: As outlined below.

Feed: 2000 g charge of Master Composite -6 mesh

Grind: 45 minutes at 65% solids in the laboratory Rod Mill (RM#2) Ro Flotation % Solids = 35

Notes: Flot time and reagent dosage may change based on visual. Ro Fd K_{80} = 111 µm

LREE=La,Ce,Pr,Nd,Sm,Eu,Sc LR-19: 62% FA2, 27% Aero 855, 9% Kerosene, 2% MIBC
HREE=Gd,Tb,Dy,Ho,Er,Tm,Yb,Lu,Y
TREE=LREE+HREE

Conditions:

Stage	Reagents Added, g/t										Time, minutes			Pulp		
	Metso	LR19	PAX	DF250	V4085						Grind	Cond.	Froth	pH	Temp, ºC	
Grind											45				RT	
Condition 1 - 70% solids	500											1			RT	
Condition 2 - 70% solids		150										5		10.2	RT	
															RT	
Rougher 1													1	10	RT	lots of clean pyrite
Rougher 2		150										5	2		RT	
Rougher 3		150		6								5	4	9.3	RT	lots of RE and granular pyrite
Rougher 4		150		6								5	6	8.4	RT	
Rougher 5		150		6								5	6	8.2		
Rougher 6					75							5	2	8.0		
Rougher Total	**500**	**750**		**18**	**75**							**31**	**21**			

Stage	Rougher
Flotation Cell	4L (2 kg)
Speed rpm	1800

Metallurgical Balance

Products	Weight		Assays: U, Th, REE g/t, S %																			
	g	%	S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
Ro Conc 1	156.1	7.8	19.4	3980	6990	692	2220	341	13.3	192	21.9	88.8	13.3	30.1	3.7	21.2	2.9	330	<25	760	1930	14261.3	703.9	14965.2
Ro Conc 2	139.3	6.9	4.30	1090	1990	198	633	107	4.6	71.7	10.8	51.0	8.4	20.6	2.7	16.2	2.2	204	<25	802	1150	4047.6	387.6	4435.2
Ro Conc 3	93.6	4.7	0.90	374	683	71	232	48.1	2.7	40.9	7.3	38.8	6.5	16.6	2.1	13.7	1.8	161	<25	769	888	1435.8	288.7	1724.5
Ro Conc 4	84.1	4.2	0.44	190	359	38.6	134	34.9	2.2	33.2	6.6	36.7	6.4	16.2	2.2	13.0	1.8	149	<25	845	863	783.7	265.1	1048.8
Ro Conc 5	101.0	5.0	0.16	87.4	170	19.0	65.7	19.0	1.4	18.9	3.7	20.6	3.7	9.6	1.3	8.4	1.3	85	<25	570	397	387.5	152.5	540.0
Ro Conc 6	29.0	1.4	0.07	65.3	127	14.1	48.8	13.4	0.9	13.1	2.6	15.3	2.7	7.2	1.1	7.2	1.0	61	<25	500	222	294.5	111.2	405.7
Ro Tls	1401.4	69.9	0.01	14.4	25	2.6	8.6	1.9	<0.3	1.6	<0.5	1.7	<0.3	0.7	<0.3	0.8	<0.5	<10	<25	43.1	22.7	77.8	16.4	94.2
Head (Calc.)	2004.5	100.0	1.89	426.5	757.5	75.6	243.4	40.2	1.9	25.5	3.6	16.3	2.6	6.3	1.0	5.0	1.0	65.8	25.0	252.4	347.0	1570.0	127.1	1697.1
Head (Direct)			1.92	407.0	767.0	81.0	254.0	42.6	1.90	26.10	3.3	16.50	2.6	6.70	0.80	5.20	0.70	62.00	25.0	270.0	354.0	1578.5	123.9	1702

Products	Weight	% Distribution																					
	%	S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
Ro Conc 1	7.8	80.1	72.7	71.9	71.3	71.0	66.1	55.5	58.6	46.8	42.5	39.4	37.2	30.1	32.7	23.3	39.1	7.8	23.5	43.3	70.7	43.1	68.7
Ro Conc 2	6.9	15.8	17.8	18.3	18.2	18.1	18.5	17.1	19.5	20.6	21.8	22.2	22.7	19.6	22.3	15.8	21.5	6.9	22.1	23.0	17.9	21.2	18.2
Ro Conc 3	4.7	2.2	4.1	4.2	4.4	4.5	5.6	6.8	7.5	9.3	11.1	11.6	12.3	10.2	12.7	8.7	11.4	4.7	14.2	12.0	4.3	10.6	4.7
Ro Conc 4	4.2	1.0	1.9	2.0	2.1	2.3	3.6	4.9	5.5	7.6	9.5	10.2	10.8	9.6	10.8	7.8	9.5	4.2	14.0	10.4	2.1	8.8	2.6
Ro Conc 5	5.0	0.4	1.0	1.1	1.3	1.4	2.4	3.8	3.7	5.1	6.4	7.1	7.7	6.8	8.4	6.8	6.5	5.0	11.4	5.8	1.2	6.0	1.6
Ro Conc 6	1.4	0.1	0.2	0.2	0.3	0.3	0.5	0.7	0.7	1.0	1.4	1.5	1.7	1.7	2.1	1.5	1.3	1.4	2.9	0.9	0.3	1.3	0.3
Ro Tls	69.9	0.4	2.4	2.3	2.4	2.5	3.3	11.2	4.4	9.6	7.3	8.0	7.8	21.9	11.1	36.1	10.6	69.9	11.9	4.6	3.5	9.0	3.9
Head (Calc.)	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0

Cumulative Products

Products	Weight	Assays: U, Th, REE g/t, S %																					
	%	S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
Ro Conc 1	7.8	19.4	3980.0	6990.0	692.0	2220.0	341.0	13.3	192.0	21.9	88.8	13.3	30.1	3.7	21.2	2.9	330.0	25.0	760.0	1930.0	14261.3	703.9	14965.2
Ro Conc 1-2	14.7	12.3	2617.2	4632.2	459.0	1471.6	230.7	9.2	135.3	16.7	71.0	11.0	25.6	3.2	18.8	2.6	270.6	25.0	779.8	1562.2	9444.9	554.7	9999.6
Ro Conc 1-3	19.4	9.5	2077.4	3681.9	365.7	1173.4	186.7	7.6	112.6	14.4	63.2	9.9	23.4	3.0	17.6	2.4	244.2	25.0	777.2	1400.0	7517.8	490.7	8008.5
Ro Conc 1-4	23.6	7.9	1741.9	3091.2	307.5	988.6	159.7	6.7	98.5	13.0	58.5	9.3	22.2	2.8	16.8	2.3	227.3	25.0	789.3	1304.5	6320.7	450.6	6771.3
Ro Conc 1-5	28.6	6.6	1450.8	2577.3	256.8	826.2	135.0	5.7	84.5	11.4	51.8	8.3	20.0	2.6	15.3	2.1	202.3	25.0	750.7	1144.9	5276.9	398.2	5675.1
Ro Conc 1-6	30.1	6.2	1384.2	2459.5	245.1	788.8	129.1	5.5	81.0	11.0	50.1	8.0	19.3	2.5	14.9	2.1	195.5	25.0	738.6	1100.5	5037.3	384.4	5421.7
Ro Tls	69.9	0.01	14.4	25.0	2.6	8.6	1.90	0.30	1.60	0.50	1.70	0.30	0.70	0.30	0.80	0.50	10.0	25.0	43.1	22.7	77.8	16.4	94.2

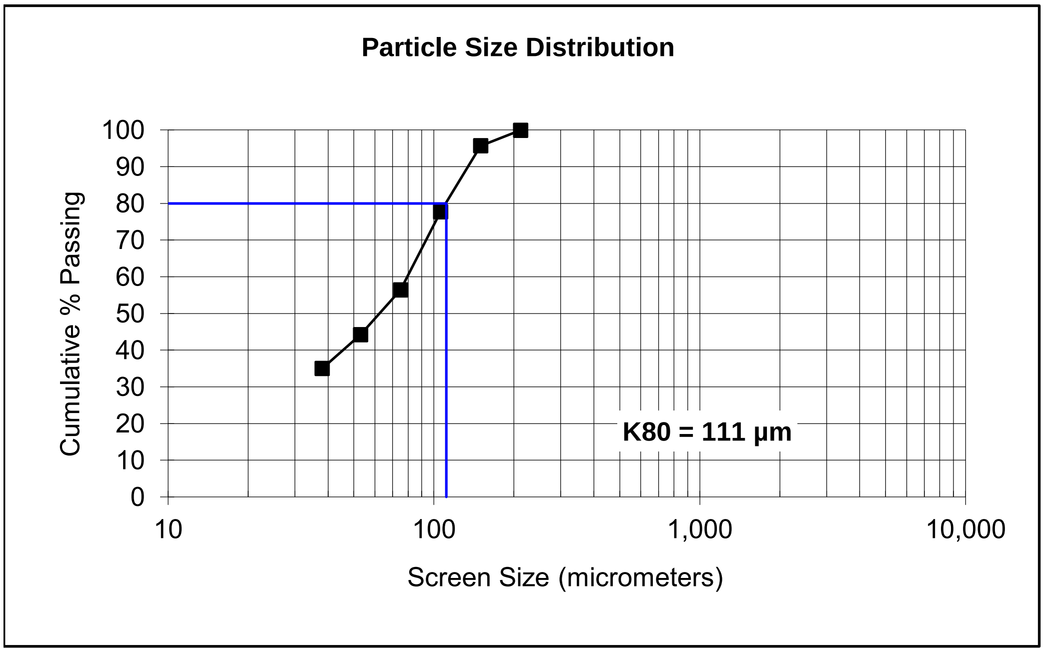
Products	% Distribution																					
	S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
Ro Conc 1	80.1	72.7	71.9	71.3	71.0	66.1	55.5	58.6	46.8	42.5	39.4	37.2	30.1	32.7	23.3	39.1	7.8	23.5	43.3	70.7	43.1	68.7
Ro Conc 1-2	95.9	90.4	90.1	89.5	89.1	84.6	72.6	78.2	67.3	64.3	61.7	59.9	49.7	55.0	39.1	60.6	14.7	45.5	66.3	88.7	64.3	86.8
Ro Conc 1-3	98.2	94.5	94.3	93.9	93.6	90.2	79.3	85.7	76.7	75.5	73.2	72.1	59.9	67.7	47.8	72.0	19.4	59.8	78.3	92.9	74.9	91.6
Ro Conc 1-4	99.1	96.4	96.3	96.1	95.9	93.8	84.3	91.1	84.3	84.9	83.4	82.9	69.6	78.5	55.6	81.5	23.6	73.8	88.7	95.0	83.7	94.2
Ro Conc 1-5	99.6	97.4	97.5	97.3	97.2	96.2	88.1	94.9	89.4	91.3	90.5	90.6	76.4	86.9	62.4	88.0	28.6	85.2	94.5	96.3	89.7	95.8
Ro Conc 1-6	99.6	97.6	97.7	97.6	97.5	96.7	88.8	95.6	90.4	92.7	92.0	92.2	78.1	88.9	63.9	89.4	30.1	88.1	95.4	96.5	91.0	96.1
Ro Tls	0.4	2.4	2.3	2.4	2.5	3.3	11.2	4.4	9.6	7.3	8.0	7.8	21.9	11.1	36.1	10.6	69.9	11.9	4.6	3.5	9.0	3.9

SGS Minerals Services
Size Distribution Analysis

Project No.
13849-002

Sample: **Comb Prod** Test No.: **F2**

Size		Weight	% Retained		% Passing
Mesh	µm	grams	Individual	Cumulative	Cumulative
65	212	0.1	0.1	0.1	99.9
100	150	4.2	4.2	4.3	95.7
150	106	18.0	18.0	22.3	77.7
200	75	21.3	21.3	43.6	56.4
270	53	12.2	12.2	55.8	44.2
400	38	9.2	9.2	65.0	35.0
Pan	-38	35.0	35.0	100.0	0.0
Total	**-**	**100.0**	100.0	**-**	**-**
K80	**111**				



Test No. : F2 **Project No:** 13849-002 **Operator**: Dan Lang **Date**: Dec 10/2012

Purpose: To repeat test F-1 bur reduce the pH with H_2SO_4

Procedure: As outlined below.

Feed: 2000 g charge of Master Composite -6 mesh

Grind: 45 minutes at 65% solids in the laboratory Rod Mill (RM#2) Ro Flotation % Solids = 35

Notes: Flot time and reagent dosage may change based on visual. Ro Fd K_{80} = 111 µm

LREE=La,Ce,Pr,Nd,Sm,Eu,Sc LR-19: 62% FA2, 27% Aero 855, 9% Kerosene, 2% MIBC
HREE=Gd,Tb,Dy,Ho,Er,Tm,Yb,Lu,Y
TREE=LREE+HREE

Conditions:

Stage	Reagents Added, g/t										Time, minutes			Pulp	
	Metso	LR19	H_2SO_4								Grind	Cond.	Froth	pH	Temp, °C
Grind											45				19
Condition 1 - 70% solids	500		140									1			19
Condition 2 - 70% solids		300										5		8.4	19
															19
Rougher 1													3		19
Rougher 2		150										5	4	7.9	19
Rougher 3		150										5	4	7.8	19
Rougher 4		150										5	4	7.6	19
Rougher 5		150										5	4	7.6	19
Rougher Total	**500**	**900**	**140**									**26**	**19**		

Stage	Rougher
Flotation Cell	4L (2 kg)
Speed rpm	1800

Metallurgical Balance

Products	Weight		Assays: U, Th, REE g/t, S %																				
	g	%	S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
Ro Conc 1	301.7	15.0	12.2	2590	4590	457	1460	234	9.7	136	17	73.6	11.5	26.7	3.5	19.9	2.7	287	<25	867	1660	9365.7	577.9	9943.6
Ro Conc 2	128.9	6.4	0.40	311	562	58.3	190	38.2	2.1	30.6	5.3	27.4	4.6	12	1.6	9.6	1.3	116	<25	589	613	1186.6	208.4	1395.0
Ro Conc 3	131.2	6.5	0.14	179	321	33.6	116	25.3	1.5	21.5	4	21.6	3.7	9.7	1.3	8.1	1.1	91	<25	519	474	701.4	162.0	863.4
Ro Conc 4	94.1	4.7	0.07	125	234	24.4	82	18.5	1.2	15.6	3	16.1	2.9	8	1.1	7.2	1	69	<25	510	271	510.1	123.9	634.0
Ro Conc 5	70.3	3.5	0.04	83.4	154	16.7	56.9	14.2	0.9	12.5	2.5	14.5	2.5	7	1	6.4	0.9	57	<25	487	210	351.1	104.3	455.4
Ro Tls	1281.5	63.8	0.01	13	21.4	2.1	7.1	1.4	<0.3	1.2	<0.5	1.4	<0.3	<0.5	<0.3	<0.5	<0.5	<10	<25	31.5	14.5	70.3	15.2	85.5
Head (Calc.)	2007.7	100.0	1.88	437.9	776.8	77.7	249.5	41.5	1.97	25.7	3.7	16.4	2.7	6.4	1.0	5.0	1.0	68.1	25.0	263.1	349.1	1610	130	1740
Head (Direct)			1.92	407.0	767.0	81.0	254.0	42.6	1.90	26.10	3.3	16.50	2.6	6.70	0.80	5.20	0.70	62.00	25.0	270.0	354.0	1578.5	123.9	1702

Products	Weight	% Distribution																				
	%	S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
Ro Conc 1	15.0	97.6	88.9	88.8	88.4	87.9	84.7	74.0	79.4	69.0	67.5	64.5	63.1	53.0	59.6	42.3	63.3	15.0	49.5	71.5	87.4	66.8	85.9
Ro Conc 2	6.4	1.4	4.6	4.6	4.8	4.9	5.9	6.8	7.6	9.2	10.7	11.0	12.1	10.4	12.3	8.7	10.9	6.4	14.4	11.3	4.7	10.3	5.1
Ro Conc 3	6.5	0.5	2.7	2.7	2.8	3.0	4.0	5.0	5.5	7.1	8.6	9.0	10.0	8.6	10.6	7.5	8.7	6.5	12.9	8.9	2.8	8.1	3.2
Ro Conc 4	4.7	0.2	1.3	1.4	1.5	1.5	2.1	2.9	2.8	3.8	4.6	5.1	5.9	5.2	6.7	4.9	4.7	4.7	9.1	3.6	1.5	4.5	1.7
Ro Conc 5	3.5	0.1	0.7	0.7	0.8	0.8	1.2	1.6	1.7	2.4	3.1	3.3	3.9	3.5	4.5	3.3	2.9	3.5	6.5	2.1	0.8	2.8	0.9
Ro Tls	63.8	0.3	1.9	1.8	1.7	1.8	2.2	9.7	3.0	8.6	5.5	7.1	5.0	19.3	6.4	33.3	9.4	63.8	7.6	2.7	2.8	7.5	3.1
Head (Calc.)	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0

Cumulative Products

Products	Weight	Assays: U, Th, REE g/t, S %																				
	%	S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
Ro Conc 1	15.0	12.2	2590.0	4590.0	457.0	1460.0	234.0	9.7	136.0	17.0	73.6	11.5	26.7	3.5	19.9	2.7	287.0	25.0	867.0	1660.0	9365.7	577.9	9943.6
Ro Conc 1-2	21.4	8.7	1907.8	3384.2	337.6	1079.8	175.4	7.4	104.4	13.5	59.8	9.4	22.3	2.9	16.8	2.3	235.8	25.0	783.8	1346.6	6917.3	467.3	7384.6
Ro Conc 1-3	28.0	6.7	1504.1	2668.9	266.6	854.7	140.3	6.0	85.1	11.3	50.9	8.1	19.4	2.6	14.8	2.0	202.0	25.0	721.9	1142.8	5465.7	396.0	5861.7
Ro Conc 1-4	32.7	5.7	1306.2	2319.5	231.9	743.9	122.9	5.3	75.1	10.1	45.9	7.3	17.7	2.3	13.7	1.9	182.9	25.0	691.5	1017.7	4754.7	357.0	5111.7
Ro Conc 1-5	36.2	5.2	1187.8	2109.9	211.1	677.4	112.3	4.9	69.0	9.4	42.8	6.9	16.7	2.2	13.0	1.8	170.7	25.0	671.7	939.5	4328.4	332.5	4660.9
Ro Tls	63.8	0.01	13.0	21.4	2.1	7.1	1.40	0.30	1.20	0.50	1.40	0.30	0.50	0.30	0.50	0.50	10.0	25.0	31.5	14.5	70.3	15.2	85.5

Products	% Distribution																					
	S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
Ro Conc 1	97.6	88.9	88.8	88.4	87.9	84.7	74.0	79.4	69.0	67.5	64.5	63.1	53.0	59.6	42.3	63.3	15.0	49.5	71.5	87.4	66.8	85.9
Ro Conc 1-2	98.9	93.4	93.4	93.2	92.8	90.6	80.8	87.0	78.2	78.2	75.5	75.2	63.4	71.9	51.0	74.2	21.4	63.9	82.7	92.1	77.1	91.0
Ro Conc 1-3	99.4	96.1	96.1	96.0	95.8	94.6	85.8	92.5	85.2	86.8	84.5	85.2	72.0	82.4	58.5	83.0	28.0	76.8	91.6	95.0	85.3	94.2
Ro Conc 1-4	99.6	97.4	97.5	97.5	97.4	96.7	88.7	95.3	89.0	91.4	89.6	91.1	77.2	89.2	63.4	87.7	32.7	85.9	95.2	96.5	89.7	95.9
Ro Conc 1-5	99.7	98.1	98.2	98.3	98.2	97.8	90.3	97.0	91.4	94.5	92.9	95.0	80.7	93.6	66.7	90.6	36.2	92.4	97.3	97.2	92.5	96.9
Ro Tls	0.3	1.9	1.8	1.7	1.8	2.2	9.7	3.0	8.6	5.5	7.1	5.0	19.3	6.4	33.3	9.4	63.8	7.6	2.7	2.8	7.5	3.1

Test No. : F3 **Project No:** 13849-002 **Operator:** Dan Lang **Date:** 13-Dec-12

Purpose: To examine effect of collector Cupferron

Procedure: As outlined below.

Feed: 2000 g charge of Master Composite -6 mesh

Grind: 45 minutes at 65% solids in the laboratory Rod Mill (RM#2) Ro Flotation % Solids = 35

Notes: Flot time and reagent dosage may change based on visual. Ro Fd K_{80} = 111 µm

LREE=La,Ce,Pr,Nd,Sm,Eu,Sc
HREE=Gd,Tb,Dy,Ho,Er,Tm,Yb,Lu,Y
TREE=LREE+HREE

LR-19: 62% FA2, 27% Aero 855, 9% Kerosene, 2% MIBC

Conditions:

Stage	Reagents Added, g/t										Time, minutes			Pulp		
	KAX	DF250	Cupferron	H_2SO_4	FS-2						Grind	Cond.	Froth	pH	Temp, ºC	
Grind											45				19	
Ro 1	50	5		180								1+1	3	6.5	19	
Condition			2000	800								2+2		6.5	19	Nothing floats
															19	added FS-2 as a secondary collector
Rougher 2					150							2	3	6.5	19	
Rougher 3					225							2	4	7.0	19	
Rougher 4					120							2	5	7.2	19	
Rougher Total	**50**	**5**	**2000**	**980**	**495**							**6**	**15**			too much mass floataed

Stage	Rougher
Flotation Cell	4L (2 kg)
Speed rpm	1800

Metallurgical Balance

Products	Weight			Assays: U, Th, REE g/t; S %																			
	g	%	S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
Ro Conc 1	91.4	4.9	34.4	633	1150	116	390	81	4	59	7	34	6	14	2	11	<2	149	<100	765	737	2474.0	284.0	2758.0
Ro Conc 2	170.6	9.2	0.75	1690	3030	296	1020	160	7	111	13	58	10	24	3	19	3	235	<100	904	1260	6303.0	476.0	6779.0
Ro Conc 3	441.3	23.8	0.15	450	809	79	285	37	2	32	4	16	2	7	<2	5	<2	63	<100	234	340	1762.0	133.0	1895.0
Ro Conc 4	652.7	35.3	0.06	185	339	31	102	20	<2	11	<2	7	<2	3	<2	2	<2	<40	<100	103	139	779.0	71.0	850.0
Ro Tls	494.4	26.7	0.10	225	408	37	133	21	<2	14	2	9	<2	3	<2	3	<2	42	<100	149	160	926.0	79.0	1005.0
Head (Calc.)	1850.4	100.0	1.85	419.8	757.7	72.7	252.8	40.2	2.56	28.4	3.74	15.7	2.94	6.43	2.09	4.99	2.09	69.4	<100	253.1	325.4	1645.7	135.8	1781.5
Head (Direct)			1.92	407.0	767.0	81.0	254.0	42.6	1.90	26.10	3.3	16.50	2.6	6.70	0.80	5.20	0.70	62.00	25.0	270.0	354.0	1578.5	123.9	1702

Products	Weight	% Distribution																					
	%	S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
Ro Conc 1	4.9	91.7	7.4	7.5	7.9	7.6	9.9	7.7	10.3	9.2	10.7	10.1	10.7	4.7	10.9	4.7	10.6	4.9	14.9	11.2	7.4	10.3	7.6
Ro Conc 2	9.2	3.7	37.1	36.9	37.5	37.2	36.7	25.2	36.0	32.1	34.0	31.4	34.4	13.2	35.1	13.2	31.2	9.2	32.9	35.7	35.3	32.3	35.1
Ro Conc 3	23.8	1.9	25.6	25.5	25.9	26.9	21.9	18.6	26.9	25.5	24.3	16.3	25.9	22.8	23.9	22.8	21.7	23.8	22.1	24.9	25.5	23.4	25.4
Ro Conc 4	35.3	1.1	15.5	15.8	15.0	14.2	17.5	27.6	13.7	18.9	15.7	24.0	16.4	33.7	14.1	33.7	20.3	35.3	14.4	15.1	16.7	18.4	16.8
Ro Tls	26.7	1.4	14.3	14.4	13.6	14.1	13.9	20.9	13.2	14.3	15.3	18.2	12.5	25.5	16.0	25.5	16.2	26.7	15.7	13.1	15.0	15.5	15.1
Head (Calc.)	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0

Cumulative Products

Products	Weight	Assays: U, Th, REE g/t; S %																					
	%	S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
Ro Conc 1	4.9	34.4	633.0	1150.0	116.0	390.0	81.0	4.0	59.0	7.0	34.0	6.0	14.0	2.0	11.0	2.0	149.0	100.0	765.0	737.0	2474.0	284.0	2758.0
Ro Conc 1-2	14.2	12.5	1321.3	2374.2	233.2	800.2	132.4	6.0	92.9	10.9	49.6	8.6	20.5	2.7	16.2	2.7	205.0	100.0	855.5	1077.5	4967.2	409.0	5376.3
Ro Conc 1-3	38.0	4.75	774.6	1392.1	136.4	476.9	72.6	3.5	54.7	6.6	28.5	4.5	12.0	2.2	9.2	2.2	115.9	100.0	465.5	614.8	2956.0	235.8	3191.9
Ro Conc 1-4	73.3	2.49	490.8	885.2	85.7	296.5	47.3	2.8	33.7	4.4	18.2	3.3	7.7	2.1	5.7	2.1	79.4	100.0	291.0	385.8	1908.1	156.5	2064.6
Ro Tls	26.7	0.10	225.0	408.0	37.0	133.0	21.0	2.0	14.0	2.0	9.0	2.0	3.0	2.0	3.0	2.0	42.0	100.0	149.0	160.0	926.0	79.0	1005.0

Products	% Distribution																					
	S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
Ro Conc 1	91.7	7.4	7.5	7.9	7.6	9.9	7.7	10.3	9.2	10.7	10.1	10.7	4.7	10.9	4.7	10.6	4.9	14.9	11.2	7.4	10.3	7.6
Ro Conc 1-2	95.5	44.6	44.4	45.4	44.8	46.6	32.9	46.3	41.3	44.7	41.5	45.1	17.9	46.0	17.9	41.8	14.2	47.9	46.9	42.7	42.7	42.7
Ro Conc 1-3	97.4	70.1	69.8	71.4	71.7	68.5	51.6	73.2	66.8	69.0	57.8	71.1	40.7	69.8	40.7	63.5	38.0	69.9	71.8	68.3	66.0	68.1
Ro Conc 1-4	98.6	85.7	85.6	86.4	85.9	86.1	79.1	86.8	85.7	84.7	81.8	87.5	74.5	84.0	74.5	83.8	73.3	84.3	86.9	85.0	84.5	84.9
Ro Tls	1.4	14.3	14.4	13.6	14.1	13.9	20.9	13.2	14.3	15.3	18.2	12.5	25.5	16.0	25.5	16.2	26.7	15.7	13.1	15.0	15.5	15.1

Test No. : F4 **Project No:** 13849-002 **Operator**: Dan Lang **Date**: 13-Dec-12

Purpose: To examine effect of collector LR-19 with a sulphide prefloat, and float at a reduced pH

Procedure: As outlined below.

Feed: 2000 g charge of Master Composite

Grind: 45 minutes at 65% solids in the laboratory Rod Mill (RM#2)

Ro Flotation % Solids = 35

Notes: Flot time and reagent dosage may change based on visual.
LREE=La,Ce,Pr,Nd,Sm,Eu,Sc
HREE=Gd,Tb,Dy,Ho,Er,Tm,Yb,Lu,Y
TREE=LREE+HREE

Ro Fd K_{80} = 111 µm

Conditions:

Stage	Reagents Added, g/t										Time, minutes			Pulp		
	KAX	DF250	LR19	H_2SO_4	Metso						Grind	Cond.	Froth	pH	Temp, ºC	
Grind											45				19	
Ro 1	50	5		200								1+1	3	6.5	19	
Condition 1					500							1		6.5	19	
Condition 2				400								1			19	
Rougher 2			150	100								5	6	6.0	19	pH drifts up to 7.2
Rougher 3			150	120								5	6	6.5	19	
Rougher 4			150	80								5	5	6.5	19	
Rougher Total	**50**	**5**	**450**	**900**	**500**							**17**	**20**			

Stage	Rougher
Flotation Cell	4L (2 kg)
Speed rpm	1800

Metallurgical Balance

Products	Weight			Assays: U, Th, REE g/t; S %																			
	g	%	S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
Ro Conc 1	114.1	5.7	30.7	621	1130	115	376	69	3	47	7	34	6	14	<2	11	<2	133	<100	684	692	2414.0	256.0	2670.0
Ro Conc 2	259.7	12.9	0.79	1910	3430	342	1210	194	8	114	12	58	8	21	3	16	<2	183	<100	672	1210	7194.0	417.0	7611.0
Ro Conc 3	218.6	10.9	0.21	398	758	72	237	39	2	29	4	19	3	8	<2	7	<2	70	<100	339	395	1606.0	144.0	1750.0
Ro Conc 4	371.3	18.5	0.05	145	276	24	82	14	<2	11	<2	7	<2	3	<2	3	<2	<40	<100	134	151	643.0	72.0	715.0
Ro Tls	1043.6	52.0	0.04	105	206	15	62	12	<2	9	<2	5	<2	2	<2	2	<2	<40	<100	109	98	502.0	66.0	568.0
Head (Calc.)	2007.3	100.0	1.90	407.2	748.7	70.9	251.1	42.1	2.8	27.3	3.8	15.4	3.1	6.0	2.1	5.1	<2	67.1	<100	244.2	317.8	1622.8	131.8	1754.6
Head (Direct)			1.92	407.0	767.0	81.0	254.0	42.6	1.90	26.10	3.3	16.50	2.6	6.70	0.80	5.20	0.70	62.00	25.0	270.0	354.0	1578.5	123.9	1702

Products	Weight	% Distribution																				
	%	S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
Ro Conc 1	5.7	91.8	8.7	8.6	9.2	8.5	9.3	6.0	9.8	10.5	12.5	11.0	13.3	5.3	12.4	5.7	11.3	5.7	15.9	12.4	8.5	11.0	8.6
Ro Conc 2	12.9	5.4	60.7	59.3	62.4	62.3	59.6	36.5	54.0	40.9	48.7	33.3	45.4	18.2	41.0	12.9	35.3	12.9	35.6	49.3	57.4	40.9	56.1
Ro Conc 3	10.9	1.2	10.6	11.0	11.1	10.3	10.1	7.7	11.6	11.5	13.4	10.5	14.6	10.2	15.1	10.9	11.4	10.9	15.1	13.5	10.8	11.9	10.9
Ro Conc 4	18.5	0.5	6.6	6.8	6.3	6.0	6.2	13.1	7.5	9.7	8.4	11.9	9.3	17.4	11.0	18.5	11.0	18.5	10.2	8.8	7.3	10.1	7.5
Ro Tls	52.0	1.1	13.4	14.3	11.0	12.8	14.8	36.7	17.1	27.4	16.9	33.4	17.4	48.8	20.6	52.0	31.0	52.0	23.2	16.0	16.1	26.0	16.8
Head (Calc.)	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0

Cumulative Products

Products	Weight	Assays: U, Th, REE g/t; S %																				
	%	S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
Ro Conc 1	5.7	30.7	621.0	1130.0	115.0	376.0	69.0	3.0	47.0	7.0	34.0	6.0	14.0	2.0	11.0	2.0	133.0	100.0	684.0	692.0	2414.0	256.0	2670.0
Ro Conc 1-2	18.6	9.9	1516.5	2727.9	272.7	955.4	155.8	6.5	93.5	10.5	50.7	7.4	18.9	2.7	14.5	2.0	167.7	100.0	675.7	1051.9	5734.9	367.9	6102.8
Ro Conc 1-3	29.5	6.34	1103.8	2001.0	198.6	690.3	112.7	4.8	69.7	8.1	39.0	5.8	14.9	2.4	11.7	2.0	131.7	100.0	551.4	809.5	4211.3	285.3	4496.6
Ro Conc 1-4	48.0	3.91	734.4	1336.4	131.4	455.9	74.7	3.7	47.1	5.7	26.7	4.3	10.3	2.3	8.4	2.0	96.4	100.0	390.6	555.8	2836.5	203.1	3039.6
Ro Tls	52.0	0.04	105.0	206.0	15.0	62.0	12.0	2.0	9.0	2.0	5.0	2.0	2.0	2.0	2.0	2.0	40.0	100.0	109.0	98.0	502.0	66.0	568.0

Products	% Distribution																					
	S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
Ro Conc 1	91.8	8.7	8.6	9.2	8.5	9.3	6.0	9.8	10.5	12.5	11.0	13.3	5.3	12.4	5.7	11.3	5.7	15.9	12.4	8.5	11.0	8.6
Ro Conc 1-2	97.2	69.4	67.9	71.7	70.8	68.9	42.6	63.8	51.4	61.3	44.2	58.8	23.6	53.3	18.6	46.6	18.6	51.5	61.6	65.8	52.0	64.8
Ro Conc 1-3	98.4	80.0	78.9	82.7	81.1	79.0	50.2	75.4	62.9	74.7	54.7	73.3	33.8	68.4	29.5	58.0	29.5	66.6	75.2	76.6	63.9	75.6
Ro Conc 1-4	98.9	86.6	85.7	89.0	87.2	85.2	63.3	82.9	72.6	83.1	66.6	82.6	51.2	79.4	48.0	69.0	48.0	76.8	84.0	83.9	74.0	83.2
Ro Tls	1.1	13.4	14.3	11.0	12.8	14.8	36.7	17.1	27.4	16.9	33.4	17.4	48.8	20.6	52.0	31.0	52.0	23.2	16.0	16.1	26.0	16.8

Test No. : F5 **Project No:** 13849-002 **Operator**: Dan Lang **Date**: 21-Dec-12

Purpose: To examine effect of collecto SHD with a sulphide prefloat.
Demonstration test for John Goode.

Procedure: As outlined below.

Feed: 2000 g charge of Master Composite

Grind: 45 minutes at 65% solids in the laboratory Rod Mill (RM#2) Ro Flotation % Solids = 35

Notes: Flot time and reagent dosage may change based on visual. Ro Fd K_{80} = 111 µm
LREE=La,Ce,Pr,Nd,Sm,Eu,Sc
HREE=Gd,Tb,Dy,Ho,Er,Tm,Yb,Lu,Y
TREE=LREE+HREE

Conditions:

Stage	**Reagents Added, g/t**										**Time, minutes**			**Pulp**		
	KAX	**DF250**	**LR19**	**H_2SO_4**	**Metso**	**SHD**	**NaOH**				**Grind**	**Cond.**	**Froth**	**pH**	**Temp, ºC**	
Grind											45					
Ro 1	50	2.5										1	2.5	8.3	32	
Condition 1				150	500							2		10.2-9.2		
Condition 2						1000						5		7.4		poor froth loading no floatation
REO Ro Conc1						1000	150					6	3	9.0 - 7.8		some fine black mineralization on an empty froth
REO Ro Conc2						1000						5	6	7.8		very light froth only fines
REO Ro Conc3			150									5	4	7.9		Dark loaded froth seems like good REO recovery only in this stage of floatation
Rougher Total	**50**	**2.5**	**150**	**150**	**500**	**3000**	**150**					**24**	**15.5**			

Stage	Rougher
Flotation Cell	4L (2 kg)
Speed rpm	1800

Metallurgical Balance

Products	**Weight**		**Assays: U, Th, REE g/t; S %**																					
	g	**%**	**S**	**S$^{=}$**	**La**	**Ce**	**Pr**	**Nd**	**Sm**	**Eu**	**Gd**	**Tb**	**Dy**	**Ho**	**Er**	**Tm**	**Yb**	**Lu**	**Y**	**Sc**	**U**	**Th**	**LREE**	**HREE**	**TREE**
S Ro Conc	116.2	5.8	30.0	28.5	819	1570	149	464	89.4	4.1	64.2	9.3	44.6	6.9	17.3	1.9	12.4	1.6	180	<25	773	878	3120.5	158.2	2639.7
REO Ro Conc1	65.8	3.3	0.60	0.47	690	1260	121	371	66.8	2.8	42.1	4.9	24.3	3.6	9.5	0.9	7.0	1.0	104	<25	378	521	2536.6	93.3	2043.9
REO Ro Conc2	199.0	10.0	0.25	0.18	466	862	80.8	249	42.9	2.3	27.4	3.4	15.7	2.2	5.9	0.5	3.7	0.7	66	<25	249	301	1728	59.5	1387.5
REO Ro Conc3	98.6	4.9	1.17	1.05	4520	8590	796	2420	424	16.2	257	28.2	130	19.4	48.8	6.1	35.5	4.9	500	<25	1510	2670	16791.2	529.9	13301.1
REO Ro Tls	1516.8	76.0	0.06	0.07	116	218	19.5	62	11.0	0.6	6.2	1.0	4.0	0.5	1.1	<0.3	0.9	<0.5	26	<25	99.9	81.4	452.1	14.5	376.6
Head (Calc.)	1996.4	100.0	1.89	1.80	428.2	808.7	74.8	230.7	41.0	1.8	25.3	3.2	14.4	2.1	5.2	0.7	3.8	0.8	64.9	<25	232.7	292.0	1610.3	120.3	1730.6
Head (Direct)			1.92	--	407.0	767.0	81.0	254.0	42.6	1.90	26.1	3.3	16.5	2.6	6.70	0.80	5.2	0.7	62.0	25.0	270.0	354.0	1578.5	123.9	1702

Products	**Weight**	**% Distribution**																					
	%	**S**	**S$^{=}$**	**La**	**Ce**	**Pr**	**Nd**	**Sm**	**Eu**	**Gd**	**Tb**	**Dy**	**Ho**	**Er**	**Tm**	**Yb**	**Lu**	**Y**	**Sc**	**U**	**Th**	**LREE**	**HREE**	**TREE**
S Ro Conc	5.8	92.2	92.3	11.1	11.3	11.6	11.7	12.7	13.1	14.8	16.9	18.0	19.3	19.5	15.4	19.2	11.4	16.1	5.8	19.3	17.5	11.3	16.4	11.6
REO Ro Conc1	3.3	1.0	0.9	5.3	5.1	5.3	5.3	5.4	5.1	5.5	5.1	5.6	5.7	6.1	4.1	6.1	4.0	5.3	3.3	5.4	5.9	5.2	5.4	5.2
REO Ro Conc2	10.0	1.3	1.0	10.8	10.6	10.8	10.8	10.4	12.6	10.8	10.6	10.9	10.6	11.4	6.9	9.8	8.5	10.1	10.0	10.7	10.3	10.7	10.4	10.7
REO Ro Conc3	4.9	3.1	2.9	52.1	52.5	52.5	51.8	51.1	44.1	50.3	43.6	44.5	46.1	46.8	41.9	46.7	29.6	38.0	4.9	32.0	45.2	51.5	42.3	50.9
REO Ro Tls	76.0	2.4	3.0	20.6	20.5	19.8	20.4	20.4	25.1	18.6	23.8	21.1	18.3	16.2	31.7	18.2	46.5	30.4	76.0	32.6	21.2	21.3	25.6	21.6
Head (Calc.)	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0

Cumulative Products

Products	**Weight**	**Assays: U, Th, REE g/t; S %**																					
	%	**S**	**S$^{=}$**	**La**	**Ce**	**Pr**	**Nd**	**Sm**	**Eu**	**Gd**	**Tb**	**Dy**	**Ho**	**Er**	**Tm**	**Yb**	**Lu**	**Y**	**Sc**	**U**	**Th**	**LREE**	**HREE**	**TREE**
S Ro Conc	5.8	30.0	28.5	819.0	1570.0	149.0	464.0	89.4	4.1	64.2	9.3	44.6	6.9	17.3	1.9	12.4	1.6	180.0	25.0	773.0	878.0	3120.5	338.2	3458.7
S Ro Conc + REO Ro Conc1	9.1	19.4	18.4	772.4	1457.9	138.9	430.4	81.2	3.6	56.2	7.7	37.3	5.7	14.5	1.5	10.4	1.4	152.5	25.0	630.2	748.9	2909.4	287.3	3196.7
S Ro Conc + REO Ro Conc1-2	19.1	9.38	8.87	612.3	1146.7	108.5	335.6	61.2	2.9	41.2	5.5	26.0	3.9	10.0	1.0	6.9	1.0	107.3	25.0	431.1	515.0	2292.3	202.8	2495.1
S Ro Conc + REO Ro Conc1-3	24.0	7.70	7.26	1415.7	2676.9	249.9	764.2	135.8	5.7	85.5	10.1	47.4	7.1	18.0	2.0	12.8	1.8	188.1	25.0	652.9	958.0	5273.1	372.8	5646.0
REO Ro Tls	76.0	0.06	0.07	116.0	218.0	19.5	62.0	11.0	0.6	6.2	1.0	4.0	0.5	1.1	0.3	0.9	0.5	26.0	25.0	99.9	81.4	452.1	40.5	492.6

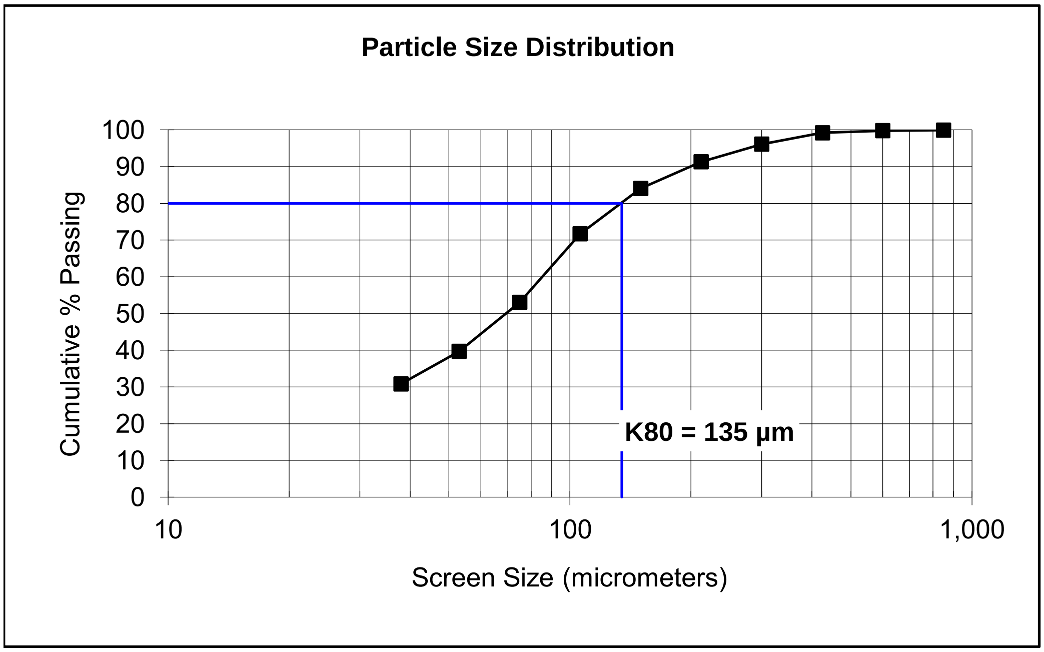
Products	**% Distribution**																						
	S	**S$^{=}$**	**La**	**Ce**	**Pr**	**Nd**	**Sm**	**Eu**	**Gd**	**Tb**	**Dy**	**Ho**	**Er**	**Tm**	**Yb**	**Lu**	**Y**	**Sc**	**U**	**Th**	**LREE**	**HREE**	**TREE**
S Ro Conc	92.2	92.3	11.1	11.3	11.6	11.7	12.7	13.1	14.8	16.9	18.0	19.3	19.5	15.4	19.2	11.4	16.1	5.8	19.3	17.5	11.3	16.4	11.6
S Ro Conc + REO Ro Conc1	93.2	93.2	16.4	16.4	16.9	17.0	18.1	18.2	20.3	22.0	23.6	25.0	25.6	19.5	25.3	15.4	21.4	9.1	24.7	23.4	16.5	21.8	16.8
S Ro Conc + REO Ro Conc1-2	94.5	94.2	27.3	27.1	27.7	27.8	28.5	30.8	31.1	32.6	34.4	35.6	37.0	26.4	35.2	24.0	31.5	19.1	35.3	33.7	27.2	32.2	27.5
S Ro Conc + REO Ro Conc1-3	97.6	97.0	79.4	79.5	80.2	79.6	79.6	74.9	81.4	76.2	78.9	81.7	83.8	68.3	81.8	53.5	69.6	24.0	67.4	78.8	78.7	74.4	78.4
REO Ro Tls	2.4	3.0	20.6	20.5	19.8	20.4	20.4	25.1	18.6	23.8	21.1	18.3	16.2	31.7	18.2	46.5	30.4	76.0	32.6	21.2	21.3	25.6	21.6

SGS Minerals Services
Size Distribution Analysis

Project No. **13849-002**

Sample: **COMB PROD** Test No.: **F6**

Size		Weight	% Retained		% Passing
Mesh	µm	grams	Individual	Cumulative	Cumulative
20	850	0.1	0.1	0.1	99.9
28	600	0.2	0.1	0.2	99.8
35	425	0.9	0.6	0.8	99.2
48	300	4.6	3.1	3.9	96.1
65	212	7.2	4.8	8.7	91.3
100	150	10.9	7.3	15.9	84.1
150	106	18.5	12.3	28.3	71.7
200	75	28.0	18.7	47.0	53.0
270	53	20.0	13.3	60.3	39.7
400	38	13.3	8.9	69.2	30.8
Pan	-38	46.2	30.8	100.0	0.0
Total	**-**	**149.9**	100.0	**-**	**-**
K80	**135**				

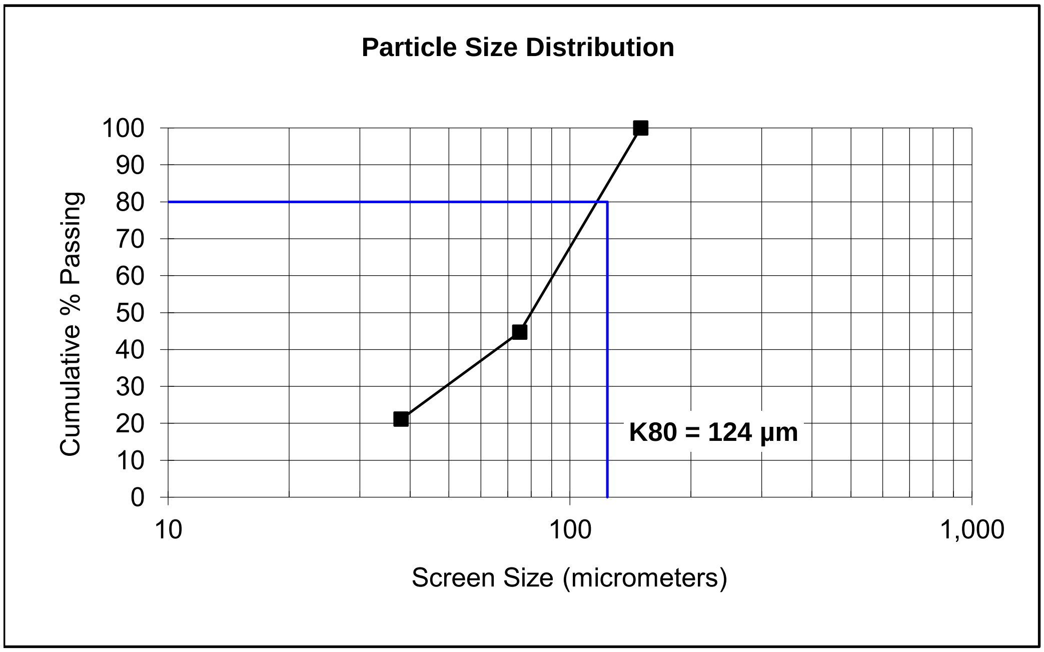


SGS Minerals Services
Size Distribution Analysis

Project No.
13849-002

Sample: **Scav Tail** Test No.: **F6**

Size		Weight	% Retained		% Passing
Mesh	µm	grams	Individual	Cumulative	Cumulative
100	150	0.0	0.0	0.0	100.0
200	75	110.6	55.3	55.3	44.7
400	38	47.1	23.6	78.9	21.2
Pan	-38	42.3	21.2	100.0	0.0
Total	**-**	**200.0**	100.0	**-**	**-**
K80	**124**				



Test No. : F6 **Project No:** 13849-002 **Operator**: Dan Lang **Date**: Jan 09/2013

Purpose: To examine the cleaner effect with duplicate rougher conditions in F1

Procedure: As outlined below.

Feed: 2000 g charge of Master Composite -6 mesh

Grind: 45 minutes at 65% solids in the laboratory Rod Mill GG RM#3

Ro Flotation % Solids = 35

Notes: Flot time and reagent dosage may change based on visual.
LREE=La,Ce,Pr,Nd,Sm,Eu,Sc
HREE=Gd,Tb,Dy,Ho,Er,Tm,Yb,Lu,Y
TREE=LREE+HREE

LR-19: 62% FA2, 27% Aero 855, 9% Kerosene, 2% MIBC

Ro Fd K_{80} = 135 µm

Conditions:

Stage	Reagents Added, g/t										Time, minutes			Pulp		
	Metso	LR19	PAX	DF250							Grind	Cond.	Froth	pH	Temp, ºC	
Grind											45					
Condition 1	500											1		10.2		
Rougher 1		300		2								5	4	9.8		seems a little sluggish compared to F1
Rougher 2		150		6								5	4	9.6		perhaps high density cond and higher collector up front would bbe a benefit
Rougher 3		150		4								5	6	8.8		
Rougher 4		225		4								5	5	7.9		
Rougher 5		300		4								5	5	7.9		need extra stages and additional collector to get a barren tailing
Rougher 6		300		6								5	5	7.8		
Rougher Sub Total	**500**	**1425**		**26**								**31**	**29**			
REO Clnr 1	250											1	6	9.6		
REO Clnr Scav		150										5	5	9.3		
REO Clnr 2	150											1	5	9.7		
REO Clnr 3	100	30										1	5	9.8		
Cleaner Sub Total	**500**	**180**										**8**	**21**			

Stage	Rougher	Cleaner	Cleaner Scavenger
Flotation Cell	4L (2 kg)		
Speed rpm	1800		

Metallurgical Balance

Products	Weight		Assays: U, Th, REE g/t, S %																					
	g	%	S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
REO 3rd Cl Conc	158.6	7.9	22.3	4660	8170	819	2820	422	16	242	28	122	20	46	6	35	5	524	<25	1470	2930	16932	1028	17960
REO 3rd Cl Tail	28.0	1.4	1.46	844	1460	151	523	94	4	66	10	53	9	22	3	20	2	235	<25	1060	1220	3101	420	3521
REO 2nd Cl Tail	42.3	2.1	0.78	457	793	79	277	51	2	38	6	33	6	14	<2	11	<2	143	<25	655	746	1684	255	1939
REO 1st Cl Sc Conc	21.3	1.1	1.16	661	1160	115	401	68	4	47	7	36	6	15	2	13	2	162	<25	693	832	2434	290	2724
REO 1st Cl Sc Tail	159.0	7.9	0.26	157	277	27	106	20	<2	17	3	15	3	7	<2	6	<2	68	<25	358	338	614	123	737
REO Scav Conc	69.7	3.5	0.16	121	225	23	82	19	<2	15	3	16	3	7	<2	8	<2	72	<25	469	258	497	128	625
REO Scav Tail	1526.0	76.1	0.03	<20	28	3	<20	<2	<2	<2	<2	<2	<2	<2	<2	<2	<2	<25	<25	39	21	100	41	141
REO Scav Tail +75um	843.9	42.1	--	28	43	4	<20	3	<2	2	<2	<2	<2	<2	<2	<2	<2	<25	<25	48	25	125	41	166
REO Scav Tail -75+38um	359.4	17.9	--	<20	<20	<2	<20	<2	<2	<2	<2	<2	<2	<2	<2	<2	<2	<25	<25	19	11	91	41	132
REO Scav Tail -38um	322.7	16.1	--	<20	26	3	<20	3	<2	2	<2	2.3	<2	<2	<2	<2	<2	<25	<25	57	32	99	41.3	140.3
Head (Calc.)	2004.9	100.0	1.86	432.3	751.4	75.3	267.0	40.8	3.2	24.8	4.4	14.8	3.8	6.7	2.3	5.7	2.2	76.4	<25	229.8	326.8	1595.0	141.1	1736.1
Head (Direct)			1.92	407.0	767.0	81.0	254.0	42.6	1.90	26.10	3.3	16.50	2.6	6.70	0.80	5.20	0.70	62.00	25.0	270.0	354.0	1578.5	123.9	1702

Products	Weight	% Distribution																					
	%	S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
REO 3rd Cl Conc	7.9	94.7	85.3	86.0	86.1	83.6	81.7	40.1	77.3	50.1	65.3	42.0	54.1	20.4	48.6	17.7	54.3	7.9	50.6	70.9	84.0	57.6	81.8
REO 3rd Cl Tail	1.4	1.1	2.7	2.7	2.8	2.7	3.2	1.8	3.7	3.2	5.0	3.3	4.6	1.8	4.9	1.2	4.3	1.4	6.4	5.2	2.7	4.2	2.8
REO 2nd Cl Tail	2.1	0.9	2.2	2.2	2.2	2.2	2.6	1.3	3.2	2.9	4.7	3.4	4.4	1.8	4.1	1.9	3.9	2.1	6.0	4.8	2.2	3.8	2.4
REO 1st Cl Sc Conc	1.1	0.7	1.6	1.6	1.6	1.6	1.8	1.3	2.0	1.7	2.6	1.7	2.4	0.9	2.4	0.9	2.3	1.1	3.2	2.7	1.6	2.2	1.7
REO 1st Cl Sc Tail	7.9	1.1	2.9	2.9	2.8	3.1	3.9	5.0	5.4	5.4	8.0	6.3	8.3	6.8	8.4	7.1	7.1	7.9	12.4	8.2	3.1	6.9	3.4
REO Scav Conc	3.5	0.3	1.0	1.0	1.1	1.1	1.6	2.2	2.1	2.4	3.8	2.8	3.6	3.0	4.9	3.1	3.3	3.5	7.1	2.7	1.1	3.2	1.3
REO Scav Tail	76.1	1.2	3.5	2.8	3.0	5.7	3.7	48.2	6.1	34.4	10.3	40.5	22.6	65.3	26.7	68.0	24.9	76.1	12.9	4.9	4.8	22.1	6.2
REO Scav Tail +75um	42.1	--	2.7	2.4	2.2	3.2	3.1	26.7	3.4	19.0	5.7	22.4	12.5	36.1	14.8	37.6	13.8	42.1	8.8	3.2	3.3	12.2	4.0
REO Scav Tail -75+38um	17.9	--	0.8	0.5	0.5	1.3	0.9	11.4	1.4	8.1	2.4	9.5	5.3	15.4	6.3	16.0	5.9	17.9	1.5	0.6	1.0	5.2	1.4
REO Scav Tail -38um	16.1	--	0.7	0.6	0.6	1.2	1.2	10.2	1.3	7.3	2.5	8.6	4.8	13.8	5.7	14.4	5.3	16.1	4.0	1.6	1.0	4.7	1.3
Head (Calc.)	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0

Combined Products

Products	Weight	Assays: U, Th, REE g/t, S %																					
	%	S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
REO 3rd Cl Conc	7.9	22.3	4660.0	8170.0	819.0	2820.0	422.0	16.0	242.0	28.0	122.0	20.0	46.0	6.0	35.0	5.0	524.0	25.0	1470.0	2930.0	16932	1028	17960
REO 2nd Cl Conc	9.3	19.2	4087.4	7163.1	718.8	2475.3	372.8	14.2	215.6	25.3	111.6	18.3	42.4	5.5	32.7	4.5	480.6	25.0	1408.5	2673.4	14856.6	936.8	15793.4
REO 1st Cl Conc	11.4	15.8	3416.5	5986.0	600.5	2069.1	313.3	11.9	182.8	21.7	97.1	16.1	37.2	4.9	28.7	4.1	418.2	25.0	1269.2	2317.2	12422.4	810.8	13233.1
REO 1st Cl Conc +1st Cl Sc Conc	12.5	14.5	3181.9	5575.1	559.2	1927.1	292.4	11.3	171.2	20.5	91.9	15.2	35.3	4.6	27.4	3.9	396.4	25.0	1220.2	2190.8	11572.0	766.4	12338.5
REO Ro Conc	20.4	9.0	2006.6	3516.5	352.4	1219.5	186.6	7.7	111.3	13.7	62.0	10.5	24.3	3.6	19.1	3.2	268.8	25.0	885.2	1470.9	7314.1	516.4	7830.6
REO Ro Conc + Ro Sc Conc	23.9	7.7	1732.1	3037.4	304.5	1053.9	162.2	6.8	97.3	12.1	55.3	9.4	21.8	3.4	17.5	3.0	240.2	25.0	824.6	1294.3	6322.0	459.9	6781.9
REO Ro Conc + Ro Sc Conc + Ro Sc Tail -38 um	40.0	--	1042.8	1825.0	183.1	637.7	98.1	4.9	58.9	8.1	34.0	6.4	13.8	2.8	11.2	2.6	153.5	25.0	515.6	786.1	3816.6	291.4	4107.9
Ro Scav Tail +38um	60.0	--	25.6	36.1	3.4	20.0	2.70	2.00	2.00	2.00	2.00	2.00	2.00	2.00	2.00	2.00	25.0	25.0	39.3	20.8	114.8	41.0	155.8

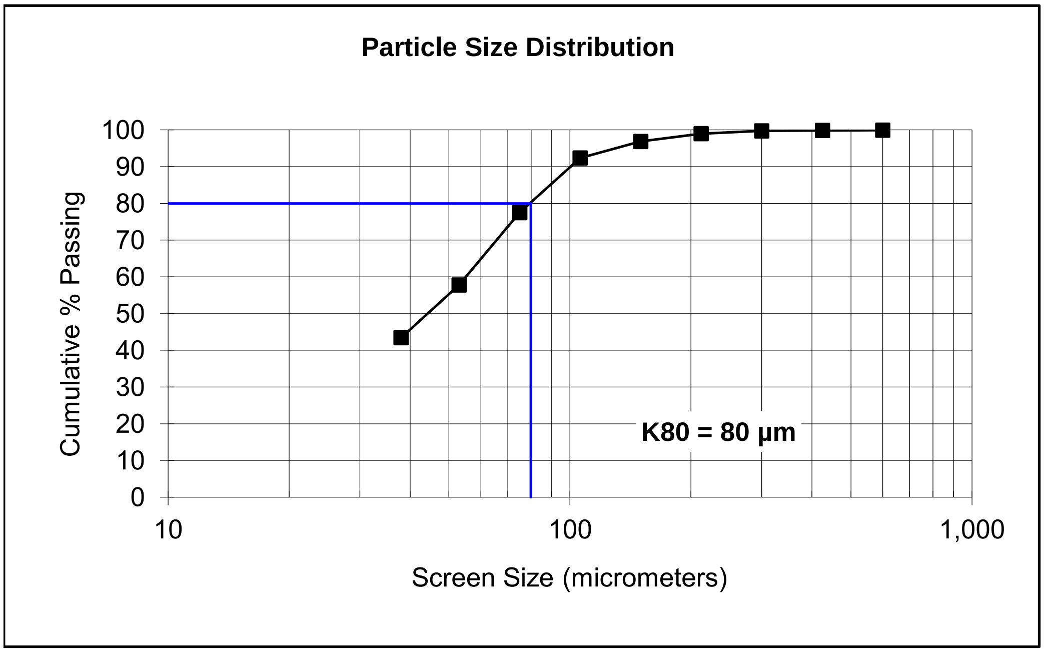
Products	% Distribution																					
	S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
REO 3rd Cl Conc	94.7	85.3	86.0	86.1	83.6	81.7	40.1	77.3	50.1	65.3	42.0	54.1	20.4	48.6	17.7	54.3	7.9	50.6	70.9	84.0	57.6	81.8
REO 2nd Cl Conc	95.8	88.0	88.7	88.9	86.3	84.9	41.9	81.0	53.3	70.3	45.4	58.7	22.2	53.5	18.9	58.6	9.3	57.1	76.1	86.7	61.8	84.7
REO 1st Cl Conc	96.7	90.2	91.0	91.1	88.5	87.6	43.2	84.3	56.1	75.0	48.8	63.1	24.0	57.6	20.8	62.5	11.4	63.1	80.9	88.9	65.6	87.0
REO 1st Cl Conc +1st Cl Sc Conc	97.4	91.8	92.6	92.7	90.1	89.3	44.5	86.3	57.8	77.6	50.4	65.5	24.9	60.0	21.8	64.8	12.5	66.3	83.7	90.5	67.8	88.7
REO Ro Conc	98.5	94.7	95.5	95.6	93.2	93.2	49.6	91.7	63.2	85.6	56.8	73.7	31.7	68.4	28.9	71.8	20.4	78.6	91.9	93.6	74.7	92.1
REO Ro Conc + Ro Sc Conc	98.8	95.7	96.6	96.6	94.3	94.8	51.8	93.9	65.6	89.4	59.5	77.4	34.7	73.3	32.0	75.1	23.9	85.7	94.6	94.7	77.9	93.3
REO Ro Conc + Ro Sc Conc + Ro Sc Tail -38 um	--	96.4	97.1	97.3	95.5	96.0	62.0	95.2	72.8	91.9	68.1	82.1	48.5	78.9	46.4	80.4	40.0	89.7	96.2	95.7	82.6	94.6
Ro Scav Tail +38um	--	3.6	2.9	2.7	4.5	4.0	38.0	4.8	27.2	8.1	31.9	17.9	51.5	21.1	53.6	19.6	60.0	10.3	3.8	4.3	17.4	5.4

SGS Minerals Services
Size Distribution Analysis

Project No.
13849-002

Sample: **COMB PROD** Test No.: **F7**

Size		Weight	% Retained		% Passing
Mesh	µm	grams	Individual	Cumulative	Cumulative
28	600	0.1	0.1	0.1	99.9
35	425	0.1	0.1	0.1	99.9
48	300	0.2	0.1	0.3	99.7
65	212	1.1	0.7	1.0	99.0
100	150	3.2	2.1	3.2	96.8
150	106	6.7	4.5	7.6	92.4
200	75	22.2	14.9	22.5	77.5
270	53	29.4	19.7	42.2	57.8
400	38	21.4	14.3	56.6	43.4
Pan	-38	64.8	43.4	100.0	0.0
Total	-	**149.2**	100.0	-	-
K80	**80**				



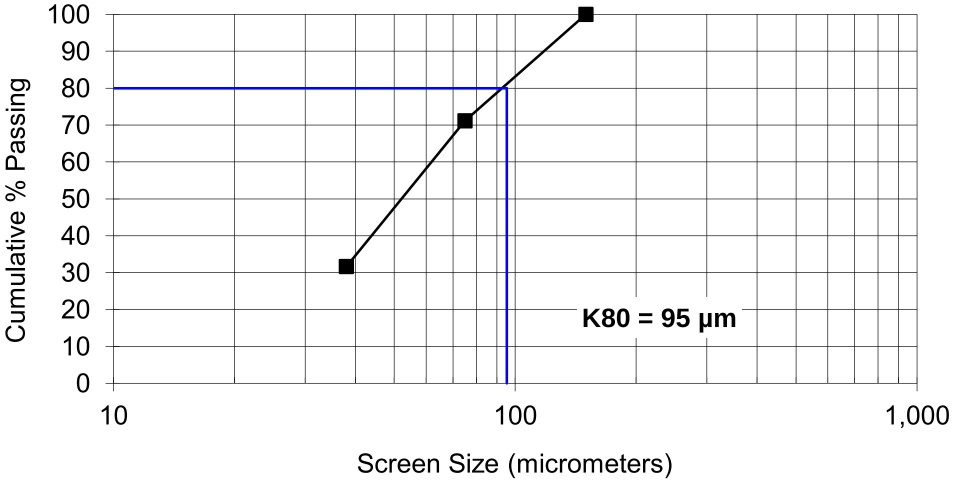
SGS Minerals Services
Size Distribution Analysis

Project No.
13849-002

Sample: **Scav Tail** Test No.: **F7**

Size		Weight	% Retained		% Passing
Mesh	µm	grams	Individual	Cumulative	Cumulative
100	150	0.0	0.0	0.0	100.0
200	75	57.7	28.9	28.9	71.2
400	38	79.0	39.5	68.4	31.7
Pan	-38	63.3	31.7	100.0	0.0
Total	**-**	**200.0**	100.0	**-**	**-**
K80	**95**				

Particle Size Distribution



Test No. : F7 **Project No:** 13849-002 **Operator:** Dan Lang **Date:** Jan 09/2013

Purpose: To examine the cleaner effect with duplicate rougher conditions in F1 but with finer grinding size fraction

Procedure: As outlined below.

Feed: 2000 g charge of Master Composite -6 mesh

Grind: 70 minutes at 65% solids in the laboratory Rod Mill GG RM#3

Ro Flotation % Solids = 35

Notes: Flot time and reagent dosage may change based on visual.

Ro Fd K_{80} = 80 µm

LREE=La,Ce,Pr,Nd,Sm,Eu,Sc
HREE=Gd,Tb,Dy,Ho,Er,Tm,Yb,Lu,Y
TREE=LREE+HREE

LR-19: 62% FA2, 27% Aero 855, 9% Kerosene, 2% MIBC

Conditions:

Stage	Reagents Added, g/t										Time, minutes			Pulp		
	Metso	LR19		DF250							Grind	Cond.	Froth	pH	Temp, ºC	
Grind											70				RT	
Condition 1	500											1			RT	
															RT	
Rougher 1		300										5	4	9.8	RT	more mass floated than F6
Rougher 2		150		6								5	4	9.6	RT	
Rougher 3		150										5	6	8.6	RT	
Rougher 4		225		6								5	5	7.8	RT	
Rougher 5		300		6								5	5	7.8		tailing looks good earlier than F6
Rougher Sub Total	**500**	**1125**		**18**												
REO Clnr 1	250											1	6	9.6		
REO Clnr Scav		150										5	5	9.3		
REO Clnr 2	150											1	5	9.7		
REO Clnr 3	100	30										1	5	9.8		
Cleaner Sub Total	**500**	**180**														

Stage	Rougher	Cleaner	Cleaner Scavenger
Flotation Cell	4L (2 kg)		
Speed rpm	1800		

Metallurgical Balance

Products	Weight		Assays: U, Th, REE g/t, S %																				
	g	%	S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
REO 3rd Cl Conc	164.8	8.2	21.6	4590	8280	787	2780	408	15	248	29	122	20	45	6	34	4	491	<25	1410	2930	16885	999	17884
REO 3rd Cl Tail	32.2	1.6	0.95	484	849	92	316	58	3	46	8	40	7	17	2	15	2	184	<25	798	948	1827	321	2148
REO 2nd Cl Tail	58.0	2.9	0.58	299	537	56	198	40	2	31	6	28	5	13	<2	11	<2	129	<25	603	656	1157	227	1384
REO 1st Cl Sc Conc	55.3	2.8	0.80	397	715	71	258	47	3	39	6	32	6	14	<2	12	<2	143	<25	671	737	1516	256	1772
REO 1st Cl Sc Tail	261.9	13.0	0.18	86	162	16	63	14	<2	12	2	13	2	6	<2	5	<2	61	<25	307	279	368	105	473
REO Ro Tail	1434.8	71.5	0.02	<20	<20	2	<20	<2	<2	<2	<2	<2	<2	<2	<2	<2	<2	<25	<25	25	18	91	41	132
REO Ro Tail +75um	413.9	20.6	--	<20	24	2	<20	<2	<2	<2	<2	<2	<2	<2	<2	<2	<2	<25	<25	22	15	95	41	136
REO Ro Tail -75+38um	566.7	28.2	--	<20	<20	<2	<20	<2	<2	<2	<2	<2	<2	<2	<2	<2	<2	<25	<25	12	9	91	41	132
REO Ro Tail -38um	454.1	22.6	--	<20	22	2	<20	2	<2	2	<2	<2	<2	<2	<2	<2	<2	<25	<25	37	26	93	41	134
Head (Calc.)	2007.0	100.0	1.87	429.8	765.4	73.2	268.7	40.1	3.1	26.1	4.5	15.5	3.8	6.9	2.3	5.8	2.2	76.8	<25	220.9	343.0	1605.3	143.8	1749.2
Head (Direct)			1.92	407.0	767.0	81.0	254.0	42.6	1.90	26.10	3.3	16.50	2.6	6.70	0.80	5.20	0.70	62.00	25.0	270.0	354.0	1578.5	123.9	1702

Products	Weight	% Distribution																				
	%	S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
REO 3rd Cl Conc	8.2	95.1	87.7	88.8	88.3	85.0	83.5	39.6	78.1	52.5	64.7	43.7	53.2	21.2	48.4	15.2	52.5	8.2	52.4	70.1	86.4	57.0	84.0
REO 3rd Cl Tail	1.6	0.8	1.8	1.8	2.0	1.9	2.3	1.5	2.8	2.8	4.1	3.0	3.9	1.4	4.2	1.5	3.8	1.6	5.8	4.4	1.8	3.6	2.0
REO 2nd Cl Tail	2.9	0.9	2.0	2.0	2.2	2.1	2.9	1.9	3.4	3.8	5.2	3.8	5.4	2.5	5.5	2.7	4.9	2.9	7.9	5.5	2.1	4.6	2.3
REO 1st Cl Sc Conc	2.8	1.2	2.5	2.6	2.7	2.6	3.2	2.7	4.1	3.6	5.7	4.4	5.6	2.4	5.7	2.5	5.1	2.8	8.4	5.9	2.6	4.9	2.8
REO 1st Cl Sc Tail	13.0	1.3	2.6	2.8	2.9	3.1	4.6	8.4	6.0	5.7	11.0	7.0	11.3	11.2	11.3	12.1	10.4	13.0	18.1	10.6	3.0	9.5	3.5
REO Ro Tail	71.5	0.8	3.3	1.9	2.0	5.3	3.6	46.0	5.5	31.5	9.2	38.1	20.6	61.4	24.8	66.1	23.3	71.5	8.1	3.8	4.1	20.4	5.4
REO Ro Tail +75um	20.6	--	1.0	0.6	0.6	1.5	1.0	13.3	1.6	9.1	2.7	11.0	5.9	17.7	7.2	19.1	6.7	20.6	2.1	0.9	1.2	5.9	1.6
REO Ro Tail -75+38um	28.2	--	1.3	0.7	0.8	2.1	1.4	18.2	2.2	12.4	3.6	15.0	8.1	24.3	9.8	26.1	9.2	28.2	1.5	0.7	1.6	8.1	2.1
REO Ro Tail -38um	22.6	--	1.1	0.7	0.6	1.7	1.1	14.5	1.7	10.0	2.9	12.1	6.5	19.4	7.9	20.9	7.4	22.6	3.8	1.7	1.3	6.5	1.7
Head (Calc.)	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0

Combined Products

Products	Weight	Assays: U, Th, REE g/t, S %																				
	%	S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
REO 3rd Cl Conc	8.2	21.6	4590.0	8280.0	787.0	2780.0	408.0	15.0	248.0	29.0	122.0	20.0	45.0	6.0	34.0	4.0	491.0	25.0	1410.0	2930.0	16885	999	17884
REO 2nd Cl Conc	9.8	18.2	3918.9	7065.4	673.4	2377.3	350.8	13.0	215.0	25.6	108.6	17.9	40.4	5.3	30.9	3.7	440.8	25.0	1310.0	2606.0	14423.7	888.2	15311.9
REO 1st Cl Conc	12.7	14.2	3095.5	5580.5	533.0	1881.6	280.1	10.5	173.1	21.1	90.3	14.9	34.2	4.6	26.4	3.3	369.9	25.0	1149.2	2162.5	11406.2	737.8	12144.0
REO 1st Cl Conc +1st Cl Sc Conc	15.5	11.8	2614.6	4713.4	450.6	1592.2	238.6	9.2	149.2	18.4	79.9	13.4	30.6	4.1	23.8	3.1	329.5	25.0	1064.0	1908.5	9643.6	651.9	10295.6
REO Ro Conc	28.5	6.5	1457.2	2630.2	251.7	892.3	135.8	5.9	86.4	10.9	49.3	8.2	19.3	3.2	15.2	2.6	206.6	25.0	717.5	1162.6	5398.1	401.6	5799.7
REO Ro Conc + Ro Tail -38 um	51.1	--	821.3	1476.1	141.2	506.3	76.6	4.2	49.1	7.0	28.4	5.4	11.7	2.6	9.4	2.3	126.2	25.0	416.4	659.7	3050.8	242.0	3292.8
Ro Ro Tail +38um	48.9	--	20.0	21.7	2.0	20.0	2.00	2.00	2.00	2.00	2.00	2.00	2.00	2.00	2.00	2.00	25.0	25.0	16.2	11.5	92.7	41.0	133.7

Products	% Distribution																				
	S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
REO 3rd Cl Conc	95.1	87.7	88.8	88.3	85.0	83.5	39.6	78.1	52.5	64.7	43.7	53.2	21.2	48.4	15.2	52.5	8.2	52.4	70.1	86.4	57.0	84.0
REO 2nd Cl Conc	95.9	89.5	90.6	90.3	86.8	85.8	41.1	80.9	55.3	68.9	46.7	57.2	22.5	52.6	16.7	56.4	9.8	58.2	74.6	88.2	60.6	85.9
REO 1st Cl Conc	96.8	91.5	92.6	92.5	89.0	88.7	43.0	84.4	59.1	74.1	50.6	62.6	25.0	58.1	19.3	61.2	12.7	66.1	80.1	90.3	65.2	88.2
REO 1st Cl Conc +1st Cl Sc Conc	98.0	94.1	95.2	95.2	91.6	91.9	45.7	88.5	62.8	79.8	55.0	68.1	27.4	63.9	21.9	66.4	15.5	74.5	86.0	92.9	70.1	91.0
REO Ro Conc	99.2	96.7	98.0	98.0	94.7	96.4	54.0	94.5	68.5	90.8	61.9	79.4	38.6	75.2	33.9	76.7	28.5	92.6	96.6	95.9	79.6	94.5
REO Ro Conc + Ro Tail -38 um	--	97.7	98.6	98.7	96.4	97.6	68.6	96.3	78.5	93.7	74.0	85.9	58.0	83.0	54.8	84.1	51.1	96.4	98.4	97.2	86.1	96.3
Ro Ro Tail +38um	--	2.3	1.4	1.3	3.6	2.4	31.4	3.7	21.5	6.3	26.0	14.1	42.0	17.0	45.2	15.9	48.9	3.6	1.6	2.8	13.9	3.7

Test No. : F8 **Project No:** 13849-002 **Operator**: Dan Lang **Date**: 11-Jan-12

Purpose: To examine effect of aggressive LR19 and less stage in Rougher and investigate the pre-float pyrite and clean the pyrite and rougher concentrate

Procedure: As outlined below.

Feed: 4000 g charge of Master Composite

Grind: 50 minutes at 65% solids in the laboratory Rod Mill (RM#2) Ro Flotation % Solids = 35

Notes: Flot time and reagent dosage may change based on visual. Ro Fd K_{80} = 111 µm

LREE=La,Ce,Pr,Nd,Sm,Eu,Sc
HREE=Gd,Tb,Dy,Ho,Er,Tm,Yb,Lu,Y
TREE=LREE+HREE

Conditions:

Stage	Reagents Added, g/t										Time, minutes			Pulp		
	KAX	DF250	LR19	H_2SO_4	Metso						Grind	Cond.	Froth	pH	Temp, ºC	
Grind											50					2 x 2 kg grinds combined for float feed
Sulphide Rougher	50	4										1	5	8.4		
Sulphide Cleaner													4			
Sulphide Rougher Tailings Agitate, settle and decant in a 14L decantation cylinder for 12 minutes, and decant to 5 L mark, transfer to 10L cell for high density conditioning																
REO Cond 1					500							1		10.2	18	
REO Cond 2			500									5				
REO Rougher 1													4	9.8	19	
REO Rougher 2												5	5	9.4		
REO Rougher 3												5	7	8.6		
REO Rougher 4												5	6	7.8	19	
Combine REO Ro Conc 1-4 for cleaning																
REO 1st Clnr				50	150							1	7	7.9		some coarse drop out lots of fine silicates
			100									5	3			
REO Clnr Scav			75									5	4			
REO 2nd Clnr				40	100							1	6	7.7		Nice REO conc
REO 3rd Clnr				20	75							1	4	7.7		Nice REO conc
REO Cleaner Subtol			**175**	**110**	**325**											

Stage	Rougher	Cleaner	Cleaner Scavenger
Flotation Cell	4L (2 kg)	2L (1 kg)	2L (1 kg)
Speed rpm	1800	1500	1500

Metallurgical Balance

Products	Weight		Assays: U, Th, REE g/t, S %																				
	g	%	S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
REO 3rd Cl Conc	249.6	6.3	1.55	5230	9100	974	3090	500	21	299	34	166	23	56	7	41	7	614	<25	1820	3210	18940	1247	20187
REO 3rd Cl Tail	79.1	2.0	0.50	530	928	99	317	57	3	40	4	32	5	12	<2	10	2	122	<25	552	543	1959	229	2188
REO 2nd Cl Tail	111.2	2.8	0.31	211	380	41	127	25	<2	18	<2	17	3	6	<2	5	<2	64	<25	313	276	811	119	930
REO 1st Cl Sc Conc	59.1	1.5	0.20	134	241	26	85	19	<2	14	<2	15	2	6	<2	5	<2	62	<25	340	221	532	110	642
REO 1st Cl Sc Tail	207.3	5.2	0.10	54	95	10	33	7	<2	4	<2	6	<2	2	<2	<2	<2	23	<25	127	87	226	45	271
REO Ro Tail	2798.6	70.2	0.07	<20	25	3	<20	2	<2	<2	<2	<2	<2	<2	<2	<2	<2	<10	<25	48	22	97	26	123
Slime	273.6	6.9	0.57	516	880	94	303	51	2	32	2	21	3	8	<2	6	<2	79	<25	308	367	1871	155	2026
Sulph Cl Conc	143.9	3.6	44.5	579	1020	110	351	65	3	46	5	37	6	14	<2	10	<2	143	<25	636	664	2153	265	2418
Sulph Cl Tail	65.3	1.6	7.02	842	1460	159	511	90	4	61	7	45	7	16	2	12	<2	177	<25	630	881	3091	329	3420
Head (Calc.)	3987.7	100.0	1.93	432.7	745.7	80.1	262.1	42.5	3.3	26.7	4.2	16.9	3.7	6.8	2.3	5.5	2.3	65.2	<25	233.3	306.2	1591.4	133.7	1725.1
Head (Direct)			1.92	407.0	767.0	81.0	254.0	42.6	1.90	26.10	3.3	16.50	2.6	6.70	0.80	5.20	0.70	62.00	25.0	270.0	354.0	1578.5	123.9	1702

Products	Weight	% Distribution																				
	%	S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
REO 3rd Cl Conc	6.3	5.0	75.7	76.4	76.1	73.8	73.7	40.1	70.1	50.3	61.3	38.9	51.4	18.9	47.0	18.9	58.9	6.3	48.8	65.6	74.5	58.4	73.2
REO 3rd Cl Tail	2.0	0.5	2.4	2.5	2.5	2.4	2.7	1.8	3.0	1.9	3.7	2.7	3.5	1.7	3.6	1.7	3.7	2.0	4.7	3.5	2.4	3.4	2.5
REO 2nd Cl Tail	2.8	0.4	1.4	1.4	1.4	1.4	1.6	1.7	1.9	1.3	2.8	2.3	2.5	2.4	2.6	2.4	2.7	2.8	3.7	2.5	1.4	2.5	1.5
REO 1st Cl Sc Conc	1.5	0.2	0.5	0.5	0.5	0.5	0.7	0.9	0.8	0.7	1.3	0.8	1.3	1.3	1.4	1.3	1.4	1.5	2.2	1.1	0.5	1.2	0.6
REO 1st Cl Sc Tail	5.2	0.3	0.6	0.7	0.6	0.7	0.9	3.2	0.8	2.5	1.8	2.8	1.5	4.5	1.9	4.5	1.8	5.2	2.8	1.5	0.7	1.7	0.8
REO Ro Tail	70.2	2.5	3.2	2.4	2.6	5.4	3.3	42.8	5.3	33.2	8.3	38.0	20.6	60.7	25.7	60.7	10.8	70.2	14.4	5.0	4.3	13.6	5.0
Slime	6.9	2.0	8.2	8.1	8.1	7.9	8.2	4.2	8.2	3.2	8.5	5.6	8.0	5.9	7.5	5.9	8.3	6.9	9.1	8.2	8.1	8.0	8.1
Sulph Cl Conc	3.6	83.1	4.8	4.9	5.0	4.8	5.5	3.3	6.2	4.3	7.9	5.9	7.4	3.1	6.6	3.1	7.9	3.6	9.8	7.8	4.9	7.2	5.1
Sulph Cl Tail	1.6	5.9	3.2	3.2	3.3	3.2	3.5	2.0	3.7	2.7	4.3	3.1	3.8	1.4	3.6	1.4	4.4	1.6	4.4	4.7	3.2	4.0	3.2
Head (Calc.)	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0

Combined Products

Products	Weight	Assays: U, Th, REE g/t, S %																				
	%	S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
REO 3rd Cl Conc	6.3	1.55	5230.0	9100.0	974.0	3090.0	500.0	21.0	299.0	34.0	166.0	23.0	56.0	7.0	41.0	7.0	614.0	25.0	1820.0	3210.0	18940	1247	20187
REO 2nd Cl Conc	8.2	1.30	4099.0	7133.4	763.4	2422.7	393.4	16.7	236.7	26.8	133.8	18.7	45.4	5.8	33.5	5.8	495.6	25.0	1514.9	2568.2	14853.6	1002.0	15855.6
REO 1st Cl Conc	11.0	1.05	3116.1	5426.3	580.8	1842.4	300.3	13.0	181.4	20.5	104.2	14.7	35.4	4.8	26.3	4.8	386.5	25.0	1211.0	1988.8	11303.9	778.8	12082.7
REO 1st Cl Conc +1st Cl Sc Conc	12.5	0.95	2763.0	4812.2	515.1	1634.2	267.0	11.7	161.6	18.3	93.7	13.2	32.0	4.5	23.8	4.5	348.1	25.0	1107.9	1779.4	10028.1	699.6	10727.7
REO Ro Conc	17.7	0.70	1967.9	3427.7	366.9	1164.3	190.7	8.8	115.3	13.5	67.9	9.9	23.2	3.8	17.4	3.8	252.7	25.0	820.0	1282.7	7151.1	507.5	7658.6
REO Ro Conc + Slime	24.6	0.66	1562.5	2716.3	290.7	923.8	151.7	6.9	92.1	10.3	54.8	8.0	18.9	3.3	14.2	3.3	204.2	25.0	677.0	1027.0	5676.9	409.1	6085.9
REO Ro Conc + Slime + S Cl Tail	26.2	1.06	1517.5	2637.8	282.4	898.0	147.8	6.7	90.1	10.1	54.2	7.9	18.7	3.2	14.1	3.2	202.5	25.0	674.1	1017.9	5515.3	404.1	5919.4
REO Ro Conc + Slime + S	29.8	6.32	1403.9	2442.0	261.6	831.8	137.79	6.29	84.78	9.49	52.14	7.69	18.17	3.05	13.59	3.05	195.3	25.0	669.5	975.1	5108.4	387.2	5495.6

Products	% Distribution																					
	S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
REO 3rd Cl Conc	5.0	75.7	76.4	76.1	73.8	73.7	40.1	70.1	50.3	61.3	38.9	51.4	18.9	47.0	18.9	58.9	6.3	48.8	65.6	74.5	58.4	73.2
REO 2nd Cl Conc	5.5	78.1	78.8	78.6	76.2	76.3	41.9	73.1	52.2	65.0	41.6	54.9	20.7	50.7	20.7	62.6	8.2	53.5	69.1	76.9	61.8	75.8
REO 1st Cl Conc	6.0	79.4	80.3	80.0	77.6	78.0	43.6	75.0	53.5	67.8	43.9	57.3	23.1	53.2	23.1	65.3	11.0	57.3	71.7	78.4	64.3	77.3
REO 1st Cl Conc +1st Cl Sc Conc	6.1	79.9	80.7	80.5	78.0	78.6	44.5	75.8	54.2	69.2	44.7	58.6	24.4	54.6	24.4	66.8	12.5	59.4	72.7	78.9	65.5	77.8
REO Ro Conc	6.4	80.6	81.4	81.1	78.7	79.5	47.7	76.5	56.6	71.0	47.5	60.1	28.8	56.5	28.8	68.6	17.7	62.2	74.2	79.6	67.2	78.6
REO Ro Conc + Slime	8.4	88.7	89.5	89.2	86.6	87.7	51.9	84.8	59.9	79.5	53.1	68.2	34.8	64.1	34.8	76.9	24.6	71.3	82.4	87.7	75.2	86.7
REO Ro Conc + Slime + S Cl Tail	14.4	91.9	92.7	92.4	89.8	91.2	53.9	88.5	62.6	83.8	56.2	72.0	36.2	67.7	36.2	81.3	26.2	75.7	87.1	90.8	79.2	89.9
REO Ro Conc + Slime + S	97.5	96.8	97.6	97.4	94.6	96.7	57.2	94.7	66.8	91.7	62.0	79.4	39.3	74.3	39.3	89.2	29.8	85.6	95.0	95.7	86.4	95.0

Test No. : F9 **Project No:** 13849-002 **Operator**: Dan Lang **Date**: 29-Jan-13

Purpose: To produce concentrate for hydrometallurgical tests with the large cell under similar conditions with F6

Procedure: As outlined below.

Feed: 10,000 g charge of LR Composite

Grind: 70 minutes at 65% solids in the laboratory 10kg Rod Mill Ro Flotation % Solids = 35

Notes: Flot time and reagent dosage may change based on visual. Ro Fd K_{80} = 131 µm (Malvern)
LREE=La,Ce,Pr,Nd,Sm,Eu,Sc
HREE=Gd,Tb,Dy,Ho,Er,Tm,Yb,Lu,Y
TREE=LREE+HREE

Conditions:

Stage	Reagents Added, g/t										Time, minutes			Pulp	
	LR19	Metso									Grind	Cond.	Froth	pH	Temp, ºC
Grind											70				
REO Cond 1												1			
REO Cond 2		500										5			
REO Rougher 1	400											5	7	9.7	19
REO Rougher 2	250											5	7	9.3	
REO Rougher 3	150											5	8		
REO Rougher 4	150											5	8		
REO Rougher 5	150											5	10		
REO Rougher Subtol	**1100**	**500**										**31**	**40**		

Stage	Rougher
Flotation Cell	(~25L 10 kg)
Speed rpm	1800

Metallurgical Balance

Products	Weight		Assays: U, Th, REE g/t, S %																			
	g	%	S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
REO Ro Conc	3452	32.9	11.0	1170	2060	214	731	114	6.2	80.3	10.5	47.7	7.6	19.2	2.6	14.6	1.7	198	<25	1270	1030	4330.2	382.2	4712.4
REO Ro Tail	7030	67.1	0.08	27.3	47.3	4.5	16.5	3.8	0.4	3.3	0.5	2.7	0.5	1.2	<0.3	1.0	<0.5	<10	<25	106	36.3	124.8	20	144.8
Head (Calc.)	10482	100.0	3.68	403.6	710.1	73.5	251.8	40.1	2.3	28.7	3.8	17.5	2.8	7.1	1.1	5.5	0.9	71.9	<25	489.3	363.6	1506.5	139.3	1645.7
Head (Direct)			3.32	352.0	681.0	73.5	235.0	41.9	2.7	28.5	4.2	19.2	3.4	8.2	1.3	5.9	1.0	65.0	<25	520.0	350.0	1411.1	136.7	1547.8

Products	Weight	% Distribution																				
	%	S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
REO Ro Conc	32.9	98.5	95.5	95.5	95.9	95.6	93.6	88.4	92.3	91.2	89.7	88.2	88.7	81.0	87.8	62.5	90.7	40.7	85.5	93.3	94.5	90.4	94.1
REO Ro Tail	67.1	1.5	4.5	4.5	4.1	4.4	6.4	11.6	7.7	8.8	10.3	11.8	11.3	19.0	12.2	37.5	9.3	59.3	14.5	6.7	5.5	9.6	5.9
Head (Calc.)	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0

Result Analysis Report

Sample Name: 13849-002 F9 Feed - Average	**SOP Name:** default 2	**Measured:** January 29, 2013 10:43:40 AM
Sample Source & type: Factory	**Measured by:** LR_Malvern1	**Analysed:** January 29, 2013 10:43:42 AM
Sample bulk lot ref: td	**Result Source:** Averaged	

Particle Name: Default	**Accessory Name:** Hydro 2000G (A)	**Analysis model:** General purpose	**Sensitivity:** Normal
Particle RI: 1.520	**Absorption:** 0.1	**Size range:** 0.020 to 2000.000 um	**Obscuration:** 14.24 %
Dispersant Name: Water	**Dispersant RI:** 1.330	**Weighted Residual:** 0.685 %	**Result Emulation:** Off

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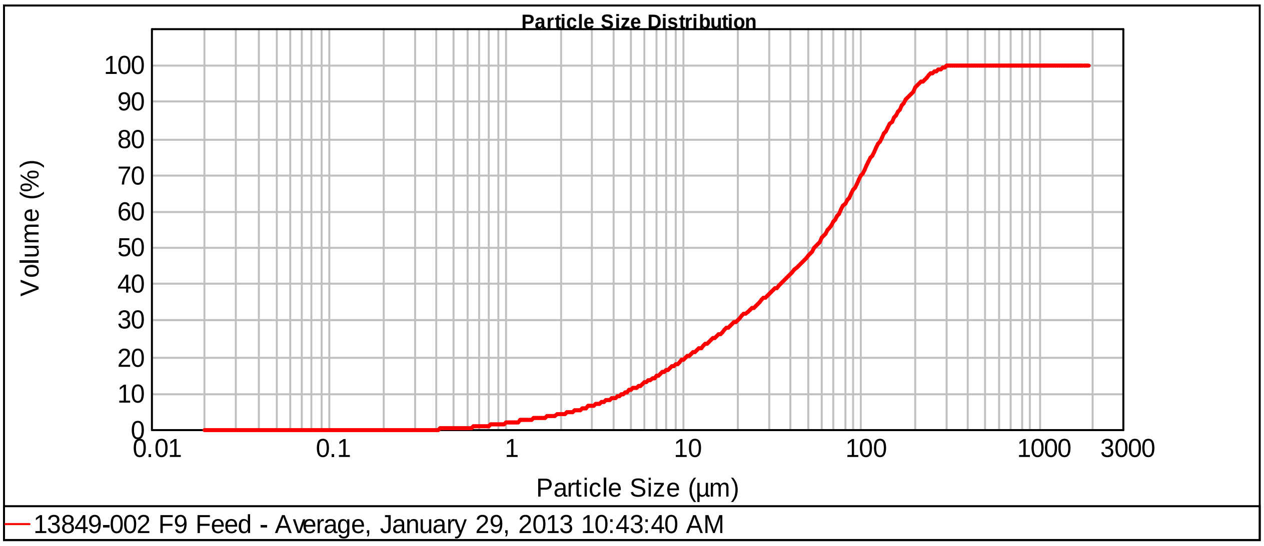
Concentration: 0.0246 %Vol	**Span :** 3.111	**Uniformity:** 0.996	**Result units:** Volume
Specific Surface Area: 0.549 m²/g	**Surface Weighted Mean D[3,2]:** 10.924 um	**Vol. Weighted Mean D[4,3]:** 74.236 um	

d(0.1): 4.702 um **d(0.5): 55.064 um** **D(0.80) : 130.89 µm**



Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %
0.010	0.00	0.105	0.00	1.096	1.93	11.482	21.13	120.226	76.77	1258.925	100.00
0.011	0.00	0.120	0.00	1.259	2.34	13.183	23.20	138.038	81.96	1445.440	100.00
0.013	0.00	0.138	0.00	1.445	2.78	15.136	25.35	158.489	86.76	1659.587	100.00
0.015	0.00	0.158	0.00	1.660	3.26	17.378	27.56	181.970	90.95	1905.461	100.00
0.017	0.00	0.182	0.00	1.905	3.81	19.953	29.84	208.930	94.38	2187.762	100.00
0.020	0.00	0.209	0.00	2.188	4.44	22.909	32.17	239.883	96.99	2511.886	100.00
0.023	0.00	0.240	0.00	2.512	5.17	26.303	34.58	275.423	98.80	2884.032	100.00
0.026	0.00	0.275	0.00	2.884	6.01	30.200	37.06	316.228	99.89	3311.311	100.00
0.030	0.00	0.316	0.00	3.311	6.97	34.674	39.67	363.078	100.00	3801.894	100.00
0.035	0.00	0.363	0.00	3.802	8.07	39.811	42.44	416.869	100.00	4365.158	100.00
0.040	0.00	0.417	0.00	4.365	9.29	45.709	45.45	478.630	100.00	5011.872	100.00
0.046	0.00	0.479	0.09	5.012	10.64	52.481	48.76	549.541	100.00	5754.399	100.00
0.052	0.00	0.550	0.27	5.754	12.12	60.256	52.46	630.957	100.00	6606.934	100.00
0.060	0.00	0.631	0.52	6.607	13.72	69.183	56.60	724.436	100.00	7585.776	100.00
0.069	0.00	0.724	0.83	7.586	15.42	79.433	61.18	831.764	100.00	8709.636	100.00
0.079	0.00	0.832	1.17	8.710	17.23	91.201	66.17	954.993	100.00	10000.000	100.00
0.091	0.00	0.955	1.54	10.000	19.13	104.713	71.43	1096.478	100.00		

Operator notes:

Test No. : F10 **Project No:** 13849-002 **Operator**: Dan Lang **Date**: 29-Jan-13

Purpose: To produce concentrate for hydrometallurgical tests with the large cell under similar conditions with F7

Procedure: As outlined below.

Feed: 10,000 g charge of LR Composite

Grind: 90 minutes at 65% solids in the laboratory 10kg Rod Mill — Ro Flotation % Solids = 35

Notes: Flot time and reagent dosage may change based on visual. — Ro Fd K_{80} = 100 µm (Malvern)
LREE=La,Ce,Pr,Nd,Sm,Eu,Sc
HREE=Gd,Tb,Dy,Ho,Er,Tm,Yb,Lu,Y
TREE=LREE+HREE

Conditions:

Stage	Reagents Added, g/t										Time, minutes			Pulp	
	LR19	Metso									Grind	Cond.	Froth	pH	Temp, ºC
Grind											90				
REO Cond 1												1			
REO Cond 2		500													
REO Rougher 1	400											5	9	9.7	19
REO Rougher 2	250											5	15		
REO Rougher 3	250											5	19		
Filtered concentrate ~ 64% weight, too much, Reulp conc and clean															
REO Cleaner		400											10	9.6	
	150	200										5	10		
	50	100										5	3		
	50											5	3		
	50											5	3	9.6	
	50											5	2		
REO Rougher Subtol	**900**	**500**													

Barely any air on, poor concentrate grade, lots of entrainment

Stage	Rougher
Flotation Cell	(~25L 10 kg)
Speed rpm	1800

Metallurgical Balance

Products	Weight		Assays: U, Th, REE g/t, S %																				
	g	%	S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
REO Clnr Conc	2368	22.7	16.1	1390	2580	260	852	139	6.5	85.7	9.8	49.4	7.6	19	2.4	13.4	1.8	191	<25	1100	1020	5252.5	380.1	5632.6
REO Clnr Tail	4211	40.4	0.20	63	121	13.7	46	11.9	0.9	9.7	1.8	9.7	1.7	4.8	0.7	3.5	0.6	42	<25	405	161	281.5	74.5	356
REO Ro Tail	3836	36.8	0.27	56	98.5	9.7	32.5	7.7	0.6	7.8	1.2	6.9	1	2.7	0.3	2.4	<0.5	26	<25	252	98.5	230	48.8	278.8
Head (Calc.)	10415	100.0	3.84	362.1	671.8	68.2	224.3	39.3	2.1	26.3	3.4	17.7	2.8	7.3	0.9	5.3	0.8	70.0	<25	506.7	333.3	1392.8	134.5	1527.3
Head (Direct)			3.32	352.0	681.0	73.5	235.0	41.9	2.7	28.5	4.2	19.2	3.4	8.2	1.3	5.9	1.0	65.0	<25	520.0	350.0	1411.1	136.7	1547.8

Products	Weight	% Distribution																				
	%	S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
REO Clnr Conc	22.7	95.3	87.3	87.3	86.6	86.4	80.5	71.6	74.1	65.6	63.5	62.1	59.5	58.1	57.0	49.0	62.1	22.7	49.4	69.6	85.7	64.2	83.9
REO Clnr Tail	40.4	2.1	7.0	7.3	8.1	8.3	12.3	17.6	14.9	21.4	22.2	24.7	26.7	30.1	26.5	29.0	24.3	40.4	32.3	19.5	8.2	22.4	9.4
REO Ro Tail	36.8	2.6	5.7	5.4	5.2	5.3	7.2	10.7	10.9	13.0	14.4	13.2	13.7	11.8	16.5	22.0	13.7	36.8	18.3	10.9	6.1	13.4	6.7
Head (Calc.)	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0

Combined Products

Products	Weight	Assays: U, Th, REE g/t, S %																				
	%	S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
REO Ro Conc	63.2	5.92	540.6	1006.1	102.4	336.1	57.6	2.9	37.1	4.7	24.0	3.8	9.9	1.3	7.1	1.0	95.6	25.0	655.2	470.2	2070.7	184.5	2255.2
REO Ro Tail	36.8	0.27	56.0	98.5	9.7	32.5	7.70	0.60	7.80	1.20	6.90	1.00	2.70	0.30	2.40	0.50	26.0	25.0	252.0	98.5	230.0	48.8	278.8

Products	% Distribution																					
	S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
REO Ro Conc	97.4	94.3	94.6	94.8	94.7	92.8	89.3	89.1	87.0	85.6	86.8	86.3	88.2	83.5	78.0	86.3	63.2	81.7	89.1	93.9	86.6	93.3
REO Ro Tail	2.6	5.7	5.4	5.2	5.3	7.2	10.7	10.9	13.0	14.4	13.2	13.7	11.8	16.5	22.0	13.7	36.8	18.3	10.9	6.1	13.4	6.7

Test No. : F11 **Project No:** 13849-002 **Operator**: Dan Lang **Date**: Feb 04/2013

Purpose: To examine the oxalic acid effects on depressing gangue minerals with duplicated F6 rougher conditions

Procedure: As outlined below.

Feed: 2000 g charge of LR Composite -6 mesh

Grind: 46 minutes at 65% solids in the laboratory Rod Mill GG RM#1 Ro Flotation % Solids = 46

Notes: Flot time and reagent dosage may change based on visual. Ro Fd K_{80} = 135 µm

LREE=La,Ce,Pr,Nd,Sm,Eu,Sc LR-19: 62% FA2, 27% Aero 855, 9% Kerosene, 2% MIBC
HREE=Gd,Tb,Dy,Ho,Er,Tm,Yb,Lu,Y
TREE=LREE+HREE

Conditions:

Stage	Reagents Added, g/t										Time, minutes			Pulp		
	LR19	METSO	Oxalic acid								Grind	Cond.	Froth	pH	Temp, ºC	
Grind											46					
Condition 1		500	500									1+3		10.5-9.0	17	High density conditioning
Rougher 1	300											5	4	8.7	25	
Rougher 2	150		100									5	4	7.4		
Rougher 3	150		100									5	4	7.2	23	
Rougher 4	300		100									5	5	6.6	24	
Rougher 5	300		100									5	6	6.5		Barren Tail
Rougher Sub Total	**1200**	**500**	**900**									**29**	**23**			

Stage	Rougher
Flotation Cell	4L (2 kg)
Speed rpm	1800

Metallurgical Balance

Products	Weight		Assays: U, Th, REE g/t, S %																			
	g	%	S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
REO Ro Conc 1	284.7	14.2	22.7	2150	3990	390	1390	216	10	120	14	64	9	23	3	19	2	264	<100	1270	1330	8246	518	8764
REO Ro Conc 2	124.8	6.2	0.83	299	585	61	220	53	3	46	8	49	8	20	2	17	2	198	<100	1600	890	1321	350	1671
REO Ro Conc 3	119.9	6.0	0.37	175	347	36	139	35	2	29	5	28	5	13	<2	12	<2	126	<100	1200	457	834	222	1056
REO Ro Conc 4	104.8	5.2	0.23	122	243	25	91	25	<2	21	4	21	3	8	<2	9	<2	84	<100	808	271	608	154	762
REO Ro Conc 5	56.5	2.8	0.16	82	159	18	65	15	2	10	4	15	3	7	<2	6	<2	50	<100	521	164	441	99	540
REO Ro Tls	1316.1	65.6	0.03	<20	25	2	<20	<2	<2	<2	<2	2	<2	<2	<2	<2	<2	<40	<100	66	26	171	56	227
Head (Calc.)	2006.8	100.0	3.33	355.9	656.7	64.4	238.9	39.1	3.2	24.3	4.4	16.6	3.6	7.2	2.1	6.4	2.0	89.3	<100	451.5	307.2	1458.1	156.1	1614.2
Head (Direct)			3.32	352.0	681.0	73.5	235.0	41.9	2.7	28.5	4.2	19.2	3.4	8.2	1.3	5.9	1.0	65.0	<100	520.0	350.0	1486.1	136.7	1622.8

Products	Weight	% Distribution																				
	%	S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
REO Ro Conc 1	14.2	96.7	85.7	86.2	85.9	82.5	78.4	44.4	70.0	45.0	54.6	35.2	45.3	19.9	42.0	14.2	41.9	14.2	39.9	61.4	80.2	47.1	77.0
REO Ro Conc 2	6.2	1.5	5.2	5.5	5.9	5.7	8.4	5.8	11.8	11.3	18.3	13.7	17.3	5.8	16.5	6.2	13.8	6.2	22.0	18.0	5.6	13.9	6.4
REO Ro Conc 3	6.0	0.7	2.9	3.2	3.3	3.5	5.4	3.7	7.1	6.8	10.1	8.2	10.8	5.6	11.2	6.0	8.4	6.0	15.9	8.9	3.4	8.5	3.9
REO Ro Conc 4	5.2	0.4	1.8	1.9	2.0	2.0	3.3	3.3	4.5	4.7	6.6	4.3	5.8	4.9	7.3	5.2	4.9	5.2	9.3	4.6	2.2	5.2	2.5
REO Ro Conc 5	2.8	0.1	0.6	0.7	0.8	0.8	1.1	1.8	1.2	2.6	2.5	2.3	2.7	2.6	2.6	2.8	1.6	2.8	3.2	1.5	0.9	1.8	0.9
REO Ro Tls	65.6	0.6	3.7	2.5	2.0	5.5	3.4	41.0	5.4	29.7	7.9	36.2	18.2	61.2	20.4	65.6	29.4	65.6	9.6	5.6	7.7	23.5	9.2
Head (Calc.)	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0

Cumulative Products

Products	Weight	Assays: U, Th, REE g/t, S %																				
	%	S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
Ro Conc 1	14.2	22.7	2150.0	3990.0	390.0	1390.0	216.0	10.0	120.0	14.0	64.0	9.0	23.0	3.0	19.0	2.0	264.0	100.0	1270.0	1330.0	8246	518	8764
Ro Conc 1-2	20.4	16.0	1585.9	2952.3	289.7	1033.4	166.3	7.9	97.4	12.2	59.4	8.7	22.1	2.7	18.4	2.0	243.9	100.0	1370.6	1195.9	6135.5	466.8	6602.3
Ro Conc 1-3	26.4	12.5	1266.3	2362.2	232.3	830.9	136.6	6.5	81.9	10.5	52.3	7.9	20.0	2.5	16.9	2.0	217.2	100.0	1331.9	1028.6	4934.8	411.4	5346.2
Ro Conc 1-4	31.6	10.5	1077.2	2012.0	198.0	708.6	118.1	5.8	71.9	9.5	47.1	7.1	18.0	2.4	15.6	2.0	195.2	100.0	1245.4	903.4	4219.8	368.8	4588.7
Ro Conc 1-5	34.4	9.62	995.8	1860.5	183.3	655.9	109.7	5.5	66.8	9.0	44.5	6.7	17.1	2.4	14.8	2.0	183.3	100.0	1186.1	842.9	3910.7	346.8	4257.5
REO Ro Tls	65.6	0.03	20.0	25.0	2.0	20.0	2.00	2.00	2.00	2.00	2.00	2.00	2.00	2.00	2.00	2.00	40.0	100.0	66.0	26.0	171.0	56.0	227.0

Products	% Distribution																				
	S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
Ro Conc 1	96.7	85.7	86.2	85.9	82.5	78.4	44.4	70.0	45.0	54.6	35.2	45.3	19.9	42.0	14.2	41.9	14.2	39.9	61.4	80.2	47.1	77.0
Ro Conc 1-2	98.2	90.9	91.7	91.8	88.3	86.9	50.2	81.8	56.2	72.9	48.9	62.5	25.7	58.5	20.4	55.7	20.4	61.9	79.4	85.9	61.0	83.5
Ro Conc 1-3	98.9	93.9	94.9	95.1	91.8	92.2	53.9	88.9	63.0	83.0	57.2	73.3	31.3	69.6	26.4	64.1	26.4	77.8	88.3	89.3	69.5	87.4
Ro Conc 1-4	99.3	95.7	96.8	97.2	93.7	95.6	57.2	93.4	67.7	89.6	61.5	79.1	36.1	76.9	31.6	69.1	31.6	87.2	92.9	91.5	74.7	89.8
Ro Conc 1-5	99.4	96.3	97.5	98.0	94.5	96.6	59.0	94.6	70.3	92.1	63.8	81.8	38.8	79.6	34.4	70.6	34.4	90.4	94.4	92.3	76.5	90.8
REO Ro Tls	0.6	3.7	2.5	2.0	5.5	3.4	41.0	5.4	29.7	7.9	36.2	18.2	61.2	20.4	65.6	29.4	65.6	9.6	5.6	7.7	23.5	9.2

Test No. : F12 **Project No:** 13849-002 **Operator**: Dan Lang **Date**: Feb 04/2013

Purpose: To examine the sodium fluorosilicate effects on depressing gangue minerals with duplicated F6 rougher conditions

Procedure: As outlined below.

Feed: 2000 g charge of LR Composite -6 mesh

Grind: 46 minutes at 65% solids in the laboratory Rod Mill GG RM#1

Ro Flotation % Solids = 46

Notes: Flot time and reagent dosage may change based on visual.

Ro Fd K_{80} = 135 µm

LREE=La,Ce,Pr,Nd,Sm,Eu,Sc

LR-19: 62% FA2, 27% Aero 855, 9% Kerosene, 2% MIBC

HREE=Gd,Tb,Dy,Ho,Er,Tm,Yb,Lu,Y

TREE=LREE+HREE

Conditions:

Stage	Reagents Added, g/t										Time, minutes			Pulp		
	LR19	METSO	Na_2SiF_6								Grind	Cond.	Froth	pH	Temp, ºC	
Grind											46					
Condition 1		500	500									1+3		10.4-6.0	18	High density conditioning
Rougher 1	300											5	4.5	6.4	23	
Rougher 2	150		100									5	4	6.3		
Rougher 3	150		100									5	4	6.6		
Rougher 4	300		100									5	5	6.6		Nice Concentrate
Rougher 5	300		100									5	6	6.3		
Rougher Sub Total	**1200**	**500**	**900**									**29**	**23.5**			

Stage	Rougher
Flotation Cell	4L (2 kg)
Speed rpm	1800

Metallurgical Balance

Products	Weight		Assays: U, Th, REE g/t, S %																				
	g	%	S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
REO Ro Conc 1	356.1	17.7	18.8	996	1870	183	652	110	5	71	10	50	7	20	2	16	2	205	<100	1330	957	3916	383	4299
REO Ro Conc 2	112.9	5.6	0.91	1350	2540	246	875	138	6	82	10	44	7	17	2	15	<2	188	<100	1180	918	5255	367	5622
REO Ro Conc 3	58.7	2.9	0.53	698	1310	124	452	75	4	47	7	31	5	12	<2	11	<2	133	<100	957	579	2763	250	3013
REO Ro Conc 4	167.3	8.3	0.18	643	1200	114	412	65	3	42	5	24	4	9	<2	8	<2	100	<100	687	480	2537	196	2733
REO Ro Conc 5	89.6	4.4	0.11	157	299	29	105	21	<2	14	2	10	<2	5	<2	4	<2	47	<100	356	151	713	88	801
REO Ro Tls	1230.6	61.1	0.03	35	64	6	26	3	<2	<2	<2	3	<2	<2	<2	<2	<2	<40	<100	78	37	236	57	293
Head (Calc.)	2015.2	100.0	3.43	353.7	662.9	64.1	232.1	37.5	2.9	23.8	4.3	16.5	3.4	7.0	2.0	6.1	2.0	85.4	<100	449.5	306.6	1453.3	150.5	1603.8
Head (Direct)			3.32	352.0	681.0	73.5	235.0	41.9	2.7	28.5	4.2	19.2	3.4	8.2	1.3	5.9	1.0	65.0	<100	520.0	350.0	1486.1	136.7	1622.8

Products	Weight	% Distribution																					
	%	S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
REO Ro Conc 1	17.7	96.9	49.8	49.8	50.4	49.6	51.8	30.5	52.6	41.5	53.6	36.2	50.3	17.7	46.7	17.7	42.4	17.7	52.3	55.2	47.6	45.0	47.4
REO Ro Conc 2	5.6	1.5	21.4	21.5	21.5	21.1	20.6	11.6	19.3	13.2	15.0	11.5	13.6	5.6	13.9	5.6	12.3	5.6	14.7	16.8	20.3	13.7	19.6
REO Ro Conc 3	2.9	0.5	5.7	5.8	5.6	5.7	5.8	4.0	5.7	4.8	5.5	4.3	5.0	2.9	5.3	2.9	4.5	2.9	6.2	5.5	5.5	4.8	5.5
REO Ro Conc 4	8.3	0.4	15.1	15.0	14.8	14.7	14.4	8.6	14.6	9.8	12.1	9.7	10.6	8.3	11.0	8.3	9.7	8.3	12.7	13.0	14.5	10.8	14.1
REO Ro Conc 5	4.4	0.1	2.0	2.0	2.0	2.0	2.5	3.1	2.6	2.1	2.7	2.6	3.2	4.4	2.9	4.4	2.4	4.4	3.5	2.2	2.2	2.6	2.2
REO Ro Tls	61.1	0.5	6.0	5.9	5.7	6.8	4.9	42.2	5.1	28.7	11.1	35.7	17.4	61.1	20.2	61.1	28.6	61.1	10.6	7.4	9.9	23.1	11.2
Head (Calc.)	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0

Cumulative Products

Products	Weight	Assays: U, Th, REE g/t, S %																					
	%	S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
Ro Conc 1	17.7	18.8	996.0	1870.0	183.0	652.0	110.0	5.0	71.0	10.0	50.0	7.0	20.0	2.0	16.0	2.0	205.0	100.0	1330.0	957.0	3916	383	4299
Ro Conc 1-2	23.3	14.5	1081.2	2031.3	198.2	705.7	116.7	5.2	73.6	10.0	48.6	7.0	19.3	2.0	15.8	2.0	200.9	100.0	1293.9	947.6	4238.3	379.1	4617.5
Ro Conc 1-3	26.2	12.9	1038.6	1951.1	189.9	677.5	112.1	5.1	70.7	9.7	46.6	6.8	18.5	2.0	15.2	2.0	193.4	100.0	1256.4	906.6	4074.2	364.8	4439.0
Ro Conc 1-4	34.5	9.87	943.4	1770.3	171.6	613.6	100.8	4.6	63.8	8.5	41.2	6.1	16.2	2.0	13.5	2.0	170.9	100.0	1119.3	803.9	3704.2	324.2	4028.3
Ro Conc 1-5	38.9	8.75	853.6	1602.2	155.4	555.5	91.7	4.3	58.1	7.8	37.6	5.6	14.9	2.0	12.4	2.0	156.7	100.0	1032.2	729.4	3362.6	297.2	3659.8
REO Ro Tls	61.1	0.03	35.0	64.0	6.0	26.0	3.00	2.00	2.00	2.00	3.00	2.00	2.00	2.00	2.00	2.00	40.0	100.0	78.0	37.0	236.0	57.0	293.0

Products	% Distribution																					
	S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
Ro Conc 1	96.9	49.8	49.8	50.4	49.6	51.8	30.5	52.6	41.5	53.6	36.2	50.3	17.7	46.7	17.7	42.4	17.7	52.3	55.2	47.6	45.0	47.4
Ro Conc 1-2	98.4	71.1	71.3	71.9	70.7	72.4	42.1	71.9	54.7	68.6	47.7	63.8	23.3	60.6	23.3	54.7	23.3	67.0	71.9	67.9	58.6	67.0
Ro Conc 1-3	98.9	76.9	77.1	77.5	76.4	78.2	46.1	77.6	59.5	74.1	51.9	68.8	26.2	65.9	26.2	59.3	26.2	73.2	77.4	73.4	63.5	72.5
Ro Conc 1-4	99.3	92.0	92.1	92.3	91.1	92.6	54.7	92.3	69.2	86.2	61.7	79.5	34.5	76.9	34.5	69.0	34.5	85.9	90.4	87.9	74.3	86.6
Ro Conc 1-5	99.5	94.0	94.1	94.3	93.2	95.1	57.8	94.9	71.3	88.9	64.3	82.6	38.9	79.8	38.9	71.4	38.9	89.4	92.6	90.1	76.9	88.8
REO Ro Tls	0.5	6.0	5.9	5.7	6.8	4.9	42.2	5.1	28.7	11.1	35.7	17.4	61.1	20.2	61.1	28.6	61.1	10.6	7.4	9.9	23.1	11.2

Test No. : F13 **Project No:** 13849-002 **Operator**: Dan Lang **Date**: Feb 04/2013

Purpose: To examine the low pulp density effects with duplicate F11 rougher conditions

Procedure: As outlined below. Condition at a 4.5L cell but transfer to a 10L cell for Rougher

Feed: 2000 g charge of Master Composite -6 mesh

Grind: 46 minutes at 65% solids in the laboratory Rod Mill GG RM#1

Ro Flotation % Solids = ~18

Notes: Flot time and reagent dosage may change based on visual.

Ro Fd K_{80} = 135 µm

LREE=La,Ce,Pr,Nd,Sm,Eu,Sc

LR-19: 62% FA2, 27% Aero 855, 9% Kerosene, 2% MIBC

HREE=Gd,Tb,Dy,Ho,Er,Tm,Yb,Lu,Y

TREE=LREE+HREE

Conditions:

Stage	Reagents Added, g/t										Time, minutes			Pulp		
	LR19	METSO	Oxalic								Grind	Cond.	Froth	pH	Temp, ºC	
Grind											46					
Condition 1		500	500									1+3			18	High density conditioning at 4.5L cell
Rougher 1	300											5	4	7.7-7.5	22	Flotation at 10L cell
Rougher 2	150		100									5	3	7.1		Froth lasts 3 min
Rougher 3	150		100									5	4	6.8		
Rougher 4	300		100									5	5	6.6		
Rougher 5	300		100									5	6	6.3		Barren Tail
Rougher Sub Total	**1200**	**500**	**900**									**29**	**22**			

Stage	Rougher
Flotation Cell	10L (2 kg)
Speed rpm	1800

Metallurgical Balance

Products	Weight		Assays: U, Th, REE g/t, S %																					
	g	%	S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
REO Ro Conc 1	197.8	9.9	18.6	3290	6130	590	2050	320	13	188	20	93	15	35	4	28	4	388	<100	1100	2200	12493	775	13268
REO Ro Conc 2	79.3	4.0	0.85	1130	2050	202	698	119	5	76	10	51	8	21	3	18	3	226	<100	937	1150	4304	416	4720
REO Ro Conc 3	62.8	3.1	0.35	352	662	66	239	43	2	35	6	28	5	12	<2	11	<2	116	<100	655	569	1464	217	1681
REO Ro Conc 4	87.2	4.4	0.16	188	363	37	132	28	<2	25	4	24	4	10	<2	10	<2	98	<100	616	427	850	179	1029
REO Ro Conc 5	58.1	2.9	0.09	96	185	18	67	16	<2	12	2	12	2	6	<2	6	<2	55	<100	370	178	484	99	583
REO Ro Tls	1516.6	75.8	0.04	<20	24	4	<20	3	2	<2	3	4	3	3	3	3	3	<40	<100	29	13	173	64	237
Head (Calc.)	2001.8	100.0	1.92	407.0	747.1	73.5	260.6	41.6	3.2	25.6	5.1	16.5	4.5	7.5	3.0	6.7	3.0	87.1	<100	225.9	314.4	1633.0	159.0	1792.0
Head (Direct)			1.92	407.0	767.0	81.0	254.0	42.6	1.90	26.10	3.3	16.50	2.6	6.70	0.80	5.20	0.70	62.00	<100	270.0	354.0	1578.5	123.9	1702

Products	Weight	% Distribution																					
	%	S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
REO Ro Conc 1	9.9	95.6	79.9	81.1	79.3	77.7	75.9	40.1	72.5	39.0	55.7	33.2	45.8	13.2	41.2	13.2	44.0	9.9	48.1	69.1	75.6	48.2	73.2
REO Ro Conc 2	4.0	1.8	11.0	10.9	10.9	10.6	11.3	6.2	11.7	7.8	12.2	7.1	11.0	4.0	10.6	4.0	10.3	4.0	16.4	14.5	10.4	10.4	10.4
REO Ro Conc 3	3.1	0.6	2.7	2.8	2.8	2.9	3.2	2.0	4.3	3.7	5.3	3.5	5.0	2.1	5.1	2.1	4.2	3.1	9.1	5.7	2.8	4.3	2.9
REO Ro Conc 4	4.4	0.4	2.0	2.1	2.2	2.2	2.9	2.7	4.2	3.4	6.3	3.9	5.8	2.9	6.5	2.9	4.9	4.4	11.9	5.9	2.3	4.9	2.5
REO Ro Conc 5	2.9	0.1	0.7	0.7	0.7	0.7	1.1	1.8	1.4	1.1	2.1	1.3	2.3	1.9	2.6	1.9	1.8	2.9	4.8	1.6	0.9	1.8	0.9
REO Ro Tls	75.8	1.6	3.7	2.4	4.1	5.8	5.5	47.3	5.9	44.9	18.4	50.9	30.1	75.9	33.9	75.9	34.8	75.8	9.7	3.1	8.0	30.5	10.0
Head (Calc.)	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0

Cumulative Products

Products	Weight	Assays: U, Th, REE g/t, S %																					
	%	S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
Ro Conc 1	9.9	18.6	3290.0	6130.0	590.0	2050.0	320.0	13.0	188.0	20.0	93.0	15.0	35.0	4.0	28.0	4.0	388.0	100.0	1100.0	2200.0	12493	775	13268
Ro Conc 1-2	13.8	13.5	2671.9	4962.4	479.0	1663.1	262.5	10.7	155.9	17.1	81.0	13.0	31.0	3.7	25.1	3.7	341.6	100.0	1053.4	1899.5	10149.5	672.3	10821.7
Ro Conc 1-3	17.0	11.1	2243.2	4167.9	402.7	1400.0	221.9	9.1	133.6	15.1	71.2	11.5	27.5	3.4	22.5	3.4	299.9	100.0	979.8	1653.7	8544.8	588.1	9132.9
Ro Conc 1-4	21.3	8.86	1823.6	3391.0	328.0	1141.1	182.3	7.7	111.4	12.8	61.6	10.0	23.9	3.1	20.0	3.1	258.7	100.0	905.5	1403.2	6973.7	504.6	7478.3
Ro Conc 1-5	24.2	7.81	1616.8	3007.1	290.9	1012.5	162.4	7.0	99.5	11.5	55.6	9.0	21.8	3.0	18.3	3.0	234.3	100.0	841.4	1256.5	6196.6	456.0	6652.7
REO Ro Tls	75.8	0.04	20.0	24.0	4.0	20.0	3.00	2.00	2.00	3.00	4.00	3.00	3.00	3.00	3.00	3.00	40.0	100.0	29.0	13.0	173.0	64.0	237.0

Products	% Distribution																					
	S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
Ro Conc 1	95.6	79.9	81.1	79.3	77.7	75.9	40.1	72.5	39.0	55.7	33.2	45.8	13.2	41.2	13.2	44.0	9.9	48.1	69.1	75.6	48.2	73.2
Ro Conc 1-2	97.4	90.9	92.0	90.2	88.4	87.3	46.2	84.2	46.8	67.9	40.3	56.8	17.2	51.9	17.2	54.3	13.8	64.5	83.6	86.0	58.5	83.6
Ro Conc 1-3	97.9	93.6	94.7	93.0	91.2	90.5	48.2	88.5	50.5	73.2	43.8	61.8	19.3	57.0	19.3	58.5	17.0	73.6	89.3	88.8	62.8	86.5
Ro Conc 1-4	98.3	95.6	96.8	95.2	93.4	93.4	50.9	92.7	54.0	79.5	47.8	67.6	22.2	63.5	22.2	63.4	21.3	85.5	95.2	91.1	67.7	89.0
Ro Conc 1-5	98.4	96.3	97.6	95.9	94.2	94.5	52.7	94.1	55.1	81.6	49.1	69.9	24.1	66.1	24.1	65.2	24.2	90.3	96.9	92.0	69.5	90.0
REO Ro Tls	1.6	3.7	2.4	4.1	5.8	5.5	47.3	5.9	44.9	18.4	50.9	30.1	75.9	33.9	75.9	34.8	75.8	9.7	3.1	8.0	30.5	10.0

Test No. : F14 **Project No:** 13849-002 **Operator**: Dan Lang **Date**: Feb 06/2013

Purpose: To examine the effects of oxalic acid and low pulp density in rougher and the cleaner performance

Procedure: As outlined below. Conditioning at 4.5L cell but transfering to 10L for rougher flotation.
Blending cleaner1 scavenger concentrate with cleaner 1 concentrate as the cleaner2 feed.

Feed: 2000 g charge of LR Composite -6 mesh

Grind: 46 minutes at 65% solids in the laboratory Rod Mill GG RM#3 Ro Flotation % Solids = ~18

Notes: Flot time and reagent dosage may change based on visual. Ro Fd K_{80} = 135 µm
LREE=La,Ce,Pr,Nd,Sm,Eu,Sc LR-19: 62% FA2, 27% Aero 855, 9% Kerosene, 2% MIBC
HREE=Gd,Tb,Dy,Ho,Er,Tm,Yb,Lu,Y
TREE=LREE+HREE

Conditions:

Stage	**Reagents Added, g/t**										**Time, minutes**			**Pulp**		
	LR19	**METSO**	**Oxalic**								**Grind**	**Cond.**	**Froth**	**pH**	**Temp, ºC**	
Grind											46					Rougher old behaviour, too frothy
Condition 1		500	500									2				sticky paddle, possible contaminated
Rougher 1	300											5	4	7.4	20	
Rougher 2	150		100									5	5	7.4		Frother only lost ~3mins
Rougher 3	150		100									5	4	6.8		
Rougher 4	300		100									5	10	6.6		
Rougher 5	300		100									5	6	6.5	27	Floculated tailings
Rougher Sub Total	**1200**	**500**	**900**									**27**	**29**			
REO Clnr 1		250	100									2	8	7.9		
	60											5	3	7.6		
REO Clnr Scav	75											3	1.5	7.5		Froth dies quickly
REO Clnr 2		150	125									2	7	7.2		
REO Clnr 3		100	100									2	6	7.5		
Cleaner Sub Total	**135**	**500**	**325**									**14**	**25.5**			

Stage	Rougher	Cleaner	Cleaner Scavenger
Flotation Cell	10L (2 kg)	4.5	4.5
Speed rpm	1800		

Metallurgical Balance

Products	**Weight**		**Assays: U, Th, REE g/t, S %**																					
	g	**%**	**S**	**La**	**Ce**	**Pr**	**Nd**	**Sm**	**Eu**	**Gd**	**Tb**	**Dy**	**Ho**	**Er**	**Tm**	**Yb**	**Lu**	**Y**	**Sc**	**U**	**Th**	**LREE**	**HREE**	**TREE**
REO 3rd Cl Conc	336.6	16.7	19.4	1780	3320	329	1160	187	9	107	13	61	10	24	3	18	3	267	<100	1280	1350	6885	506	7391
REO 3rd Cl Tail	82.5	4.1	0.77	302	559	56	205	40	3	28	4	23	4	9	<2	8	<2	104	<100	701	407	1265	184	1449
REO 2nd Cl Tail	201.4	10.0	0.32	155	299	32	113	23	<2	15	3	14	3	6	<2	5	<2	68	<100	453	238	724	118	842
REO 1st Cl Sc Tail	555.2	27.6	0.07	56	103	11	36	8	<2	7	<2	6	<2	3	<2	3	<2	<40	<100	229	103	316	67	383
REO Ro Tail	834.3	41.5	0.05	34	61	7	25	4	<2	<2	<2	5	<2	2	<2	<2	<2	<40	<100	156	50	233	59	292
Head (Calc.)	2010.0	100.0	3.35	355.6	662.6	66.5	234.3	39.1	3.2	23.3	4.0	16.3	3.5	6.6	2.2	5.5	2.2	83.4	<100	416.5	315.8	1461.4	147.1	1608.6
Head (Direct)			3.32	352.0	681.0	73.5	235.0	41.9	2.7	28.5	4.2	19.2	3.4	8.2	1.3	5.9	1.0	65.0	25.0	520.0	350.0	1411.1	136.7	1547.8

Products	**Weight**	**% Distribution**																					
	%	**S**	**La**	**Ce**	**Pr**	**Nd**	**Sm**	**Eu**	**Gd**	**Tb**	**Dy**	**Ho**	**Er**	**Tm**	**Yb**	**Lu**	**Y**	**Sc**	**U**	**Th**	**LREE**	**HREE**	**TREE**
REO 3rd Cl Conc	16.7	96.9	83.8	83.9	82.8	82.9	80.0	46.9	76.8	54.1	62.7	47.5	60.5	23.2	54.8	23.2	53.6	16.7	51.5	71.6	78.9	57.6	76.9
REO 3rd Cl Tail	4.1	0.9	3.5	3.5	3.5	3.6	4.2	3.8	4.9	4.1	5.8	4.7	5.6	3.8	6.0	3.8	5.1	4.1	6.9	5.3	3.6	5.1	3.7
REO 2nd Cl Tail	10.0	1.0	4.4	4.5	4.8	4.8	5.9	6.2	6.4	7.5	8.6	8.5	9.0	9.2	9.1	9.2	8.2	10.0	10.9	7.6	5.0	8.0	5.2
REO 1st Cl Sc Tail	27.6	0.6	4.4	4.3	4.6	4.2	5.6	17.2	8.3	13.7	10.2	15.7	12.5	25.5	15.1	25.5	13.2	27.6	15.2	9.0	6.0	12.6	6.6
REO Ro Tail	41.5	0.6	4.0	3.8	4.4	4.4	4.2	25.8	3.6	20.6	12.7	23.6	12.5	38.3	15.1	38.3	19.9	41.5	15.5	6.6	6.6	16.6	7.5
Head (Calc.)	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0

Combined Products

Products	**Weight**	**Assays: U, Th, REE g/t, S %**																					
	%	**S**	**La**	**Ce**	**Pr**	**Nd**	**Sm**	**Eu**	**Gd**	**Tb**	**Dy**	**Ho**	**Er**	**Tm**	**Yb**	**Lu**	**Y**	**Sc**	**U**	**Th**	**LREE**	**HREE**	**TREE**
REO 3rd Cl Conc	16.7	19.4	1780.0	3320.0	329.0	1160.0	187.0	9.0	107.0	13.0	61.0	10.0	24.0	3.0	18.0	3.0	267.0	100.0	1280.0	1350.0	6885	506	7391
REO 2nd Cl Conc	20.9	15.7	1489.1	2776.5	275.3	972.0	158.1	7.8	91.4	11.2	53.5	8.8	21.0	2.8	16.0	2.8	234.9	100.0	1166.0	1164.4	5778.7	442.6	6221.3
REO 1st Cl Conc +1st Cl Sc Conc	30.9	10.7	1056.1	1972.4	196.3	693.2	114.2	5.9	66.6	8.6	40.7	6.9	16.2	2.5	12.5	2.5	180.7	100.0	934.6	863.7	4138.1	337.3	4475.3
REO 1st Cl Sc Tail	27.6	0.07	56.0	103.0	11.0	36.0	8.0	2.0	7.0	2.0	6.0	2.0	3.0	2.0	3.0	2.0	40.0	100.0	229.0	103.0	316.0	67.0	383.0
REO Ro Conc	58.5	5.70	583.8	1089.6	108.8	382.8	64.1	4.1	38.5	5.5	24.3	4.6	9.9	2.3	8.0	2.3	114.3	100.0	601.4	504.5	2333.2	209.6	2542.8
REO Ro Tail	41.5	0.05	34.0	61.0	7.0	25.0	4.00	2.00	2.00	2.00	5.00	2.00	2.00	2.00	2.00	2.00	40.0	100.0	156.0	50.0	233.0	59.0	292.0

Products	**% Distribution**																					
	S	**La**	**Ce**	**Pr**	**Nd**	**Sm**	**Eu**	**Gd**	**Tb**	**Dy**	**Ho**	**Er**	**Tm**	**Yb**	**Lu**	**Y**	**Sc**	**U**	**Th**	**LREE**	**HREE**	**TREE**
REO 3rd Cl Conc	96.9	83.8	83.9	82.8	82.9	80.0	46.9	76.8	54.1	62.7	47.5	60.5	23.2	54.8	23.2	53.6	16.7	51.5	71.6	78.9	57.6	76.9
REO 2nd Cl Conc	97.8	87.3	87.4	86.2	86.5	84.2	50.7	81.7	58.2	68.5	52.2	66.0	27.0	60.7	27.0	58.7	20.9	58.4	76.9	82.4	62.7	80.6
REO 1st Cl Conc +1st Cl Sc Conc	98.8	91.7	91.9	91.1	91.3	90.1	57.0	88.2	65.6	77.1	60.7	75.0	36.2	69.9	36.2	66.9	30.9	69.3	84.4	87.4	70.8	85.9
REO 1st Cl Sc Tail	0.6	4.4	4.3	4.6	4.2	5.6	17.2	8.3	13.7	10.2	15.7	12.5	25.5	15.1	25.5	13.2	27.6	15.2	9.0	6.0	12.6	6.6
REO Ro Conc	99.4	96.0	96.2	95.6	95.6	95.8	74.2	96.4	79.4	87.3	76.4	87.5	61.7	84.9	61.7	80.1	58.5	84.5	93.4	93.4	83.4	92.5
REO Ro Tail	0.6	4.0	3.8	4.4	4.4	4.2	25.8	3.6	20.6	12.7	23.6	12.5	38.3	15.1	38.3	19.9	41.5	15.5	6.6	6.6	16.6	7.5

Test No. : F15 **Project No:** 13849-002 **Operator:** Dan Lang **Date:** Feb 06/2013

Purpose: To examine the effects of oxalic acid and low pulp density in rougher and the cleaner performance (Repeat F14 with low dosage of collector and short flotation time in rougher)

Procedure: As outlined below. Conditioning at 4.5L cell but transfering to 10L for rougher flotation.

Feed: 2000 g charge of Master Composite -6 mesh

Grind: 46 minutes at 65% solids in the laboratory Rod Mill GG RM#3

Ro Flotation % Solids = ~18

Notes: Flot time and reagent dosage may change based on visual.

Ro Fd K_{80} = 135 µm

LREE=La,Ce,Pr,Nd,Sm,Eu,Sc

LR-19: 62% FA2, 27% Aero 855, 9% Kerosene, 2% MIBC

HREE=Gd,Tb,Dy,Ho,Er,Tm,Yb,Lu,Y

TREE=LREE+HREE

Conditions:

Stage	Reagents Added, g/t										Time, minutes			Pulp	
	LR19	METSO	Oxalic								Grind	Cond.	Froth	pH	Temp, ºC
Grind											46				
Condition 1		500	500									2		7.5	
Rougher 1	300											5	4	7.8	
Rougher 2	150		100									5	5	7.2	
Rougher 3	150		100									5	5		
Rougher 4	300		100									5	5	7	
Rougher Sub Total	**900**	**500**	**800**									**22**	**19**		
REO Clnr 1	90	250	100									2	8	8.2	
												5	1.5		
REO Clnr Scav	90											5	3	7.7	
REO Clnr 2		125	100										5	8.0	
	90											5		7.6	
Cleaner Sub Total	**270**	**375**	**200**									**17**	**17.5**		

Nice stuff

Stage	Rougher	Cleaner	Cleaner Scavenger
Flotation Cell	10L (2 kg)	4.5L	4.5L
Speed rpm	1800		

Metallurgical Balance

Products	Weight		Assays: U, Th, REE g/t, S %																						
	g	%	S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
REO 2nd Cl Conc	357.6	17.8	10.1	2170	3760	392	1210	198	9	111	16	73	12	29	<5	21	<5	290	<100	1050	1520	7839	562	8401
REO 2nd Cl Tail	90.0	4.5	0.12	100	184	19	62	12	<5	8	<5	8	<5	<5	<5	<5	<5	<40	<100	189	133	482	86	568
REO 1st Cl Sc Conc	83.6	4.2	0.15	150	282	30	94	18	<5	12	<5	12	<5	6	<5	<5	<5	53	<100	317	200	679	108	787
REO 1st Cl Sc Tail	231.6	11.5	0.05	46	82	9	28	5	<5	4	<5	4	<5	<5	<5	<5	<5	<40	<100	121	66	275	78	353
REO Ro Tail	1248.1	62.1	0.04	24	45	4	<20	2	<5	2	<5	<2	<5	<5	<5	<5	<5	<40	<100	38	23	200	74	274
Head (Calc.)	2010.9	100.0	1.84	416.8	726.0	75.3	237.5	38.3	5.7	22.3	7.0	15.5	6.2	9.3	5.0	7.8	5.0	85.0	<100	245.9	306.4	1599.6	163.2	1762.8
Head (Direct)			1.92	407.0	767.0	81.0	254.0	42.6	1.90	26.1	3.3	16.5	2.6	6.7	0.80	5.20	0.70	62.00	<100	270.0	354.0	1578.5	123.9	1702.4

Products	Weight	% Distribution																						
	%	S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
REO 2nd Cl Conc	17.8	97.7	92.6	92.1	92.5	90.6	91.9	28.0	88.5	40.9	83.5	34.2	55.4	17.8	47.6	17.8	60.7	17.8	75.9	88.2	87.1	61.2	84.7
REO 2nd Cl Tail	4.5	0.3	1.1	1.1	1.1	1.2	1.4	3.9	1.6	3.2	2.3	3.6	2.4	4.5	2.9	4.5	2.1	4.5	3.4	1.9	1.3	2.4	1.4
REO 1st Cl Sc Conc	4.2	0.3	1.5	1.6	1.7	1.6	2.0	3.6	2.2	3.0	3.2	3.3	2.7	4.2	2.6	4.2	2.6	4.2	5.4	2.7	1.8	2.8	1.9
REO 1st Cl Sc Tail	11.5	0.3	1.3	1.3	1.4	1.4	1.5	10.1	2.1	8.3	3.0	9.2	6.2	11.5	7.3	11.5	5.4	11.5	5.7	2.5	2.0	5.5	2.3
REO Ro Tail	62.1	1.4	3.6	3.8	3.3	5.2	3.2	54.3	5.6	44.6	8.0	49.7	33.3	62.1	39.6	62.1	29.2	62.1	9.6	4.7	7.8	28.1	9.6
Head (Calc.)	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0

Combined Products

Products	Weight	Assays: U, Th, REE g/t, S %																						
	%	S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
REO 2nd Cl Conc	17.8	10.1	2170.0	3760.0	392.0	1210.0	198.0	9.0	111.0	16.0	73.0	12.0	29.0	5.0	21.0	5.0	290.0	100.0	1050.0	1520.0	7839	562	8401
REO 1st Cl Conc	22.3	8.1	1753.8	3041.0	317.0	979.2	160.6	8.2	90.3	13.8	59.9	10.6	24.2	5.0	17.8	5.0	239.7	100.0	876.9	1241.1	6359.7	466.3	6826.0
REO 1st Cl Conc +1st Cl Sc Conc	26.4	6.8	1501.4	2606.8	271.8	839.9	138.2	7.7	78.0	12.4	52.4	9.7	21.3	5.0	15.8	5.0	210.3	100.0	788.8	1077.3	5465.7	409.9	5875.6
REO Ro Conc	37.9	4.8	1059.5	1840.2	192.0	593.4	97.7	6.9	55.5	10.2	37.7	8.3	16.4	5.0	12.5	5.0	158.6	100.0	586.0	770.2	3889.7	309.1	4198.8
REO Ro Tail	62.1	0.04	24.0	45.0	4.0	20.0	2.00	5.00	2.00	5.00	2.00	5.00	5.00	5.00	5.00	5.00	40.0	100.0	38.0	23.0	200.0	74.0	274.0

Products	% Distribution																					
	S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
REO 2nd Cl Conc	97.7	92.6	92.1	92.5	90.6	91.9	28.0	88.5	40.9	83.5	34.2	55.4	17.8	47.6	17.8	60.7	17.8	75.9	88.2	87.1	61.2	84.7
REO 1st Cl Conc	98.0	93.7	93.2	93.7	91.8	93.3	31.9	90.1	44.1	85.8	37.8	57.8	22.3	50.5	22.3	62.8	22.3	79.4	90.1	88.5	63.6	86.2
REO 1st Cl Conc +1st Cl Sc Conc	98.3	95.2	94.9	95.3	93.4	95.3	35.6	92.4	47.1	89.0	41.1	60.5	26.4	53.1	26.4	65.4	26.4	84.7	92.9	90.3	66.4	88.0
REO Ro Conc	98.6	96.4	96.2	96.7	94.8	96.8	45.7	94.4	55.4	92.0	50.3	66.7	37.9	60.4	37.9	70.8	37.9	90.4	95.3	92.2	71.9	90.4
REO Ro Tail	1.4	3.6	3.8	3.3	5.2	3.2	54.3	5.6	44.6	8.0	49.7	33.3	62.1	39.6	62.1	29.2	62.1	9.6	4.7	7.8	28.1	9.6

Test No. : F16 (UR Comp) **Project No:** 13849-002 **Operator**: Dan Lang **Date**: 19-Feb-13

Purpose: To produce concentrate for hydrometallurgical tests with the large cell under similar conditions of F9

Procedure: As outlined below.

Feed: 10,000 g charge of UR Composite

Grind: minutes at 65% solids in the laboratory 10kg Rod Mill Ro Flotation % Solids = ~35

Notes: Flot time and reagent dosage may change based on visual. Ro Fd K_{80} = 117 µm (Malvern)
LREE=La,Ce,Pr,Nd,Sm,Eu,Sc
HREE=Gd,Tb,Dy,Ho,Er,Tm,Yb,Lu,Y
TREE=LREE+HREE

Conditions:

Stage	Reagents Added, g/t										Time, minutes			Pulp	
	LR19	Metso									Grind	Cond.	Froth	pH	Temp, ºC
Grind											75				
REO Cond 1															
REO Cond 2		500										3			18
REO Rougher 1	300											5	5	9.4	20
REO Rougher 2	150											5	5		22
REO Rougher 3	150											5	5		
REO Rougher 4	200											5	6		
REO Rougher 5	200											5	6		
REO Rougher 6	200														
REO Rougher Subtol	**1200**	**500**										**28**	**27**		

Stage	Rougher
Flotation Cell	(~25L 10 kg)
Speed rpm	1800

Metallurgical Balance

Products	Weight		Assays: U, Th, REE g/t, S %																					
	g	%	S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
REO Ro Conc 1-4	1920.2	18.5	7.88	2700	5080	502	1680	263	10	150	16	73	13	27	3	20	3	301	<100	620	1680	10335	606	10941
REO Ro Conc 5	639.3	6.2	0.28	207	396	42	157	37	<2	34	6	35	6	15	2	13	<2	157	<100	556	868	941	270	1211
REO Ro Conc 6	890.3	8.6	0.06	53	100	11	34	8	<2	8	<2	9	<2	4	<2	4	<2	50	<100	174	156	308	83	391
REO Ro Tail	6924.2	66.7	0.03	26	48	5	<20	4	<2	2	<2	3	<2	<2	<2	<2	<2	<40	<100	60	51	205	57	262
Head (Calc.)	10374.0	100.0	1.50	534.4	1005.3	99.8	336.9	54.3	3.5	31.9	4.8	18.4	4.3	7.6	2.2	6.2	2.2	96.4	<100	204.0	411.9	2134.3	174.0	2308.2
Head (Direct)			1.46	536	1000	105	328	54.3	2.9	34.0	4.7	20.3	3.7	8.8	2.0	7.1	2.3	76.0	<100	218.0	445.0	2126.2	158.9	2285.1

Products	Weight	% Distribution																					
	%	S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
REO Ro Conc 1-4	18.5	97.2	93.5	93.5	93.1	92.3	89.6	53.2	87.1	61.2	73.3	56.2	65.8	25.4	59.9	25.4	57.8	18.5	56.3	75.5	89.6	64.5	87.7
REO Ro Conc 5	6.2	1.1	2.4	2.4	2.6	2.9	4.2	3.5	6.6	7.6	11.7	8.6	12.2	5.6	13.0	5.6	10.0	6.2	16.8	13.0	2.7	9.6	3.2
REO Ro Conc 6	8.6	0.3	0.9	0.9	0.9	0.9	1.3	4.9	2.2	3.5	4.2	4.0	4.5	7.9	5.6	7.9	4.5	8.6	7.3	3.3	1.2	4.1	1.5
REO Ro Tail	66.7	1.3	3.2	3.2	3.3	4.0	4.9	38.4	4.2	27.6	10.9	31.2	17.6	61.1	21.6	61.1	27.7	66.7	19.6	8.3	6.4	21.9	7.6
Head (Calc.)	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0

Cumulative Products

Products	Weight	Assays: U, Th, REE g/t, S %																					
	%	S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
REO Ro Conc 1-4	18.5	7.88	2700	5080	502	1680	263	10	150	16	73	13	27	3	20	3	301	100	620	1680	10335	606	10941
REO Ro Conc 1-5	24.7	5.98	2077	3910	387	1300	207	8	121	14	64	11	24	3	18	3	265	100	604	1477	7989	522	8511
REO Ro Conc 1-6	33.3	4.45	1555	2927	290	973	155	6	92	11	49	9	19	3	15	3	210	100	493	1136	6006	409	6415
REO Ro Tail	66.7	0.03	26	48	5	20	4	2	2	2	3	2	2	2	2.0	2.0	40.0	100.0	60	51	205	57	262
Head (Calc.)	100.0	1.50	534.4	1005.3	99.8	336.9	54.3	3.5	31.9	4.8	18.4	4.3	7.6	2.2	6.2	2.2	96.4	100.0	204.0	411.9	2134.3	174.0	2308.2

Products	% Distribution																					
	S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
REO Ro Conc 1-4	97.2	93.5	93.5	93.1	92.3	89.6	53.2	87.1	61.2	73.3	56.2	65.8	25.4	59.9	25.4	57.8	18.5	56.3	75.5	89.6	64.5	87.7
REO Ro Conc 1-5	98.3	95.9	96.0	95.7	95.2	93.8	56.7	93.7	68.9	85.0	64.8	77.9	31.1	72.9	31.1	67.8	24.7	73.0	88.5	92.4	74.0	91.0
REO Ro Conc 1-6	98.7	96.8	96.8	96.7	96.0	95.1	61.6	95.8	72.4	89.1	68.8	82.4	38.9	78.4	38.9	72.3	33.3	80.4	91.7	93.6	78.1	92.4
REO Ro Tail	1.3	3.2	3.2	3.3	4.0	4.9	38.4	4.2	27.6	10.9	31.2	17.6	61.1	21.6	61.1	27.7	66.7	19.6	8.3	6.4	21.9	7.6

Result Analysis Report

Sample Name:
13849-002 F16 Comb Prod - Average

SOP Name:
Defaultar

Measured:
February 21, 2013 12:40:47 PM

Sample Source & type:

Measured by:
LR_Malvern1

Analysed:
February 21, 2013 12:40:49 PM

Sample bulk lot ref:
Les

Result Source:
Averaged

Particle Name:
Default

Accessory Name:
Hydro 2000G (A)

Analysis model:
General purpose

Sensitivity:
Enhanced

Particle RI:
1.520

Absorption:
0.1

Size range:
0.020 to 2000.000 um

Obscuration:
13.40 %

Dispersant Name:
Water

Dispersant RI:
1.330

Weighted Residual:
0.614 %

Result Emulation:
Off

Concentration:
0.0224 %Vol

Span :
2.965

Uniformity:
0.953

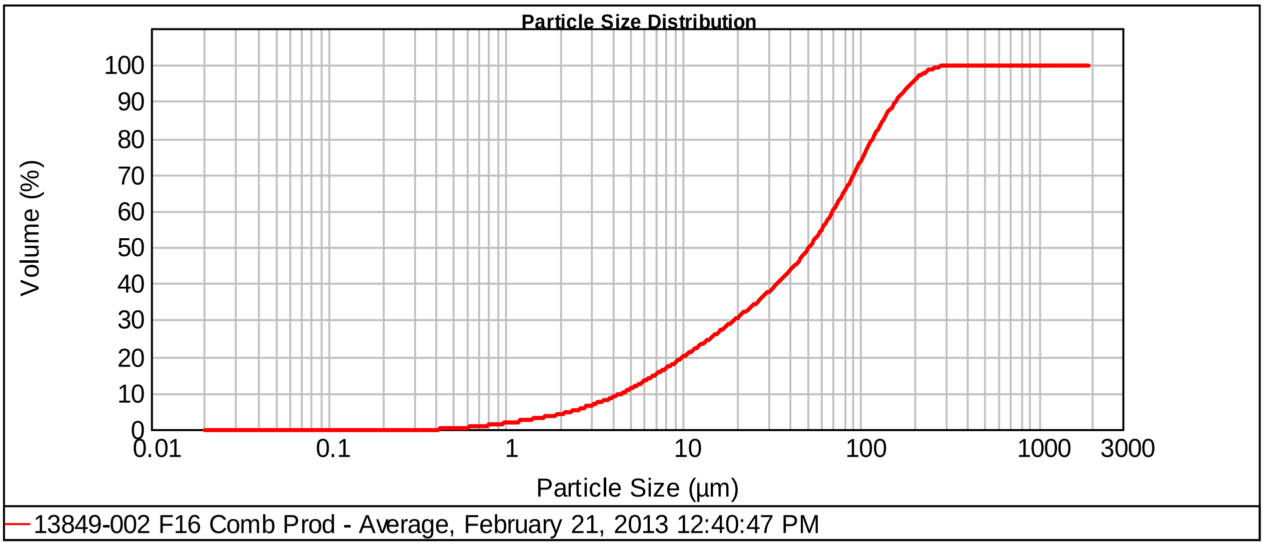
Result units:
Volume

Specific Surface Area:
0.569 m²/g

Surface Weighted Mean D[3,2]:
10.543 um

Vol. Weighted Mean D[4,3]:
67.125 um

d(0.1): 4.512 um **d(0.5): 51.117 um** **D(0.80) : 116.68 µm**



Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %
0.010	0.00	0.105	0.00	1.096	2.01	11.482	21.86	120.226	81.14	1258.925	100.00
0.011	0.00	0.120	0.00	1.259	2.41	13.183	23.92	138.038	86.11	1445.440	100.00
0.013	0.00	0.138	0.00	1.445	2.84	15.136	26.04	158.489	90.44	1659.587	100.00
0.015	0.00	0.158	0.00	1.660	3.33	17.378	28.22	181.970	93.98	1905.461	100.00
0.017	0.00	0.182	0.00	1.905	3.88	19.953	30.47	208.930	96.63	2187.762	100.00
0.020	0.00	0.209	0.00	2.188	4.53	22.909	32.80	239.883	98.44	2511.886	100.00
0.023	0.00	0.240	0.00	2.512	5.29	26.303	35.23	275.423	99.49	2884.032	100.00
0.026	0.00	0.275	0.00	2.884	6.18	30.200	37.80	316.228	99.98	3311.311	100.00
0.030	0.00	0.316	0.00	3.311	7.20	34.674	40.57	363.078	100.00	3801.894	100.00
0.035	0.00	0.363	0.00	3.802	8.37	39.811	43.61	416.869	100.00	4365.158	100.00
0.040	0.00	0.417	0.00	4.365	9.67	45.709	46.98	478.630	100.00	5011.872	100.00
0.046	0.00	0.479	0.11	5.012	11.10	52.481	50.75	549.541	100.00	5754.399	100.00
0.052	0.00	0.550	0.31	5.754	12.66	60.256	54.99	630.957	100.00	6606.934	100.00
0.060	0.00	0.631	0.58	6.607	14.32	69.183	59.70	724.436	100.00	7585.776	100.00
0.069	0.00	0.724	0.90	7.586	16.08	79.433	64.83	831.764	100.00	8709.636	100.00
0.079	0.00	0.832	1.25	8.710	17.93	91.201	70.25	954.993	100.00	10000.000	100.00
0.091	0.00	0.955	1.62	10.000	19.86	104.713	75.77	1096.478	100.00		

Operator notes:

Test No. : F17 (IQ Comp) **Project No:** 13849-002 **Operator**: Dan Lang **Date**: 19-Feb-13

Purpose: To produce concentrate for hydrometallurgical tests with the large cell under similar conditions of F9

Procedure: As outlined below.

Feed: 10,000 g charge of IQ Composite

Grind: minutes at 65% solids in the laboratory 10kg Rod Mill Ro Flotation % Solids = ~35

Notes: Flot time and reagent dosage may change based on visual. Ro Fd K_{80} = 122 µm (Malvern)

LREE=La,Ce,Pr,Nd,Sm,Eu,Sc
HREE=Gd,Tb,Dy,Ho,Er,Tm,Yb,Lu,Y
TREE=LREE+HREE

Conditions:

Stage	Reagents Added, g/t										Time, minutes			Pulp	
	LR19	Metso									Grind	Cond.	Froth	pH	Temp, °C
Grind											75				
REO Cond 1															
REO Cond 2		500										3		9.6	
REO Rougher 1	300											5	7		20
REO Rougher 2	150											5	7		
REO Rougher 3	150											5	8		
REO Rougher 4	150											5	8		
REO Rougher Subtol	**750**											**23**	**30**		

Stage	Rougher
Flotation Cell	(~25L 10 kg)
Speed rpm	1800

Metallurgical Balance

Products	Weight		Assays: U, Th, REE g/t, S %																				
	g	%	S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
REO Ro Conc 1-3	1145.6	11.0	9.36	2150	4040	400	1370	199	8	108	11	43	7	15	<2	10	<2	167	<100	343	1070	8267	365	8632
REO Ro Conc 4	327.1	3.1	0.56	208	401	41	135	38	2	34	6	31	5	12	<2	10	<2	131	<100	436	992	925	233	1158
REO Ro Tail	8958.9	85.9	0.04	<20	29	3	<20	3	<2	3	<2	4	<2	<2	<2	<2	<2	<40	<100	63	66	177	59	236
Head (Calc.)	10431.6	100.0	1.08	259.8	481.2	47.8	171.9	25.6	2.7	15.5	3.1	9.1	2.6	3.7	2.0	3.1	2.0	56.8	<100	105.4	205.3	1088.9	98.1	1187.0
Head (Direct)			1.04	259	488	51.7	162.0	26.2	1.5	15.7	1.8	8.6	1.4	3.3	0.4	2.5	< 0.5	31.0	<100	120.0	218.0	1088.4	64.7	1153.1

Products	Weight	% Distribution																				
	%	S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
REO Ro Conc 1-3	11.0	95.2	90.9	92.2	91.9	87.5	85.3	33.0	76.5	38.8	51.7	29.1	44.0	11.0	35.1	11.0	32.3	11.0	35.7	57.2	83.4	40.9	79.9
REO Ro Conc 4	3.1	1.6	2.5	2.6	2.7	2.5	4.7	2.4	6.9	6.0	10.6	5.9	10.1	3.1	10.0	3.1	7.2	3.1	13.0	15.2	2.7	7.5	3.1
REO Ro Tail	85.9	3.2	6.6	5.2	5.4	10.0	10.1	64.6	16.6	55.2	37.6	65.0	45.9	85.9	54.9	85.9	60.5	85.9	51.3	27.6	14.0	51.7	17.1
Head (Calc.)	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0

Cumulative Products

Products	Weight	Assays: U, Th, REE g/t, S %																				
	%	S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
REO Ro Conc 1-3	11.0	9.36	2150	4040	400	1370	199	8	108	11	43	7	15	2	10	2	167	100	343	1070	8267	365	8632
REO Ro Conc 1-4	14.1	7.41	1719	3232	320	1096	163	7	92	10	40	7	14	2	10	2	159	100	364	1053	6636	336	6972
REO Ro Tail	85.9	0.04	20	29	3	20	3	2	3	2	4	2	2	2	2.0	2.0	40.0	100.0	63	66	177	59	236
Head (Calc.)	100.0	1.08	259.8	481.2	47.8	171.9	25.6	2.7	15.5	3.1	9.1	2.6	3.7	2.0	3.1	2.0	56.8	100.0	105.4	205.3	1088.9	98.1	1187.0

Products	% Distribution																					
	S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
REO Ro Conc 1-3	95.2	90.9	92.2	91.9	87.5	85.3	33.0	76.5	38.8	51.7	29.1	44.0	11.0	35.1	11.0	32.3	11.0	35.7	57.2	83.4	40.9	#DIV/0!
REO Ro Conc 1-4	96.8	93.4	94.8	94.6	90.0	89.9	35.4	83.4	44.8	62.4	35.0	54.1	14.1	45.1	14.1	39.5	14.1	48.7	72.4	86.0	48.3	#DIV/0!
REO Ro Tail	3.2	6.6	5.2	5.4	10.0	10.1	64.6	16.6	55.2	37.6	65.0	45.9	85.9	54.9	85.9	60.5	85.9	51.3	27.6	14.0	51.7	17.1

Result Analysis Report

Sample Name:
13849-002 F17 Comb Prod - Average

SOP Name:
Defaultar

Measured:
February 21, 2013 12:54:55 PM

Sample Source & type:

Measured by:
LR_Malvern1

Analysed:
February 21, 2013 12:54:57 PM

Sample bulk lot ref:
Les

Result Source:
Averaged

Particle Name:
Default

Accessory Name:
Hydro 2000G (A)

Analysis model:
General purpose

Sensitivity:
Enhanced

Particle RI:
1.520

Absorption:
0.1

Size range:
0.020 to 2000.000 um

Obscuration:
13.51 %

Dispersant Name:
Water

Dispersant RI:
1.330

Weighted Residual:
0.627 %

Result Emulation:
Off

Concentration:
0.0223 %Vol

Span :
2.987

Uniformity:
0.971

Result units:
Volume

Specific Surface Area:
0.571 m²/g

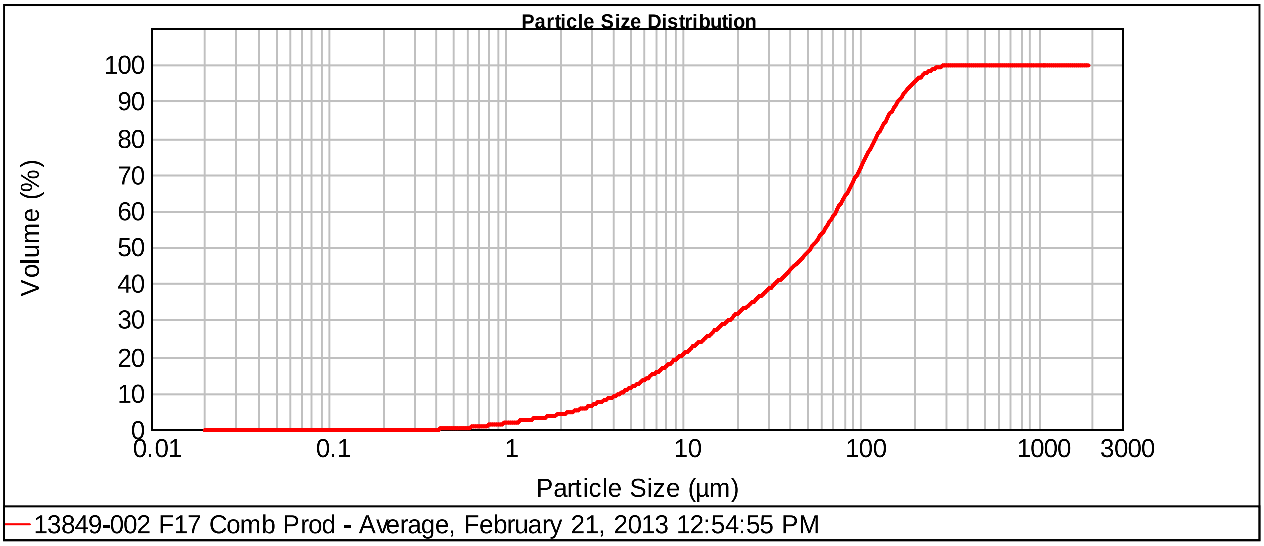
Surface Weighted Mean D[3,2]:
10.504 um

Vol. Weighted Mean D[4,3]:
69.227 um

d(0.1): 4.458 um

d(0.5): 52.733 um

D(0.80) : 121.79 µm



Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %
0.010	0.00	0.105	0.00	1.096	1.96	11.482	22.80	120.226	79.49	1258.925	100.00
0.011	0.00	0.120	0.00	1.259	2.37	13.183	24.97	138.038	84.72	1445.440	100.00
0.013	0.00	0.138	0.00	1.445	2.80	15.136	27.17	158.489	89.34	1659.587	100.00
0.015	0.00	0.158	0.00	1.660	3.28	17.378	29.38	181.970	93.18	1905.461	100.00
0.017	0.00	0.182	0.00	1.905	3.84	19.953	31.61	208.930	96.11	2187.762	100.00
0.020	0.00	0.209	0.00	2.188	4.49	22.909	33.85	239.883	98.13	2511.886	100.00
0.023	0.00	0.240	0.00	2.512	5.26	26.303	36.12	275.423	99.34	2884.032	100.00
0.026	0.00	0.275	0.00	2.884	6.16	30.200	38.45	316.228	99.93	3311.311	100.00
0.030	0.00	0.316	0.00	3.311	7.21	34.674	40.90	363.078	100.00	3801.894	100.00
0.035	0.00	0.363	0.00	3.802	8.42	39.811	43.56	416.869	100.00	4365.158	100.00
0.040	0.00	0.417	0.00	4.365	9.78	45.709	46.52	478.630	100.00	5011.872	100.00
0.046	0.00	0.479	0.10	5.012	11.29	52.481	49.88	549.541	100.00	5754.399	100.00
0.052	0.00	0.550	0.28	5.754	12.95	60.256	53.73	630.957	100.00	6606.934	100.00
0.060	0.00	0.631	0.55	6.607	14.73	69.183	58.13	724.436	100.00	7585.776	100.00
0.069	0.00	0.724	0.86	7.586	16.62	79.433	63.04	831.764	100.00	8709.636	100.00
0.079	0.00	0.832	1.21	8.710	18.61	91.201	68.38	954.993	100.00	10000.000	100.00
0.091	0.00	0.955	1.58	10.000	20.68	104.713	73.95	1096.478	100.00		

Operator notes:

Test No. **Project No:** 13849-002 **Operator**: [redacted] **Date**: [redacted]

Purpose: To combine each LR, UR and IQ composite according to blending ratio as (25.6: 48.7: 25.6) to form the "Master Composite"

Procedure: As outlined below.

Feed: 10,000 g charge for each LR, UR and IQ Composite

Grind: 70 or 75 minutes at 65% solids in the laboratory 10kg Rod Mill

Ro Flotation % Solids = 35

Notes:
Flot time and reagent dosage may change based on visual.
LREE=La,Ce,Pr,Nd,Sm,Eu,Sc
HREE=Gd,Tb,Dy,Ho,Er,Tm,Yb,Lu,Y
TREE=LREE+HREE

F9 Ro Fd K_{80} = 131 µm (Malvern)
F16 Ro Fd K_{80} = 117 µm (Malvern)
F17 Ro Fd K_{80} = 122 µm (Malvern)

General Conditions:

Stage	Reagents Added, g/t										Time, minutes			Pulp	
	LR19	Metso									Grind	Cond.	Froth	pH	Temp, ºC
Grind											70 or 75				
REO Cond 1												1			
REO Cond 2		500										5			
REO Rougher 1	400											5	7	9.7	19
REO Rougher 2	250											5	7	9.3	
REO Rougher 3	150											5	8		
REO Rougher 4	150											5	8		
REO Rougher 5	150											5	10		
REO Rougher Subtol	**1100**	**500**										**31**	**40**		

Stage	Rougher
Flotation Cell	(~25L 10 kg)
Speed rpm	1800

Metallurgical Balance

Products	Weight		Assays: U, Th, REE g/t, S %																						
	g	%	S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
F9 REO Ro Conc	883.7	8.5	11.0	1170	2060	214	731	114	6.2	80.3	10.5	47.7	7.6	19.2	2.6	14.6	1.7	198	<25	1270	1030	4330.2	382.2	4712.4
F16 REO Ro Conc 1-4	935.1	9.0	7.88	2700	5080	502	1680	263	10	150	16	73	13	27	3	20	3	301	<100	620	1680	10335	606	10941
F16 REO Ro Conc 5	311.3	3.0	0.28	207	396	42	157	37	<2	34	6	35	6	15	2	13	<2	157	<100	556	868	941	270	1211
F16 REO Ro Conc 6	433.6	4.2	0.06	53	100	11	34	8	<2	8	<2	9	<2	4	<2	4	<2	50	<100	174	156	308	83	391
F17 REO Ro Conc 1-3	293.3	2.8	9.36	2150	4040	400	1370	199	8	108	11	43	7	15	<2	10	<2	167	<100	343	1070	8267	365	8632
F17 REO Ro Conc 4	83.7	0.8	0.56	208	401	41	135	38	2	34	6	31	5	12	<2	10	<2	131	<100	436	992	925	233	1158
F9 REO Ro Tail	1799.7	17.3	0.08	27.3	47.3	4.5	16.5	3.8	0.4	3.3	0.5	2.7	0.5	1.2	<0.3	1.0	<0.5	<10	<25	106.0	36.3	124.8	20.0	144.8
F16 REO Ro Tail	3372.1	32.4	0.03	26.0	48.0	5.0	<20	4.0	<2	2.0	<2	3.0	<2	<2	<2	<2	<2	<40	<100	60.0	51.0	205.0	57.0	262.0
F17 REO Ro Tail	2293.5	22.0	0.04	<20	29.0	3.0	<20	3.0	<2	3.0	<2	4.0	<2	<2	<2	<2	<2	<40	<100	63.0	66.0	177.0	59.0	236.0
Head (Calc.)	10406.0	100.0	1.95	430.2	794.7	79.7	272.6	43.3	3.0	26.8	4.1	15.8	3.5	6.5	1.8	5.2	1.8	79.9	<100	252.3	346.4	1648.4	145.5	1794.0
Head (Direct)			1.92	407.0	767.0	81.0	254.0	42.6	1.90	26.10	3.3	16.50	2.6	6.70	0.80	5.20	0.70	62.00	<100	270.0	354.0	1578.5	123.9	1702

Products	Weight	% Distribution																					
	%	S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
F9 REO Ro Conc	8.5	47.8	23.1	22.0	22.8	22.8	22.4	17.7	25.4	21.6	25.6	18.5	25.1	12.0	23.8	8.0	21.0	3.6	42.7	25.3	21.6	22.3	21.6
F16 REO Ro Conc 1-4	9.0	36.2	56.4	57.4	56.6	55.4	54.6	30.3	50.2	34.8	41.5	33.5	37.4	14.6	34.5	14.9	33.8	11.0	22.1	43.6	54.5	37.4	53.1
F16 REO Ro Conc 5	3.0	0.4	1.4	1.5	1.6	1.7	2.6	2.0	3.8	4.4	6.6	5.1	6.9	3.2	7.5	3.3	5.9	3.7	6.6	7.5	1.7	5.6	2.0
F16 REO Ro Conc 6	4.2	0.1	0.5	0.5	0.6	0.5	0.8	2.8	1.2	2.0	2.4	2.4	2.6	4.5	3.2	4.6	2.6	5.1	2.9	1.9	0.8	2.4	0.9
F17 REO Ro Conc 1-3	2.8	13.5	14.1	14.3	14.2	14.2	13.0	7.6	11.3	7.5	7.7	5.7	6.5	3.1	5.4	3.1	5.9	3.5	3.8	8.7	13.7	7.1	13.1
F17 REO Ro Conc 4	0.8	0.2	0.4	0.4	0.4	0.4	0.7	0.5	1.0	1.2	1.6	1.2	1.5	0.9	1.5	0.9	1.3	1.0	1.4	2.3	0.4	1.3	0.5
F9 REO Ro Tail	17.3	0.7	1.1	1.0	1.0	1.0	1.5	2.3	2.1	2.1	3.0	2.5	3.2	2.8	3.3	4.8	2.2	5.3	7.3	1.8	1.3	2.4	1.4
F16 REO Ro Tail	32.4	0.5	2.0	2.0	2.0	2.4	3.0	21.8	2.4	15.7	6.1	18.6	10.0	35.1	12.4	35.9	16.2	39.8	7.7	4.8	3.9	12.7	4.6
F17 REO Ro Tail	22.0	0.5	1.0	0.8	0.8	1.6	1.5	14.9	2.5	10.7	5.6	12.6	6.8	23.9	8.4	24.4	11.0	27.0	5.5	4.2	2.3	8.9	2.8
Head (Calc.)	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0

Cumulative Products

Option A F9 conc+ F16 Conc1-4 + F17 Conc 1-3

Products	Weight	Assays: U, Th, REE g/t, S %																					
	%	S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
A: Combined REO Ro Conc	20.3	9.39	1983.5	3672.0	367.3	1239.9	191.8	8.1	115.0	13.0	58.2	9.9	22.1	2.7	16.4	2.3	239.3	72.8	853.5	1323.3	7535.5	478.9	8014.4
A: Combined REO Ro Tail	79.7	0.06	34.7	61.9	6.4	26.3	5.5	1.7	4.4	1.9	5.0	1.9	2.5	1.6	2.4	1.7	39.3	83.7	99.2	97.6	220.1	60.7	280.8
Head (Calc.)	100.0	1.95	430.2	794.7	79.7	272.6	43.3	3.0	26.8	4.1	15.8	3.5	6.5	1.8	5.2	1.8	79.9	81.5	252.3	346.4	1705.0	145.5	1850.5

Products	Assays: U, Th, REE g/t, S %																					
	S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
A: Combined REO Ro Conc	97.6	93.6	93.8	93.6	92.3	89.9	55.6	86.9	64.0	74.8	57.6	69.0	29.6	63.6	26.1	60.8	18.1	68.7	77.5	89.7	66.8	87.9
A: Combined REO Ro Tail	2.4	6.4	6.2	6.4	7.7	10.1	44.4	13.1	36.0	25.2	42.4	31.0	70.4	36.4	73.9	39.2	81.9	31.3	22.5	10.3	33.2	12.1

Option B F9 conc+ F16 Conc1-5 + F17 Conc 1-4

Products	Weight	Assays: U, Th, REE g/t, S %																					
	%	S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
B: Combined REO Ro Conc	24.1	7.96	1703.6	3156.0	316.0	1068.5	167.4	7.2	102.2	11.9	54.5	9.3	20.9	2.6	15.7	2.3	225.5	77.1	802.6	1255.7	6495.8	444.7	6940.6
B: Combined REO Ro Tail	75.9	0.05	26.0	45.2	4.6	20.0	3.9	1.6	2.9	1.7	3.6	1.7	1.9	1.6	1.9	1.7	33.7	82.9	77.6	57.8	184.3	50.6	234.8
Head (Calc.)	100.0	1.95	430.2	794.7	79.7	272.6	43.3	3.0	26.8	4.1	15.8	3.5	6.5	1.8	5.2	1.8	79.9	81.5	252.3	346.4	1705.0	145.5	1850.5

Products	Assays: U, Th, REE g/t, S %																					
	S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
A: Combined REO Ro Conc	98.2	95.4	95.7	95.6	94.4	93.2	58.2	91.8	69.5	83.0	63.9	77.4	33.7	72.6	30.3	68.0	22.8	76.6	87.3	91.8	73.6	90.4
A: Combined REO Ro Tail	1.8	4.6	4.3	4.4	5.6	6.8	41.8	8.2	30.5	17.0	36.1	22.6	66.3	27.4	69.7	32.0	77.2	23.4	12.7	8.2	26.4	9.6

Option B F9 conc+ F16 Conc1-6 + F17 Conc 1-4

Products	Weight	Assays: U, Th, REE g/t, S %																					
	%	S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
B: Combined REO Ro Conc	28.3	6.80	1460.2	2705.4	271.1	916.0	143.9	6.4	88.3	10.4	47.8	8.2	18.4	2.5	14.0	2.2	199.6	80.5	709.9	1093.6	5583.5	391.4	5974.9
B: Combined REO Ro Tail	71.7	0.05	24.5	42.0	4.3	19.2	3.6	1.6	2.6	1.6	3.2	1.6	1.8	1.6	1.8	1.6	32.8	81.9	72.0	52.1	177.1	48.7	225.8
Head (Calc.)	100.0	1.95	430.2	794.7	79.7	272.6	43.3	3.0	26.8	4.1	15.8	3.5	6.5	1.8	5.2	1.8	79.9	81.5	252.3	346.4	1705.0	145.5	1850.5

Products	Assays: U, Th, REE g/t, S %																					
	S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
A: Combined REO Ro Conc	98.3	95.9	96.2	96.2	95.0	94.0	61.0	93.0	71.5	85.3	66.3	80.0	38.2	75.8	34.9	70.6	27.9	79.5	89.2	92.5	76.0	91.2
A: Combined REO Ro Tail	1.7	4.1	3.8	3.8	5.0	6.0	39.0	7.0	28.5	14.7	33.7	20.0	61.8	24.2	65.1	29.4	72.1	20.5	10.8	7.5	24.0	8.8

Test No. : F18 **Project No:** 13849-002 **Operator**: Yashashree, Chaugule **Date**: 13-Mar-13

Purpose: To remove pyrite from the acid leaching residue of the combined rougher concentrate

Procedure: As outlined below.

Feed: ~968g wet cake of acid leaching residue from rougher concentrate

Grind: N/A Ro Flotation % Solids = ~30

Notes: Flot time and reagent dosage may change based on visual.
LREE=La,Ce,Pr,Nd,Sm,Eu,Sc
HREE=Gd,Tb,Dy,Ho,Er,Tm,Yb,Lu,Y
TREE=LREE+HREE

Conditions:

Stage	**Reagents Added, g/t**							**Time, minutes**			**Pulp**	
	PAX	**Pine Oil**	**CuSO4**					**Grind**	**Cond.**	**Froth**	**pH**	**EMF**
Grind												
REO Cond 1	40	40							2		5.6	110
REO Rougher 1										3	6	100
REO Rougher 2	30	20							2	2	6	100
REO Rougher Subtol	**70**	**60**							**4**	**5**		

Stage	Rougher
Flotation Cell	2L (1 kg)
Speed rpm	1500

Metallurgical Balance

Products	**Weight**		**Assays: S %, U, Th, REE, g/t**				**Distribution, %**			
	g	**%**	**S**	**U**	**Th**	**TREE**	**S**	**U**	**Th**	**TREE**
S Rougher Conc	184.2	26.4	34.6				91.3			
S Rougher Tail	514.0	73.6	1.18				8.7			
Head (Calc.)	698.2	100.0	10.0				100.0			
Head (Direct)										

Test No. : F19 **Project No:** 13849-002 **Operator**: Jason Garbutt **Date**: Mar 27, 1013

Purpose: To examine the conditioning time effects on flotation response and collect ~8-9% concentrate mass for hydromet tests

Procedure: Collect three rougher concentrates at 0-2 min, 2-3 and 3-4 minutes

Feed: 4000 g charge of Master Composite -6 mesh for each test (Combining two ground slurries)

Grind: 45 minutes at 65% solids in the laboratory Rod Mill GG RM#3 for each test charge Ro Flotation % Solids = 35

Notes: Flot time and reagent dosage may change based on visual. Ro Fd K_{80} = 135 μm
LREE=La,Ce,Pr,Nd,Sm,Eu,Sc LR-19: 62% FA2, 27% Aero 855, 9% Kerosene, 2% MIBC
HREE=Gd,Tb,Dy,Ho,Er,Tm,Yb,Lu,Y
TREE=LREE+HREE

Conditions:

Stage	Reagents Added, g/t										Time, minutes			Pulp		
	Metso	**LR19**		**DF250**	**Soda Ash**						**Grind**	**Cond.**	**Froth**	**pH**	**Temp, ºC**	
Grind											45			7.4		
Condition 1	500											1		9.5	20	Target pH 10.0
Condition 2		300										5				
Rougher 1				0	250								2	9.3		Target pH 10.0
Rougher 2													1			
Rougher 3													1			

Stage	Rougher
Flotation Cell	8L (4 kg)
Speed rpm	1800

Metallurgical Balance

Products	Weight		Assays: U, Th, REE g/t, S %																				
	g	**%**	**S**	**La**	**Ce**	**Pr**	**Nd**	**Sm**	**Eu**	**Gd**	**Tb**	**Dy**	**Ho**	**Er**	**Tm**	**Yb**	**Lu**	**Y**	**Sc**	**U**	**Th**	**LREE**	**HREE**	**TREE**
REO Rougher Conc 1	313.1	8.1	20.8	4230	7660	880	2570	392	18	255	27	103	16	37	5	26	4	407	<100	845	2140	15850	880	16730
REO Rougher Conc 2	36.1	0.9	3.16	550	994	115	344	57	3	41	5	25	4	11	<2	8	<2	104	<100	412	529	2163	202	2365
REO Rougher Conc 3	55.4	1.4	1.14	199	362	43	129	24	<2	20	3	16	3	8	<2	6	<2	72	<100	318	333	859	132	991
REO Rougher Tail	3449.0	89.5	0.19	71	132	14	45	10	<2	9	<2	8	<2	4	<2	3	<2	<50	<100	182	170	374	82	456
Head (Calc.)	3853.6	100.0	1.91	415.2	755.0	85.7	254.2	41.7	3.3	29.4	4.1	16.0	3.2	6.8	2.2	5.0	2.2	79.8	<100	240.0	335.8	1655.1	148.7	1803.7
Head (Direct)			1.92	407.0	767.0	81.0	254.0	42.6	1.90	26.10	3.3	16.50	2.6	6.70	0.80	5.20	0.70	62.00	<100	270.0	354.0	1653.5	123.9	1777

Products	Weight %	% Distribution																				
		S	**La**	**Ce**	**Pr**	**Nd**	**Sm**	**Eu**	**Gd**	**Tb**	**Dy**	**Ho**	**Er**	**Tm**	**Yb**	**Lu**	**Y**	**Sc**	**U**	**Th**	**LREE**	**HREE**	**TREE**
REO Rougher Conc 1	8.1	88.7	82.8	82.4	83.4	82.2	76.4	44.2	70.4	53.8	52.3	41.0	44.2	18.1	42.6	15.0	41.4	8.1	28.6	51.8	77.8	48.1	75.4
REO Rougher Conc 2	0.9	1.6	1.2	1.2	1.3	1.3	1.3	0.8	1.3	1.2	1.5	1.2	1.5	0.8	1.5	0.9	1.2	0.9	1.6	1.5	1.2	1.3	1.2
REO Rougher Conc 3	1.4	0.9	0.7	0.7	0.7	0.7	0.8	0.9	1.0	1.1	1.4	1.4	1.7	1.3	1.7	1.3	1.3	1.4	1.9	1.4	0.7	1.3	0.8
REO Rougher Tail	89.5	8.9	15.3	15.6	14.6	15.8	21.5	54.1	27.4	43.9	44.8	56.5	52.6	79.8	54.1	82.8	56.1	89.5	67.9	45.3	20.2	49.4	22.6
Head (Calc.)	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0

Cumulative Products

Products	Weight %	Assays: U, Th, REE g/t, S %																				
		S	**La**	**Ce**	**Pr**	**Nd**	**Sm**	**Eu**	**Gd**	**Tb**	**Dy**	**Ho**	**Er**	**Tm**	**Yb**	**Lu**	**Y**	**Sc**	**U**	**Th**	**LREE**	**HREE**	**TREE**
Ro Conc 1	8.1	20.8	4230.0	7660.0	880.0	2570.0	392.0	18.0	255.0	27.0	103.0	16.0	37.0	5.0	26.0	4.0	407.0	100.0	845.0	2140.0	15850	880	16730
Ro Conc 1-2	9.1	19.0	3849.4	6970.5	800.9	2339.7	357.3	16.4	232.9	24.7	94.9	14.8	34.3	4.7	24.1	3.8	375.7	100.0	800.2	1973.4	14434.3	809.9	15244.1
Ro Conc 1-3	10.5	16.5	3349.9	6066.2	697.2	2037.2	311.7	14.5	203.7	21.8	84.1	13.1	30.7	4.3	21.7	3.5	334.1	100.0	734.2	1748.9	12576.7	717.1	13293.8
REO Ro Tls	89.5	0.19	71.0	132.0	14.0	45.0	10.00	2.00	9.00	2.00	8.00	2.00	4.00	2.00	3.00	2.00	50.0	100.0	182.0	170.0	374.0	82.0	456.0

Products	% Distribution																					
	S	**La**	**Ce**	**Pr**	**Nd**	**Sm**	**Eu**	**Gd**	**Tb**	**Dy**	**Ho**	**Er**	**Tm**	**Yb**	**Lu**	**Y**	**Sc**	**U**	**Th**	**LREE**	**HREE**	**TREE**
Ro Conc 1	88.7	82.8	82.4	83.4	82.2	76.4	44.2	70.4	53.8	52.3	41.0	44.2	18.1	42.6	15.0	41.4	8.1	28.6	51.8	77.8	48.1	75.4
Ro Conc 1-2	90.2	84.0	83.7	84.7	83.4	77.7	45.0	71.7	55.0	53.8	42.2	45.7	18.9	44.1	15.9	42.6	9.1	30.2	53.3	79.0	49.4	76.6
Ro Conc 1-3	91.1	84.7	84.4	85.4	84.2	78.5	45.9	72.6	56.1	55.2	43.5	47.4	20.2	45.9	17.2	43.9	10.5	32.1	54.7	79.8	50.6	77.4
REO Ro Tls	8.9	15.3	15.6	14.6	15.8	21.5	54.1	27.4	43.9	44.8	56.5	52.6	79.8	54.1	82.8	56.1	89.5	67.9	45.3	20.2	49.4	22.6

Test No. : F20 **Project No:** 13849-002 **Operator**: Jason Garbutt **Date**: Mar 27, 1013

Purpose: To examine the conditioning time effects on flotation response and collect ~8-9% concentrate mass for hydromet tests

Procedure: Collect three rougher concentrates at 0-2 min, 2-3 and 3-4 minutes

Feed: 4000 g charge of Master Composite -6 mesh for each test (Combining two ground slurries)

Grind: 45 minutes at 65% solids in the laboratory Rod Mill GG RM#3 for each test charge

Ro Flotation % Solids = 35

Notes: Flot time and reagent dosage may change based on visual.

LREE=La,Ce,Pr,Nd,Sm,Eu,Sc
HREE=Gd,Tb,Dy,Ho,Er,Tm,Yb,Lu,Y
TREE=LREE+HREE

LR-19: 62% FA2, 27% Aero 855, 9% Kerosene, 2% MIBC

Ro Fd K_{80} = 135 μm

Conditions:

Stage	Reagents Added, g/t										Time, minutes			Pulp		
	Metso	LR19		DF250	Soda Ash						Grind	Cond.	Froth	pH	Temp, ºC	
Grind											45			7.2		
Condition 1	500											1				Target pH 10.0
Condition 2		300										10			20	
Rougher 1				0	250								2	9		Target pH 10.0
Rougher 2													1	9.4		
Rougher 3													1			

Stage	Rougher
Flotation Cell	8L (4 kg)
Speed rpm	1800

Metallurgical Balance

Products	Weight		Assays: U, Th, REE g/t, S %																					
	g	%	S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
REO Rougher Conc 1	352.8	8.9	19.7	3660	6570	747	2210	338	15	216	23	89	14	32	4	22	3	342	<100	741	1870	13640	745	14385
REO Rougher Conc 2	57.1	1.4	2.38	419	768	89	265	46	3	35	5	22	4	10	<2	8	<2	95	<100	387	453	1690	183	1873
REO Rougher Conc 3	46.0	1.2	1.13	197	360	43	129	24	<2	21	3	16	3	8	<2	6	<2	75	<100	308	335	855	136	991
REO Rougher Tail	3494.1	88.5	0.17	68	124	13	41	9	<2	8	<2	7	<2	4	<2	3	<2	<50	<100	176	162	357	80	437
Head (Calc.)	3950.0	100.0	1.96	395.4	711.8	80.0	239.0	39.1	3.2	27.1	3.9	14.6	3.1	6.6	2.2	4.8	2.1	77.0	<100	231.1	320.8	1568.5	141.5	1710.0
Head (Direct)			1.92	407.0	767.0	81.0	254.0	42.6	1.90	26.10	3.3	16.50	2.6	6.70	0.80	5.20	0.70	62.00	<100	270.0	354.0	1653.5	123.9	1777

Products	Weight	% Distribution																					
	%	S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
REO Rougher Conc 1	8.9	89.9	82.7	82.4	83.4	82.6	77.2	42.2	71.1	52.3	54.3	40.2	43.1	16.4	40.9	12.8	39.7	8.9	28.6	52.1	77.7	47.0	75.1
REO Rougher Conc 2	1.4	1.8	1.5	1.6	1.6	1.6	1.7	1.4	1.9	1.8	2.2	1.9	2.2	1.3	2.4	1.4	1.8	1.4	2.4	2.0	1.6	1.9	1.6
REO Rougher Conc 3	1.2	0.7	0.6	0.6	0.6	0.6	0.7	0.7	0.9	0.9	1.3	1.1	1.4	1.1	1.5	1.1	1.1	1.2	1.6	1.2	0.6	1.1	0.7
REO Rougher Tail	88.5	7.7	15.2	15.4	14.4	15.2	20.4	55.7	26.1	45.0	42.3	56.8	53.3	81.2	55.2	84.7	57.4	88.5	67.4	44.7	20.1	50.0	22.6
Head (Calc.)	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0

Cumulative Products

Products	Weight	Assays: U, Th, REE g/t, S %																					
	%	S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
Ro Conc 1	8.9	19.7	3660.0	6570.0	747.0	2210.0	338.0	15.0	216.0	23.0	89.0	14.0	32.0	4.0	22.0	3.0	342.0	100.0	741.0	1870.0	13640	745	14385
Ro Conc 1-2	10.4	17.3	3208.7	5762.0	655.4	1939.2	297.3	13.3	190.8	20.5	79.7	12.6	28.9	3.7	20.1	2.9	307.6	100.0	691.7	1672.7	11975.9	666.7	12642.7
Ro Conc 1-3	11.5	15.7	2904.8	5217.0	593.6	1756.5	269.8	12.2	173.7	18.7	73.2	11.6	26.8	3.5	18.6	2.8	284.1	100.0	653.0	1537.7	10853.8	613.2	11467.0
REO Ro Tls	88.5	0.17	68.0	124.0	13.0	41.0	9.00	2.00	8.00	2.00	7.00	2.00	4.00	2.00	3.00	2.00	50.0	100.0	176.0	162.0	357.0	80.0	437.0

Products	% Distribution																					
	S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
Ro Conc 1	89.9	82.7	82.4	83.4	82.6	77.2	42.2	71.1	52.3	54.3	40.2	43.1	16.4	40.9	12.8	39.7	8.9	28.6	52.1	77.7	47.0	75.1
Ro Conc 1-2	91.6	84.2	84.0	85.0	84.2	78.9	43.6	73.0	54.1	56.4	42.0	45.3	17.7	43.3	14.2	41.4	10.4	31.1	54.1	79.2	48.9	76.7
Ro Conc 1-3	92.3	84.8	84.6	85.6	84.8	79.6	44.3	73.9	55.0	57.7	43.2	46.7	18.8	44.8	15.3	42.6	11.5	32.6	55.3	79.9	50.0	77.4
REO Ro Tls	7.7	15.2	15.4	14.4	15.2	20.4	55.7	26.1	45.0	42.3	56.8	53.3	81.2	55.2	84.7	57.4	88.5	67.4	44.7	20.1	50.0	22.6

Test No. : F21 **Project No:** 13849-002 **Operator**: Jason Garbutt **Date**: Mar 27, 1013

Purpose: To examine the air flow rate effects on flotation response and collect ~8-9% concentrate mass for hydromet tests

Procedure: Collect three rougher concentrates at 0-4 min, 4-6 and 6-8 minutes

Feed: 4000 g charge of Master Composite -6 mesh for each test (Combining two ground slurries)

Grind: 45 minutes at 65% solids in the laboratory Rod Mill GG RM#3 for each test charge

Ro Flotation % Solids = 35

Notes: Flot time and reagent dosage may change based on visual.
LREE=La,Ce,Pr,Nd,Sm,Eu,Sc
HREE=Gd,Tb,Dy,Ho,Er,Tm,Yb,Lu,Y
TREE=LREE+HREE

LR-19: 62% FA2, 27% Aero 855, 9% Kerosene, 2% MIBC

Ro Fd K_{80} = 135 µm

Conditions:

Stage	Reagents Added, g/t										Time, minutes			Pulp	
	Metso	**LR19**		**DF250**	**Soda Ash**						**Grind**	**Cond.**	**Froth**	**pH**	**Temp, ºC**
Grind											45			9.5	
Condition 1	500											1			
Condition 2		300										10		9.3	20
Rougher 1				0	250								4		
Rougher 2													2	9.5	
Rougher 3													2		

Target pH 10.0

Target pH 10.0

Stage	Rougher
Flotation Cell	8L (4 kg)
Speed rpm	1800

Metallurgical Balance

Products	Weight		Assays: U, Th, REE g/t, S %																					
	g	%	S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
REO Rougher Conc 1	301.6	7.6	22.6	4300	7900	895	2610	387	18	252	26	99	15	37	4	24	3	400	<100	788	2050	16210	860	17070
REO Rougher Conc 2	52.3	1.3	2.17	420	760	90	266	45	3	35	5	23	4	11	<2	8	<2	105	<100	401	499	1684	195	1879
REO Rougher Conc 3	37.0	0.9	1.3	272	492	60	176	32	2	26	4	19	3	9	<2	7	<2	86	<100	340	382	1134	158	1292
REO Rougher Tail	3573.5	90.1	0.19	81	147	16	49	11	<2	9	<2	9	<2	4	<2	3	<2	<50	<100	199	188	406	83	489
Head (Calc.)	3964.3	100.0	1.93	408.2	748.1	84.3	247.9	40.2	3.2	28.0	3.9	16.1	3.0	6.6	2.2	4.7	2.1	77.7	<100	247.8	335.6	1631.9	144.3	1776.1
Head (Direct)			1.92	407.0	767.0	81.0	254.0	42.6	1.90	26.10	3.3	16.50	2.6	6.70	0.80	5.20	0.70	62.00	<100	270.0	354.0	1653.5	123.9	1777

Products	Weight	% Distribution																					
	%	S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
REO Rougher Conc 1	7.6	89.0	80.1	80.3	80.8	80.1	73.1	42.4	68.5	50.9	46.7	37.7	42.3	14.1	38.8	11.0	39.2	7.6	24.2	46.5	75.6	45.3	73.1
REO Rougher Conc 2	1.3	1.5	1.4	1.3	1.4	1.4	1.5	1.2	1.6	1.7	1.9	1.7	2.2	1.2	2.2	1.3	1.8	1.3	2.1	2.0	1.4	1.8	1.4
REO Rougher Conc 3	0.9	0.6	0.6	0.6	0.7	0.7	0.7	0.6	0.9	1.0	1.1	0.9	1.3	0.9	1.4	0.9	1.0	0.9	1.3	1.1	0.6	1.0	0.7
REO Rougher Tail	90.1	8.9	17.9	17.7	17.1	17.8	24.6	55.8	29.0	46.4	50.3	59.6	54.2	83.8	57.5	86.8	58.0	90.1	72.4	50.5	22.4	51.9	24.8
Head (Calc.)	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0

Cumulative Products

Products	Weight	Assays: U, Th, REE g/t, S %																					
	%	S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
Ro Conc 1	7.6	22.6	4300.0	7900.0	895.0	2610.0	387.0	18.0	252.0	26.0	99.0	15.0	37.0	4.0	24.0	3.0	400.0	100.0	788.0	2050.0	16210.0	860.0	17070.0
Ro Conc 1-2	8.9	19.6	3727.0	6845.6	776.1	2263.8	336.5	15.8	220.0	22.9	87.8	13.4	33.2	3.7	21.6	2.9	356.4	100.0	730.8	1821.0	14064.9	761.8	14826.7
Ro Conc 1-3	9.9	17.9	3400.2	6244.7	708.4	2066.4	307.7	14.5	201.6	21.1	81.3	12.4	30.9	3.5	20.3	2.8	330.9	100.0	693.9	1684.9	12841.9	704.7	13546.5
REO Ro Tls	90.1	0.19	81.0	147.0	16.0	49.0	11.00	2.00	9.00	2.00	9.00	2.00	4.00	2.00	3.00	2.00	50.0	100.0	199.0	188.0	406.0	83.0	489.0

Products	% Distribution																					
	S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
Ro Conc 1	89.0	80.1	80.3	80.8	80.1	73.1	42.4	68.5	50.9	46.7	37.7	42.3	14.1	38.8	11.0	39.2	7.6	24.2	46.5	75.6	45.3	73.1
Ro Conc 1-2	90.5	81.5	81.7	82.2	81.5	74.6	43.6	70.1	52.6	48.6	39.5	44.5	15.4	41.1	12.3	41.0	8.9	26.3	48.4	76.9	47.1	74.5
Ro Conc 1-3	91.1	82.1	82.3	82.9	82.2	75.4	44.2	71.0	53.6	49.7	40.4	45.8	16.2	42.5	13.2	42.0	9.9	27.6	49.5	77.6	48.1	75.2
REO Ro Tls	8.9	17.9	17.7	17.1	17.8	24.6	55.8	29.0	46.4	50.3	59.6	54.2	83.8	57.5	86.8	58.0	90.1	72.4	50.5	22.4	51.9	24.8

Metallurgical Balance (based on the real mass submitted to hydromet tests)

Products	Weight		Assays: U, Th, REE g/t, S %																					
	g	%	S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
F19 REO Ro Conc1	249.8		20.8	4230	7660	880	2570	392	18	255	27	103	16	37	5	26	4	407	<100	845	2140	15850.0	880.0	16730.0
F19 REO Ro Conc2	23.5		3.16	550	994	115	344	57	3	41	5	25	4	11	<2	8	<2	104	<100	412	529	2163.0	202.0	2365.0
F19 REO Ro Conc3	42.9		1.14	199	362	43	129	24	<2	20	3	16	3	8	<2	6	<2	72	<100	318	333	859.0	132.0	991.0
F20 REO Ro Conc1	328.9		19.7	3660	6570	747	2210	338	15	216	23	89	14	32	4	22	3	342	<100	741	1870	13640.0	745.0	14385.0
F20 REO Ro Conc2	46.5		2.38	419	768	89	265	46	3	35	5	22	4	10	<2	8	<2	95	<100	387	453	1690.0	183.0	1873.0
F20 REO Ro Conc3	34.5		1.13	197	360	43	129	24	<2	21	3	16	3	8	<2	6	<2	75	<100	308	335	855.0	136.0	991.0
F21 REO Ro Conc1	279.8		22.6	4300	7900	895	2610	387	18	252	26	99	15	37	4	24	3	400	<100	788	2050	16210.0	860.0	17070.0
F21 REO Ro Conc2	41.3		2.17	420	760	90	266	45	3	35	5	23	4	11	<2	8	<2	105	<100	401	499	1684.0	195.0	1879.0
F21 REO Ro Conc3	28.2		1.3	272	492	60	176	32	2	26	4	19	3	9	<2	7	<2	86	<100	340	382	1134.0	158.0	1292.0
F19 REO Ro Tail	3034.0		0.19	71	132	14	45	10	<2	9	<2	8	<2	4	<2	3	<2	<50	<100	182	170	374.0	82.0	456.0
F20 REO Ro Tail	3248.1		0.17	68	124	13	41	9	<2	8	<2	7	<2	4	<2	3	<2	<50	<100	176	162	357.0	80.0	437.0
F21 REO Ro Tail	3239.3		0.19	81	147	16	49	11	<2	9	<2	9	<2	4	<2	3	<2	<50	<100	199	188	406.0	83.0	489.0
Combined Master Conc (Calc)	1075.2		17.1	3289	5968	680	1995	303	14	196.8	20.9	80.9	12.6	29.9	3.8	20.5	3.0	321.2	<25	700.6	1686.9	12273.6	689.7	12963.2
Combined Master Tail (Calc)	9521.5		0.18	73	134	14.3	45.0	10.0	2.0	8.7	2.0	8.0	2.0	4.0	2.0	3.0	2.0	50.0	<25	185.7	173.4	304.1	81.7	385.7
Combined Master Conc (Direct)			12.3	3500	6300	700	2200	299	13.1	190	18.7	75.9	11.2	26.3	3.5	20.5	2.7	285	<25	780	1700	13037.1	633.8	13670.9
Combined Master Tail (Direct)			0.17	70	130	13.4	45	9.9	0.7	9.3	1.5	7.9	1.4	3.5	0.6	3.3	0.5	34	<25	187	182	294.0	62.0	356.0

Metallurgical Balance (based on the actual mass collected from flotation tests)

Products	Weight		Assays: U, Th, REE g/t, S %																					
	g	%	S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
F19 REO Ro Conc1	313.1	2.66	20.8	4230	7660	880	2570	392	18	255	27	103	16	37	5	26	4	407	<100	845	2140	15850.0	880.0	16730.0
F19 REO Ro Conc2	36.1	0.31	3.16	550	994	115	344	57	3	41	5	25	4	11	<2	8	<2	104	<100	412	529	2163.0	202.0	2365.0
F19 REO Ro Conc3	55.4	0.47	1.14	199	362	43	129	24	<2	20	3	16	3	8	<2	6	<2	72	<100	318	333	859.0	132.0	991.0
F20 REO Ro Conc1	352.8	3.00	19.7	3660	6570	747	2210	338	15	216	23	89	14	32	4	22	3	342	<100	741	1870	13640.0	745.0	14385.0
F20 REO Ro Conc2	57.1	0.49	2.38	419	768	89	265	46	3	35	5	22	4	10	<2	8	<2	95	<100	387	453	1690.0	183.0	1873.0
F20 REO Ro Conc3	46.0	0.39	1.13	197	360	43	129	24	<2	21	3	16	3	8	<2	6	<2	75	<100	308	335	855.0	136.0	991.0
F21 REO Ro Conc1	301.6	2.56	22.6	4300	7900	895	2610	387	18	252	26	99	15	37	4	24	3	400	<100	788	2050	16210.0	860.0	17070.0
F21 REO Ro Conc2	52.3	0.44	2.17	420	760	90	266	45	3	35	5	23	4	11	<2	8	<2	105	<100	401	499	1684.0	195.0	1879.0
F21 REO Ro Conc3	37.0	0.31	1.3	272	492	60	176	32	2	26	4	19	3	9	<2	7	<2	86	<100	340	382	1134.0	158.0	1292.0
F19 REO Ro Tail	3449.0	29.3	0.19	71	132	14	45	10	<2	9	<2	8	<2	4	<2	3	<2	<50	<100	182	170	374.0	82.0	456.0
F20 REO Ro Tail	3494.1	29.7	0.17	68	124	13	41	9	<2	8	<2	7	<2	4	<2	3	<2	<50	<100	176	162	357.0	80.0	437.0
F21 REO Ro Tail	3573.5	30.4	0.19	81	147	16	49	11	<2	9	<2	9	<2	4	<2	3	<2	<50	<100	199	188	406.0	83.0	489.0
Combined Flotation Conc	1251.2	10.6	16.6	3203	5813	663	1944	295	13.6	192.1	20.5	79.3	12.4	29.3	3.8	20.1	3.0	314.9	100.0	692.0	1651.9	12031.8	675.4	12707.1
Combined Flotation Tail	10516.6	89.4	0.18	73	134	14	45.0	10.0	2.0	8.7	2.0	8.0	2.0	4.0	2.0	3.0	2.0	50.0	100.0	185.8	173.5	379.2	81.7	460.9
Flotation Head (Calc)	11767.8	100.00	1.93	406	738	83	247	40	3	28.2	4.0	15.6	3.1	6.7	2.2	4.8	2.1	78.2	<100	239.6	330.7	1618.2	144.8	1763.0

Products	Weight	Assays: U, Th, REE g/t, S %																					
	%	S	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Tm	Yb	Lu	Y	Sc	U	Th	LREE	HREE	TREE
F19 REO Ro Conc1	2.66	28.6	27.7	27.6	28.1	27.7	25.9	14.8	24.1	18.1	17.6	13.7	14.7	6.1	14.4	5.0	13.9	2.7	9.4	17.2	26.1	16.2	25.2
F19 REO Ro Conc2	0.31	0.5	0.4	0.4	0.4	0.4	0.4	0.3	0.4	0.4	0.5	0.4	0.5	0.3	0.5	0.3	0.4	0.3	0.5	0.5	0.4	0.4	0.4
F19 REO Ro Conc3	0.47	0.3	0.2	0.2	0.2	0.2	0.3	0.3	0.3	0.4	0.5	0.5	0.6	0.4	0.6	0.4	0.4	0.5	0.6	0.5	0.2	0.4	0.3
F20 REO Ro Conc1	3.00	30.6	27.0	26.7	26.9	26.8	25.1	13.9	23.0	17.4	17.1	13.5	14.3	5.5	13.7	4.3	13.1	3.0	9.3	17.0	25.3	15.4	24.5
F20 REO Ro Conc2	0.49	0.6	0.5	0.5	0.5	0.5	0.6	0.4	0.6	0.6	0.7	0.6	0.7	0.4	0.8	0.5	0.6	0.5	0.8	0.7	0.5	0.6	0.5
F20 REO Ro Conc3	0.39	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.3	0.3	0.4	0.4	0.5	0.4	0.5	0.4	0.4	0.4	0.5	0.4	0.2	0.4	0.2
F21 REO Ro Conc1	2.56	30.0	27.1	27.4	27.5	27.1	24.6	14.2	22.9	16.8	16.3	12.4	14.2	4.7	12.8	3.6	13.1	2.6	8.4	15.9	25.7	15.2	24.8
F21 REO Ro Conc2	0.44	0.5	0.5	0.5	0.5	0.5	0.5	0.4	0.6	0.6	0.7	0.6	0.7	0.4	0.7	0.4	0.6	0.4	0.7	0.7	0.5	0.6	0.5
F21 REO Ro Conc3	0.31	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.3	0.3	0.4	0.3	0.4	0.3	0.5	0.3	0.3	0.3	0.4	0.4	0.2	0.3	0.2
F19 REO Ro Tail	29.3	2.9	5.1	5.2	4.9	5.3	7.3	18.1	9.4	14.8	15.0	18.9	17.5	26.8	18.2	27.8	18.7	29.3	22.3	15.1	6.8	16.6	7.6
F20 REO Ro Tail	29.7	2.6	5.0	5.0	4.6	4.9	6.6	18.3	8.4	15.0	13.3	19.1	17.7	27.1	18.5	28.2	19.0	29.7	21.8	14.5	6.6	16.4	7.4
F21 REO Ro Tail	30.4	3.0	6.1	6.0	5.8	6.0	8.3	18.8	9.7	15.3	17.5	19.6	18.1	27.7	18.9	28.8	19.4	30.4	25.2	17.3	7.6	17.4	8.4
Combined Flotation Conc	10.6	91.5	83.9	83.7	84.6	83.7	77.8	44.8	72.5	54.9	54.1	42.4	46.6	18.4	44.4	15.2	42.8	10.6	30.7	53.1	79.1	49.6	76.6
Combined Flotation Tail	89.4	8.5	16.1	16.3	15.4	16.3	22.2	55.2	27.5	45.1	45.9	57.6	53.4	81.6	55.6	84.8	57.2	89.4	69.3	46.9	20.9	50.4	23.4
Flotation Head (Calc)	100.00	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0

Metallurgical Balance (Assuming the same quality for flotation products and the hydromet test head)

Products	Weight		WRA Assays, %											
	g	%	SiO2	Al2O3	Fe2O3	MgO	CaO	Na2O	K2O	TiO2	P2O5	MnO	Cr2O3	V2O5
Combined Flotation Conc		10.6	47.1	10.4	16.1	0.33	3.46	0.20	4.03	0.93	0.95	0.04	0.11	<0.01
Combined Flotation Tail		89.4	88.2	5.98	1.00	0.09	0.07	0.11	3.21	0.46	0.02	0.01	0.06	<0.01
Flotation Head (Calc)		100.0	83.8	6.45	2.61	0.12	0.43	0.12	3.30	0.51	0.12	0.01	0.07	<0.01
Flotation Head (Direct)			83.4	6.43	2.75	0.10	0.37	0.11	3.30	0.49	0.11	0.01	0.02	<0.01

Products	Weight		WRA Assays, %											
	g	%	SiO2	Al2O3	Fe2O3	MgO	CaO	Na2O	K2O	TiO2	P2O5	MnO	Cr2O3	V2O5
Combined Flotation Conc		10.6	6.0	17.1	65.7	30.4	85.5	17.8	13.0	19.4	85.0	32.2	17.9	10.6
Combined Flotation Tail		89.4	94.0	82.9	34.3	69.6	14.5	82.2	87.0	80.6	15.0	67.8	82.1	89.4
Flotation Head (Calc)		100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0
Flotation Head (Direct)														

ATTACHMENT 2

An Investigation Into the Recovery of Uranium and Rare Earth Elements from the Teasdale Property

An Investigation into

THE RECOVERY OF URANIUM AND RARE EARTH ELEMENTS FROM THE TEASDALE PROPERTY

prepared for

APPIA ENERGY CORPORATION

Project 13849-001 Final Report
July 19, 2013

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Introduction

SGS Minerals Services in Lakefield, Ontario was requested to complete a metallurgical and mineralogical test program examining uranium and rare earth element recovery from a sample of quartz pebble conglomerate from the Teasdale Property by Appia Energy Corporation (Appia).

The test program was requested by Mr. Al Workman, of Watts, Griffis and McOuat Ltd (WGM), which is providing project management and consulting services to Appia, on behalf of Mr. Tom Drivas of Appia. The work was directed by Mr. John Goode, consultant to WGM. Following receipt of the samples, the program comprised sample preparation, head analysis, mineralogy, beneficiation, agitated leach tests, acid curing tests, and acid bake water leach tests.

This report summarizes the tests performed under SGS Project 13849-001 and references work performed under SGS Project 13849-002 and reported in SGS Report "An Investigation into Beneficiation Testwork to Assess the Recoverability of Uranium and Rare Earth Elements from the Teasdale Property" on June 5, 2013.

Results were provided to Mr. Goode as they became available throughout the course of the testwork.

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Testwork Summary

1. Sample Receipt and Description

SGS received a total of five shipments of material from the Teasdale Property in October and November 2012. The first two shipments contained assay rejects from Actlabs and were ultimately not used in any tests conducted at SGS because this material was very fine and could not be used to create representative composite samples of the mineralization for testing purposes. The final shipments contained a total of 172.4 kg of quarter core samples (Sample Receipts 0059-NOV12, 0081-NOV12, and 0158-NOV12). These shipments represented three composite samples covering the Upper Reef (UR, 51.8 kg), Intermediate Quartz (IQ, 27.1 kg), and Lower Reef (LR, 27.1 kg) zones and one Master Composite (66.4 kg) made in the same ratio as the three composite samples.

2. Sample Preparation

The samples from each of the zones identified by Appia (UR, IQ, and LR) were blended and crushed to 100% passing 6 mesh (3.35 mm) with samples obtained for bio-leach, head assay, mineralogical testing, and future variability testing. A portion of each of these composites was also blended to produce the Master Composite sample. A small portion of each of the four composite samples (UR, IQ, LR, and Master Comp) were submitted for detailed chemical analysis, which included whole rock analysis, REE-ICP Scan, total sulphur, $S^{=}$, and gold analysis. The analytical results are reproduced in Table 1. Total rare earth element (TREE) grades ranged from 1050 g/t (IQ) to 2190 g/t (UR) with the Master Comp containing 1680 g/t TREE. Uranium content ranged from 120 g/t (IQ) to 520 g/t (LR) with the Master Comp containing 270 g/t U.

Flotation tests were conducted by SGS as reported in “Beneficiation Testwork to Asses the Recoverability of Uranium and Rare Earth Elements from the Teasdale Property” and flotation products were blended into composite concentrates and tails for hydrometallurgical work. Conc1 and Tails1 were made up of the concentrate and tails streams of flotation tests F9, F16, and F17. Conc2 and Tails2 were made up of the concentrate and tails streams of flotation tests F19, F20, and F21. Analytical results from the flotation products are presented in Table 2. Flotation tests F19, F20, and F21 had a much better separation of rare earths from impurities with a TREE concentrate grade of 13650 g/t (Conc2) compared to 6150 g/t for tests F9, F16, and F17 producing Conc1.

Table 1 Head Assays of Composites

Sample ID	UR Comp	IQ Comp	LR Comp	Master Comp
Whole Rock Analysis, %				
SiO_2	84.2	84.1	79.7	83.4
Al_2O_3	6.27	7.25	5.77	6.43
Fe_2O_3	2.28	1.68	4.61	2.75
MgO	0.12	0.12	0.07	0.10
CaO	0.32	0.19	0.58	0.37
Na_2O	0.10	0.06	0.10	0.11
K_2O	3.43	3.59	2.80	3.30
TiO_2	0.65	0.30	0.38	0.49
P_2O_5	0.14	0.07	0.09	0.11
MnO	< 0.01	< 0.01	< 0.01	0.01
Cr_2O_3	0.04	0.02	0.02	0.02
V_2O_5	< 0.01	< 0.01	< 0.01	< 0.01
LOI	1.72	1.53	2.98	1.95
Sum	99.2	98.9	97.1	99.0
REE - ICP Scan, g/t				
La	536	259	352	407
Ce	1000	488	681	767
Pr	105	51.7	73.5	81.0
Nd	328	162	235	254
Sm	54.3	26.2	41.9	42.6
Eu	2.9	1.5	2.7	1.9
Gd	34.0	15.7	28.5	26.1
Tb	4.7	1.8	4.2	3.3
Dy	20.3	8.60	19.2	16.5
Ho	3.7	1.4	3.4	2.6
Er	8.8	3.3	8.2	6.7
Tm	2.0	0.4	1.3	0.8
Yb	7.1	2.5	5.9	5.2
Lu	2.3	< 0.5	1.0	0.7
Y	76	31	65	62
Sc	< 25	< 25	< 25	< 25
U	218	120	520	270
Th	445	218	350	354
Elemental Assays				
S, %	1.46	1.04	3.32	1.92
$S^=$, %	1.38	0.92	2.94	1.58
Au, g/t	< 0.02	< 0.02	< 0.02	< 0.02

Table 2 Assays of Flotation Products

Sample ID	Conc1	Tails1	Conc2	Tails2
Whole Rock Analysis, %				
SiO_2	54.9	90.1	47.1	88.2
Al_2O_3	12.3	4.86	10.4	5.98
Fe_2O_3	12.8	0.41	16.1	1.00
MgO	0.35	0.10	0.33	0.09
CaO	1.66	0.05	3.46	0.07
Na_2O	0.20	0.12	0.20	0.11
K_2O	4.65	2.87	4.03	3.21
TiO_2	1.00	0.27	0.93	0.46
P_2O_5	0.46	< 0.01	0.95	0.02
MnO	0.03	0.01	0.04	0.01
Cr_2O_3	0.03	< 0.01	0.11	0.06
V_2O_5	< 0.01	< 0.01	< 0.01	< 0.01
LOI	8.12	0.99	9.87	0.72
Sum	96.4	99.8	93.60	99.9
REE - ICP Scan, g/t				
La	1550	232	3500	70
Ce	2580	371	6300	130
Pr	290	35.5	700	13.4
Nd	903	131	2200	45
Sm	146	18.5	299	0.9
Eu	6.4	3.6	13.1	0.7
Gd	84.8	12.0	190	9.3
Tb	12.0	1.1	18.7	1.5
Dy	55.7	6.6	75.9	7.9
Ho	8.5	0.7	11.2	1.4
Er	20.7	2.4	26.3	3.5
Tm	2.6	0.4	3.5	0.6
Yb	15.0	1.8	20.5	3.3
Lu	2.0	< 0.5	2.7	0.5
Y	200	14	285	34
Sc	< 25	< 25	< 25	< 25
U	814	86.4	780	187
Th	1140	81.7	1700	182
Elemental Assays				
S, %	9.16	0.02	12.3	0.17
$S^=$, %	8.88	< 0.05	12.1	0.16

3. Test Procedures

3.1. Leach Test Program

A bench leach test program was conducted using the Master Comp and flotation products as feed. Charges of 1 kg charges of the Master Comp were ground in a ball mill at 50% (w/w) solids to achieve a target grind size and then filtered for tests using the whole ore as feed. Charges of varying size were used without further grinding for tests using the flotation products as feed. The various feeds and grind time as well as target or actual grind size data are presented in Table 3. Test conditions are listed in Table 4.

Table 3 Leach Feed Details

Test ID	Sample	Grind Time, min	Target P_{80}, µm	Actual P_{80}, µm	Test Charge, g
AL-1	Master Comp	30	212	152	1000
AL-1RL	AL-1 Res	--	--	152	100
AL-2	Master Comp	120 stage	75	41	1000
AL-3	Conc1	--	--	~95	1100
AL-4	Tails1	--	--	~95	400
AL-5	Tails1	--	--	~95	364
AL-6	Roast_WL Res	--	--	~95	166
AL-7	Conc2	--	--	~95	200
AL-8	Conc2	--	--	~95	200
AL-9	Tails2	--	--	~95	500
AL-10	Tails2	--	--	~95	500
POX-1	Conc1	--	--	~95	800

Table 4 AL Test Conditions

Test	Temp, ºC	% Solids	Avg. FA, g/L H_2SO_4	ORP Target, mV	Test Duration, h
AL-1	80	60	50	500	72
AL-2	80	60	50	500	72
AL-1RL	ambient	40	1 N HCl	--	2
AL-3	80	60	50	500	48
AL-4	50	60	20	500	48
AL-5	ambient	60	20	500	48
AL-6	80	40	50	500	48
AL-7	80	30	15	500	6
AL-8	80	30	pH 1.6	500	6
AL-9	ambient	50	20	500	24
AL-10	50	50	20	500	24

Test AL-1RL was a two-stage releach test performed on the final residue from AL-1 to examine redissolution of double sulphate salts possibly precipitated during AL-1 (and similarly in AL-2). The first stage was a neutralization of the leach residue in a 40 g/L sodium hydroxide lixiviant run at 70ºC and 40% (w/w) solids with additional sodium hydroxide additions to maintain pH 9 or greater for one hour before filtration and residue washing. The second stage was a leach of the neutralized solids in a 1 N hydrochloric acid lixiviant run at ambient temperature and 40% (w/w) solids for two hours before filtration.

Test POX-1 was a pressure leach test run at 210ºC, 30% (w/w) solids, preacidulated to pH 1.8 with an oxygen over-pressure of ~700 kPa for four hours.

Samples of the reaction slurry for Tests AL-1 and AL-2 were obtained at 12, 24, 48, and 72 hours (final) after the test reached operating temperature and all acid had been added. Samples of the reaction slurry for Tests AL-3, AL-4, AL-5, and AL-6 were obtained at 12 and 24 hours after the test reached operating temperature and all acid had been added. Samples of the reaction slurry for Tests AL-9 and AL-10 were obtained at 4 and 12 hours after the test reached operating temperature and all acid had been added. Kinetic samples were not taken in Tests AL-1RL, AL-7, and AL-8. The samples from Tests AL-1, AL-2, AL-9, and AL-10 were filtered with the PLS submitted for a rare earth element scan (REE), iron, and iron(II) and the residue was dried and submitted for REE. The samples from Tests AL-3, AL-4, AL-5, and AL-6 were filtered with the PLS submitted for REE, iron, iron(II), and sulphate and the residue was dried and submitted for REE and total sulphur. The samples from Test POX-1 were filtered with the PLS submitted for uranium, iron, and iron(II) and the residue was dried and submitted for uranium, total sulphur, and $S^{=}$.

Samples of the reaction slurry for Test POX-1 were obtained at one and two hours after the test reached operating temperature and all acid had been added.

At the conclusion of Tests AL-1, AL-2, AL-9, and AL-10 the entire slurry was filtered with a sample of the PLS being assayed for REE, a 30-element ICP Scan, and iron(II). The residue was dried and submitted for REE. The final slurry of Test AL-1RL was filtered with a sample of the PLS being assayed for REE while the residue was dried and submitted for REE as well. The final slurry of Tests AL-3, AL-4, AL-5, and AL-6 was filtered with a sample of the PLS being assayed for REE, ICP, iron(II), and sulphate. The residue was dried and submitted for REE, iron, sodium, potassium, and total sulphur. The final slurry of Tests AL-7 and AL-8 was filtered with a sample of the PLS being assayed for REE, ICP, and iron(II). The residue of AL-7 was used as feed for AB-6 and the residue of AL-8 was stored.

The final slurry of Test POX-1 was filtered with a sample of the PLS being assayed for REE, ICP, and iron(II). The residue was dried and submitted for REE, total sulphur, and $S^{=}$.

3.2. Acid Pug Leach Tests

One test, AP-1, was performed on a 400 g dry equivalent sample of wet (23% moisture) Conc1 feed. During this acid pug test, the wet feed was mixed with the equivalent of 40 kg/t sulphuric acid (dry sample basis) and placed in a column maintained at 70ºC with high-humidity air being pumped through the material for 24 hours. The contents of the column were then discharged into a leach reaction kettle with water at approximately 50% (w/w) solids and ambient temperature for an additional one hour with no pH or ORP control. At the conclusion of the test, the entire slurry was filtered with a sample of the PLS being assayed for REE, ICP, iron(II), and sulphate. The residue was dried and submitted for REE, iron, sodium, potassium, and total sulphur. The procedure was developed based on US Patent 4,131,639[1].

3.3. Acid Bake Water Leach Tests

Bench acid bake and water leach tests were conducted using flotation products as feed. Test conditions are listed in Table 5.

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[1] *J.T. Mather, "Sulfuric Acid Extraction of Uranium From its Ores". United States Patent 4,131,639, December 26th, 1978.*

Table 5 AB Test Conditions

Test	Feed	Temp, ºC	Acid:Ore Ratio, kg/t	Cure Time, h	Test Duration, h
AB-1	Conc1	200	1000	1	3
AB-2	Conc1	300	1000	1	3
AB-3	Conc1	250	500	1	3
AB-4	Conc1	250	300	1	3
AB-5	Roast-WL Res	250	300	1	3
AB-6	AL-7 Res	250	600	1	3

The acid bake portion of Test AB-5 was run on the residue from a roast and water leach test which used Conc1 as the original feed. AB-5 was run in parallel with an acid leach test (AL-6) to compare each extraction method.

The acid bake portion of Test AB-6 was run on the residue from acid leach Test AL-7 which used Conc2 as the original feed. AB-6 was run to evaluate acid pre-leaching and PLS recycling, with the PLS of this test being forwarded to acid leach test AL-8.

Upon removal from the furnace, the acid-baked solids were allowed to cool before being manually pulverized and then placed in a leach reaction kettle with water at 10% (w/w) solids on an original dry feed basis. The temperature was maintained at 90ºC and the pH was maintained below pH 1.5 for the four-hour duration of the water leach.

At the conclusion of the leach portion of Tests AB-1, AB-2, AB-3, and AB-4 the entire leach slurry was filtered with a sample of PLS being assayed for REE, silicon, phosphorous, iron, aluminum, calcium, manganese, chloride, and fluoride. The residue was washed and a sample of the wash was assayed for cerium, yttrium, phosphorous, and iron. The residue was dried and submitted REE by ICP-MS, silicon, phosphorous, iron, aluminum, calcium, manganese, chloride, fluoride, total sulphur, cerium, lanthanum, neodymium, and praseodymium.

At the conclusion of the leach portion of Test AB-5 the entire leach slurry was filtered with a sample of PLS being assayed for REE, ICP, and silicon. The residue was washed and a sample of the wash was assayed for cerium, yttrium, and iron. The residue was dried and submitted for REE by ICP-MS and total sulphur.

At the conclusion of the leach portion of Test AB-6 the entire leach slurry was filtered with a sample of PLS being assayed for REE and ICP. The residue was washed and dried before being submitted for REE by ICP-MS and iron.

3.4. Roast and Water Leach Test

One test was performed on a 400 g dry equivalent sample of wet (23% moisture) Conc1 feed. The wet feed was placed in a preheated muffle furnace at 750ºC for two hours. Upon removal from the furnace, the roasted solids were allowed to cool before being manually pulverized and then placed in a leach reaction kettle with water at 10% (w/w) solids on an original dry feed basis. The temperature was maintained at 90ºC and the pH was not controlled for the four-hour duration of the water leach. At the conclusion of the test, the entire slurry was filtered with a sample of the PLS being assayed for REE, silicon, and iron. The residue was washed and a sample of the wash was assayed for cerium, yttrium, and iron. The residue was dried and submitted for REE by ICP-MS, total sulphur, and $S^{=}$. The residue of this test was used as feed for Tests AL-6 and AB-5.

3.5. Liquor Neutralization and Ion Exchange (IX) Tests

A series of neutralization tests were performed on a combined PLS resulting from the leach tests. Assay results from the combined PLS can be found in Table 6. For one of these tests, the combined PLS was placed in a 1000 mL reaction kettle and agitated at ambient temperature for a one hour reaction time. The pH was increased to pH 2 using a 20% (w/w) limestone slurry. Two other tests were performed using the same feed in a 500 mL reaction kettle at 40ºC in a stepwise pH neutralization. pH targets were 2.0, 2.5, 3.0, 3.5, 4.0, 4.5, 5.0, and 6.0 with a 30 minute stabilization period between targets. One of these tests investigated the use of magnesium oxide as a 25% (w/w) slurry and the other test made use of a 25% (w/w) slurry of limestone.

A solution sample for Neut-1 was taken from the final filtrate and assayed for uranium, neodymium and iron content with the bulk of the solution forwarded to the uranium ion exchange test (UIX-1). The residue from Neut-1 was washed and dried before being submitted for uranium, iron, neodymium, yttrium, calcium, and sulphur content. Solution samples were taken at each pH point for Neut-2 and Neut-3 and assayed for REE, iron, and sodium. Final residues were washed and dried before being submitted for REE and iron content.

Table 6 Combined PLS (Neut Feed) Assay Results

Neut Feed	Ag	Al	As	Ba	Be	Bi	Ca	Cd	Co	Cr
(mg/L)	< 0.2	3430	< 3	0.25	< 0.1	< 1	822	0.09	16.7	35.4
Neut Feed	Cu	Fe	K	Li	Mg	Mn	Mo	Na	Ni	P
(mg/L)	39.6	5460	1320	6	241	97.2	0.7	113	30	152
Neut Feed	Pb	Sb	Se	Sn	Sr	Ti	Tl	V	W	Zn
(mg/L)	6	< 1	< 3	2	1.941	51.4	< 3	2.1	< 2	14.8
Neut Feed	Ce	Dy	Er	Eu	Gd	Ho	La	Lu	Nd	Pr
(mg/L)	221	15.2	6.28	1.05	18.0	2.52	111	0.61	83.5	24.3
Neut Feed	Sc	Sm	Tb	Th	Tm	U	Y	Yb	Fe^{2+}	pH
(mg/L)	0.73	18.0	2.81	268	0.81	282	62.2	4.66	2260	1.13

One uranium ion exchange test was performed to remove uranium from the neutralized combined PLS (UIX-1). This test used 600 mL of the filtrate from Neut-1 with a measured pH of approximately pH 2.1. The solution was contacted with 3 mL wet settled resin (wsr) (5 mL wsr/L filtrate) using Ambersep 920U in sulphate form for 90 min for each contact. The Neut-1 filtrate contained 281 mg/L uranium, 42 mg/L neodymium and 4630 mg/L iron. This filtrate was subjected to five consecutive contacts with fresh resin at the same ratio for each contact. A sample of the solution was collected after each contact and submitted for uranium and iron assays with the bulk of the final filtrate forwarded to the final solution neutralization test (Neut-4). The resin for each contact was screened out and dried with the weight recorded.

One neutralization test was performed using the barren solution from a uranium ion exchange test (UIX-1 Barren). The UIX-1 filtrate after the fifth and final contact contained 14 mg/L uranium and 3980 mg/L iron. This barren solution was placed in a 500 mL reaction kettle and heated to 40ºC. The pH was slowly adjusted using a 25% (w/w) magnesium oxide slurry holding for ten minutes at various pH points to allow the reaction to stabilize before proceeding to pH 3.0 and filtering the resulting slurry. A solution sample was taken for REE, ICP, and iron(II) with the bulk of the solution forwarded to the rare earth element ion exchange testwork (REEIX-1).

A series of rare earth element ion exchange contacts were performed to selectively recover rare earth elements from the neutralized UIX-1 Barren solution (Neut-4 filtrate). This test used the filtrate from Neut-4 with a measured pH of approximately pH 3.1 and contacted it with OC1026 ion exchange resin at ratios of 120 mL wsr/L barren, 60.0 mL wsr/L barren, and 20.0 mL wsr/L barren using 100 mL, 180 mL, and 180 mL of filtrate, respectively. Each contact was two hours in duration in agitated flasks at ambient temperature. Complete assay results for the Neut-4 filtrate can be found in Table 15. Key assay results are summarized in Table 7.

Table 7 Neut-4 Filtrate (REEIX-1 Feed) Assay Results

REEIX-1 Feed	Ce	La	Nd	Y	U	Th	Ca	Fe	Fe^{2+}
(mg/L)	107	64.9	42.4	58.4	14.6	107	715	2020	1900

4. Test Results

4.1. Whole Ore Master Comp Leach Results

Results for the tests performed using the Master Comp as feed (AL-1, AL-2, AL-1RL) are presented in Table 8.

Table 8 Whole Ore Master Comp Extraction Results

Test ID	Sample	Avg. T, ºC	Avg. FA, g/L H_2SO_4	Avg. ORP, mV	H_2SO_4, kg/t	$NaClO_3$, kg/t	Extraction, % La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Y	U	Th
AL-2	Master Comp	72.4	52	487	62.0	0.4	10.2	9.75	12.4	11.9	25.1	34.0	49.3	73.6	80.1	83.6	84.2	81.7	96.9	78.0
AL-1	Master Comp	70.7	54	439	69.1	0.8	9.34	8.08	9.90	11.0	24.2	34.4	48.2	62.9	77.5	72.7	80.2	80.5	97.3	79.7
+ RL[1]	AL-1 Res	23.2	30 (HCl)	658	+ 36.8 HCl	0	10.3	9.11	9.77	12.1	22.8	31.5	46.6	62.0	75.6	71.9	79.5	78.4	97.1	85.0

[1] Extractions are combined overall extraction from AL-1 feed.

Sulphuric acid addition for Tests AL-1 and AL-2 were 69.1 kg/t and 62.0 kg/t on a dry feed basis, respectively. Acid consumed was 45.6 kg/t and 35.4 kg/t respectively for AL-1 and AL-2. Sodium chlorate addition for these tests was 0.8 kg/t and 0.4 kg/t on a dry feed basis, respectively. Metal extractions were similar for the two tests with only slight benefits apparent with AL-2 conditions (stage grinding to 75 µm target P_{80}).

Figure 1 and Figure 2 show the timed elemental extractions for AL-1 and AL-2, respectively. In each test there was a significant drop in rare earth extraction, particularly the light rare earths from 48 h to 72 h. This was presumed to be due to the precipitation of insoluble double sulphate salts towards the end of the leaching time. Rare earth elements form double sulphate salts when cations such as K and Na are present. Potassium in the leach PLS was found to increase from 1.2 g/L to 3.9 g/L from 12 h to 72 h of leaching in AL-1 and similarly from 1.9 g/L to 4.8 g/L in AL-2. Sodium was also present at about 300 mg/L in each test after 72 h.

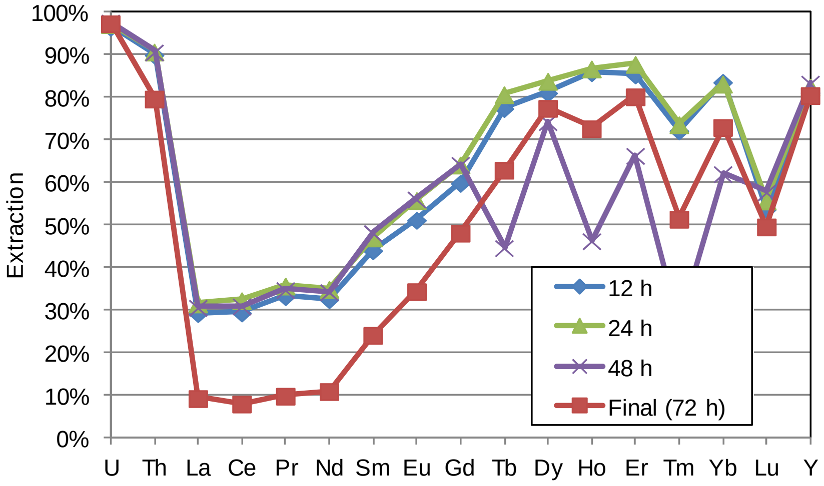


Figure 1 AL-1 Elemental Extraction

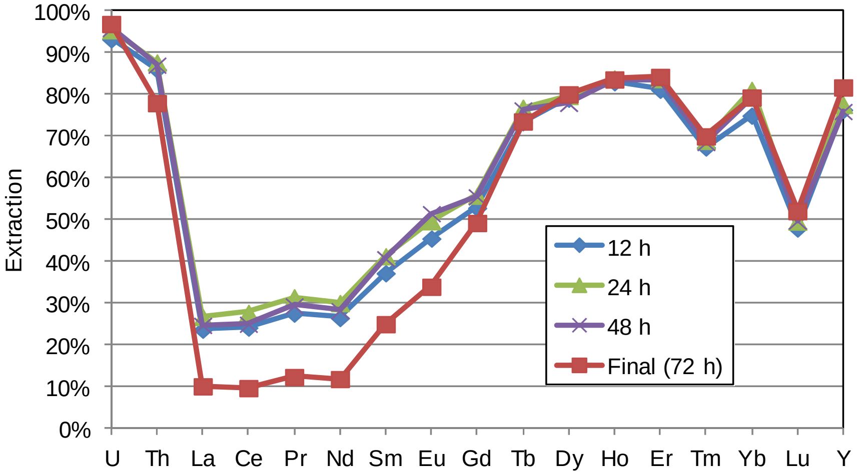


Figure 2 AL-2 Elemental Extraction

An alkaline wash followed by HCl releach of AL-1 residue was performed in attempt to redissolve the precipitated double sulphate salts. When adding the releach stage to the AL-1 residue, there was 36.8 kg/t hydrochloric acid addition on a dry feed basis. Overall metal extractions from the combined leach-releach test were generally comparable to the original results obtained with Test AL-1 (Figure 3), indicating the loss in LREE was either not due to double sulphate precipitation or the releach scheme was not effective in redissolving the salts.

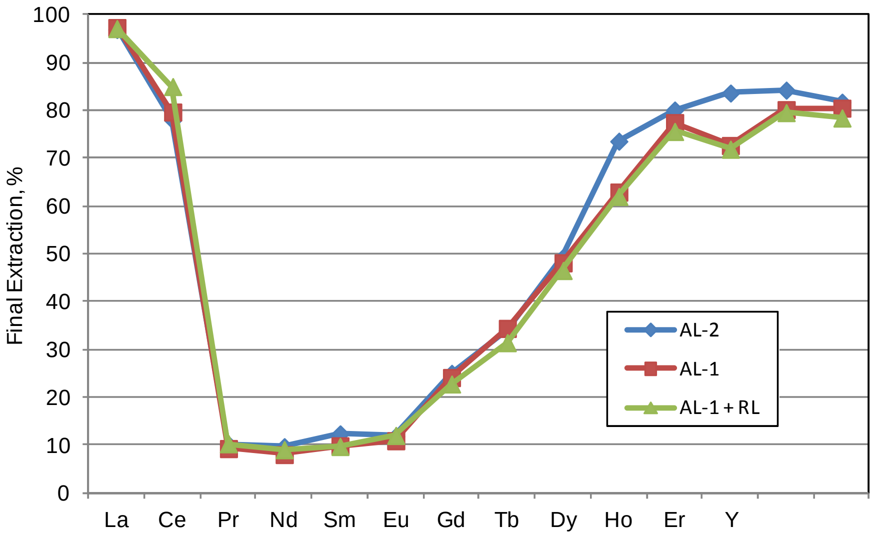


Figure 3 Whole Ore Master Comp Final Rare Earth Extractions

4.2. Flotation Conc1 Leach Results

Results for the tests performed using the flotation Conc1 as feed are summarized in Table 9. Figure 4 plots the elemental extractions for the various processing options tested. Each described in further detail in the subsequent subsections.

Table 9 Flotation Conc1 Extraction Results

Test ID	Sample	Avg. T, ºC	Avg. FA, g/L H_2SO_4	Avg. ORP, mV	H_2SO_4, kg/t	$NaClO_3$, kg/t	Extraction, %													
							La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Y	U	Th
AL-3	Conc 1	81.4	39	438	138	0.6	29.8	31.2	34.2	37.3	46.4	52.7	61.5	65.8	78.9	82.2	85.2	85.7	96.4	84.4
POX-1	Conc 1	210	44	493	39.8	--	20.1	24.7	26.4	27.6	35.9	40.0	47.9	60.0	70.6	73.4	79.7	79.9	97.9	74.7
AP-1	Conc 1	21.0	0	222	40.1	--	6.42	6.78	7.57	8.06	10.9	13.3	17.4	21.9	26.2	28.4	30.1	30.6	26.2	0.43
AB-1	Conc 1	90.5	52	576	1000	--	55.9	72.4	82.1	87.3	89.8	85.3	91.3	90.1	89.9	90.0	89.4	94.2	98.4	90.9
AB-2	Conc 1	89.7	12	561	1000	--	66.0	73.7	76.6	79.7	78.6	65.7	78.6	66.7	75.0	73.1	76.0	87.7	97.1	78.5
AB-3	Conc 1	92.4	14	510	585	--	55.9	67.9	73.6	75.8	77.6	72.3	81.0	76.5	81.1	81.1	83.3	86.2	97.3	75.1
AB-4	Conc 1	89.3	4.5	430	316	--	45.9	49.7	51.8	53.7	56.2	57.0	63.7	58.9	68.0	67.4	72.0	73.6	95.5	51.9
Roast-WL	Conc 1	88.6	0	-30.8	0	--	0.02	0.01	0.08	0.05	0.21	3.23	0.24	2.02	0.79	1.99	1.72	0.04	0.77	0.02
+ AB-5[1]	Roast-WL Res	81.3	5.1	667	321	--	43.2	44.5	49.8	53.4	55.9	47.9	59.2	64.7	65.0	63.2	65.1	63.4	78.8	33.4
+AL-6[1]	Roast-WL Res	73.8	55	701	147	0	13.7	17.1	16.3	21.4	27.5	35.7	45.7	54.3	58.2	63.2	63.8	63.8	79.3	50.7

[1] Extractions are combined overall extraction from Roast-WL feed.

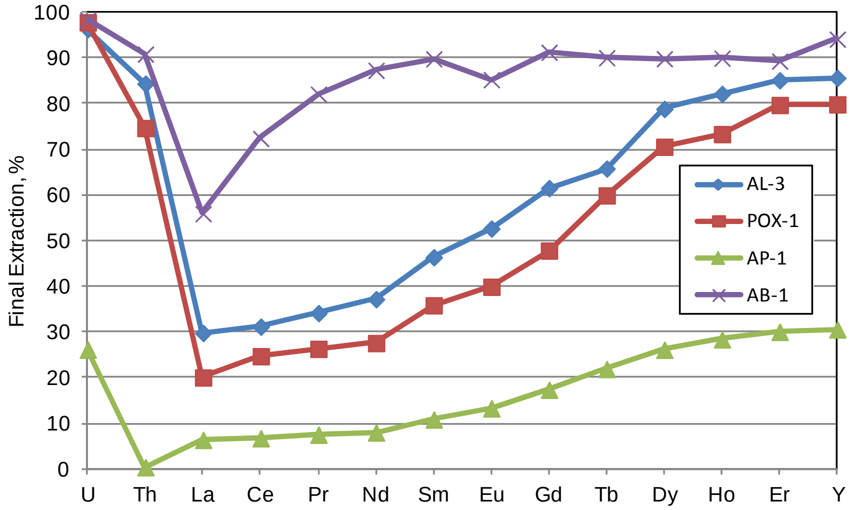


Figure 4 Conc1 Elemental Extraction for Various Process Options

4.2.1. Atmospheric Leach of Flotation Concentrate

In AL-3 the flotation concentrate was leached with sulphuric acid addition of 138 kg/t and 0.6 kg/t sodium chlorate. Elemental extractions from the flotation concentrate are shown in Figure 5. Uranium extraction was 93% after 12 h and 96% after 48 h of leaching. Final LREE extraction was between 30% and 50% with extractions generally increasing with increasing atomic number to ~85% for the HREE. After 24 hours of leaching there was little further increase in either U or REE extraction.

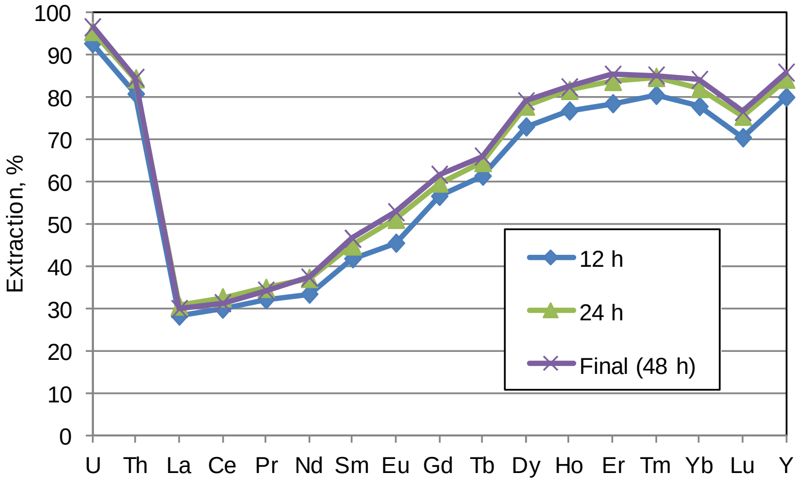


Figure 5 AL-3 Elemental Extractions

4.2.2. Pressure Oxidation

A batch pressure oxidation (POX) test was done on the flotation concentrate (Conc1), at 210ºC with 700 kPa oxygen overpressure for four hours at a pulp density of 30% solids by weight. The slurry was preconditioned to pH 1.8 prior to POX using 39.3 kg/t of sulphuric acid. Greater than 90% sulphide oxidation was achieved, generating 250 kg/t of acid of which 104 kg/t remained as free acid, 110 kg/t as sulphate associated with dissolved species and 57 kg/t as insoluble sulphate in the leach residue.

The uranium dissolution and sulphide oxidation kinetics are illustrated in Figure 6.

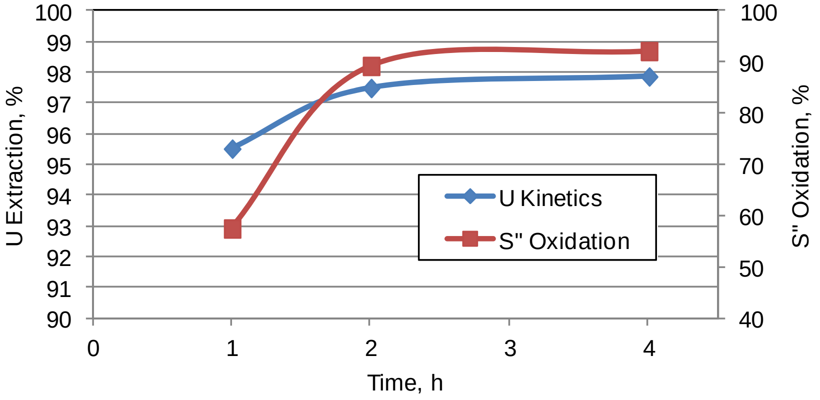


Figure 6 Uranium Leach and Sulphide Oxidation Kinetics

After two hours uranium dissolution was greater than 98% and 90% of the sulphides had been oxidized. Under the conditions tested, complete sulphide oxidation was expected – additional tests will be required to confirm the incomplete oxidation and determine why there was ~0.7% residual sulphide in the leach residue. The ferric:ferrous iron ratio of the POX PLS decreased from 0.25 to ~230 from one hour to four hours and just 20 mg/L ferrous iron suggests that the oxidation was more complete than the sulphide analyses showed and perhaps the sulphide assay was positively biased by insoluble sulphate compounds (alunite, barite etc.). Additional work would be required to confirm this.

The REE extraction is shown in Figure 7, with similar extractions compared to the atmospheric leach of the flotation concentrate; LREE extraction ranging from 20-40% and HREE from 40-80% in POX-1 compared to acid leach LREE extraction ranging from 30-50% and HREE from 50-85%.

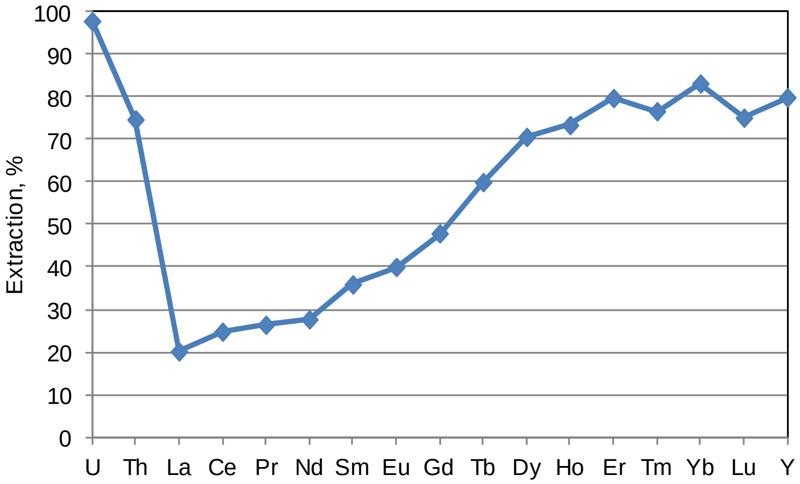


Figure 7 REE Extraction – 4 h POX

4.2.3. Acid Pugging

Acid pugging with 40 kg/t acid addition gave low extractions, as illustrated in Figure 8.

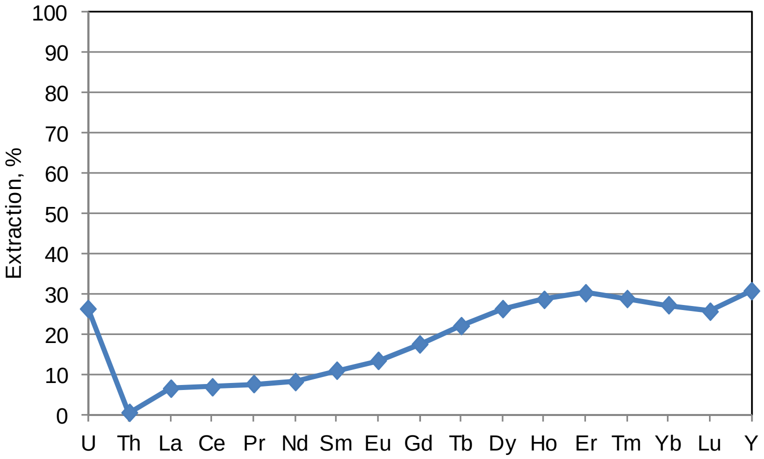


Figure 8 Acid Pugging Elemental Extraction

4.2.4. Acid Bake and Water Leach

Results from the series of acid bake and water leach tests on the flotation concentrate (Conc1) are listed in Table 10 and shown in Figure 9. The highest extractions were with 1000 kg/t acid addition, and comparable for 200ºC and 300ºC, however acid consumed was much lower at 200ºC; 599 kg/t versus 886 kg/t at 300ºC. Presumably the difference is due to acid decomposed and released in off-gasses at the higher temperature. Reducing the acid dosage to 300 kg/t resulted in a 20% reduction in LREE extraction and a 10% reduction in HREE extraction.

Table 10 Acid Bake Results

Conditions				Extractions			
Test ID	Acid Bake		Acid Cons'd	U	TREE	LREE	HREE
	Temp, ºC	A/O, kg/t	kg/t			La-Sm	Eu-Lu+Y
AB-1	200	1000	599	98%	72%	71%	92%
AB-2	300	1000	886	97%	73%	73%	81%
AB-3	250	500	341	97%	67%	67%	83%
AB-4	250	300	254	96%	51%	50%	70%

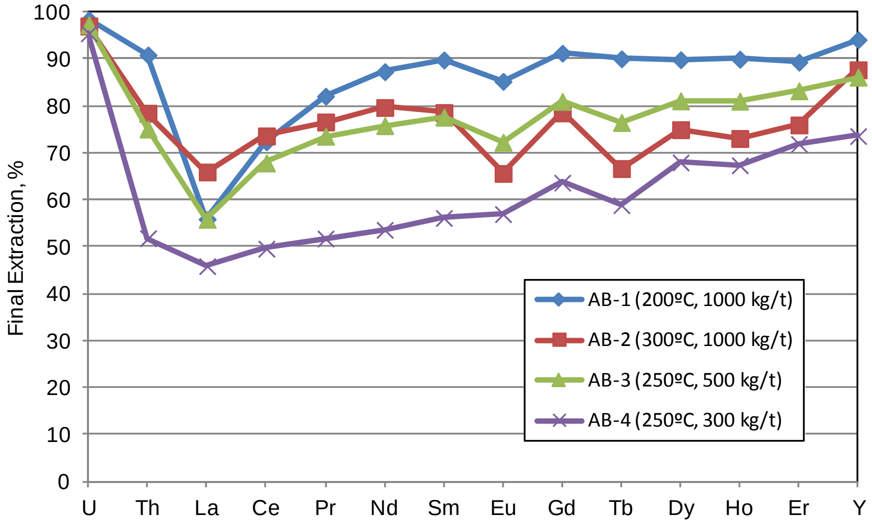


Figure 9 Acid Bake Water Leach Extractions

4.2.5. Roasting

Flotation concentrate (Conc1) was roasted at 750ºC for 2 hours and then water leached. There was no significant elemental extraction in the water leached at neutral pH (~8.5-9).

The roasted, water leached residue was then subjected to acid baking at 250ºC with 300 kg/t acid addition (AB-5). The acid bake results were comparable to those under similar conditions without prior roasting, as illustrated in Figure 10. Uranium (and thorium) extractions were negatively impacted by the roasting step and REE extractions were largely unaffected with expectation of HREE which were slightly lower after roasting.

The roasted, water leached residue was also subjected to atmospheric acid leaching (AL-6) in 50 g/L sulphuric acid at 80ºC for 48 hours. Leach extractions are plotted in Figure 11 and it is evident that roasting of the concentrate prior to atmospheric acid leaching negatively affected the leach efficiency of U and REE. Acid consumption was reduced by ~25% by roasting, from ~100 kg/t for AL-3 to 75 kg/t for AL-6.

Figure 12 illustrates the elemental extractions for the various roasted concentrate leach tests.

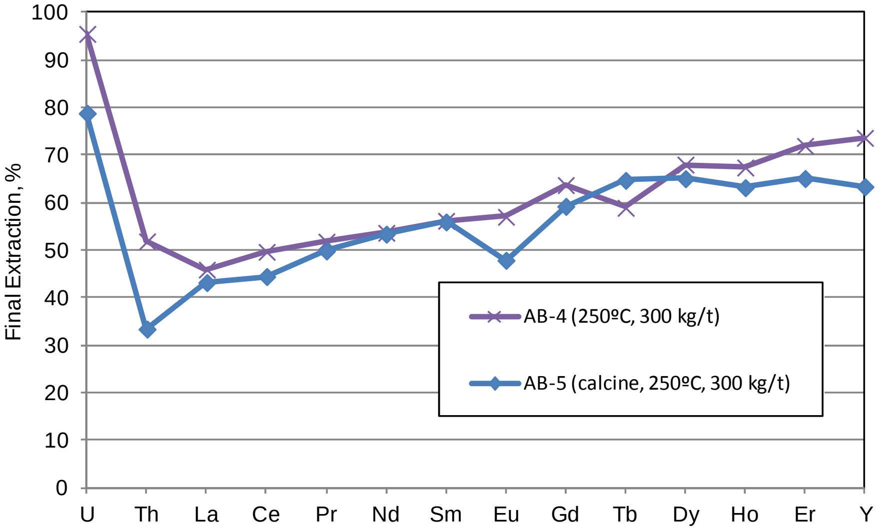


Figure 10 Effect of Roasting on Acid Bake Water Leach Extractions

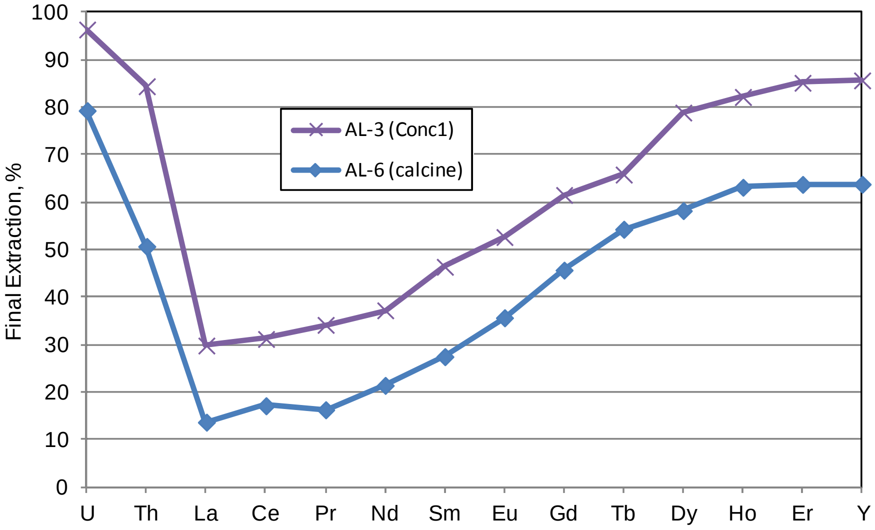


Figure 11 Effect of Roasting on Atmospheric Leach Extractions

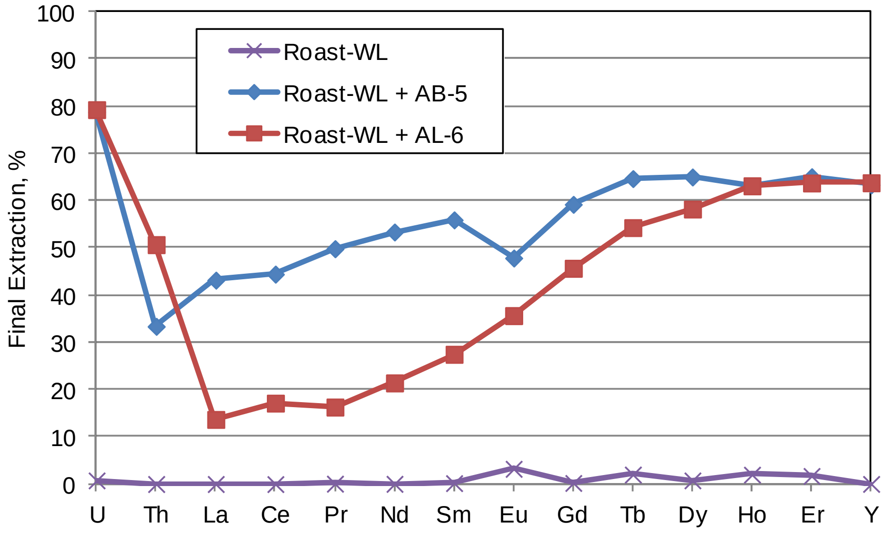


Figure 12 Roasted Concentrate - Elemental Extraction

4.3. Flotation Tails1 Leach Results

Results for the tests performed using the flotation Tails1 as feed are presented in Table 11.

Table 11 Flotation Tails1 Extraction Results

Test ID	Sample	Avg. T, °C	Avg. FA, g/L H_2SO_4	Avg. ORP, mV	Added H_2SO_4, kg/t	Cons'd H_2SO_4, kg/t	$NaClO_3$, kg/t	Extraction, % U	Th	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Y
AL-4	Tails 1	52.3	33	508	27.7	15.5	0.0	94.2	94.8	40.2	44.8	49.0	51.4	67.6	>37	80.1	>51	85.7	>62	>72	>69
AL-5	Tails 1	24.0	18	491	27.2	13.4	0.4	89.4	91.0	33.8	36.0	45.4	>32	>57	>9	>35	>20	>59	>19	>38	>22

Sulphuric acid addition for Tests AL-4 and AL-5 were 27.7 kg/t and 27.2 kg/t on a dry feed basis, respectively and 15.5 kg/t and 13.4 kg/t H_2SO_4 was consumed in each test, respectively. Sodium chlorate addition for Test AL-5 was 0.4 kg/t on a dry feed basis, while no addition was required for Test AL-4. Metal extractions were notably increased with AL-4 conditions (leach temperature of 50ºC).

Figure 13 shows the final leach extraction of Tests AL-4 and AL-5 by element. The saw-tooth effect in the graph is a result of the leach residue assaying below the detection limit for many of the elements (for those that are reported as “>” in Table 11 the residue was below analytical detection limit).

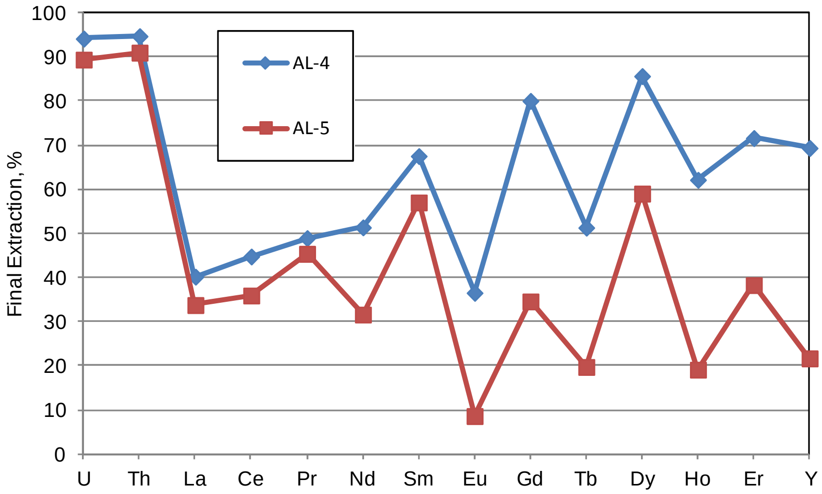


Figure 13 Flotation Tails1 Final Extractions

4.4. Flotation Conc2 Leach Results

A second flotation concentrate (Conc2) was used for a series of tests to evaluate a counter-current preleach arrangement. The concentrate was first leached under atmospheric conditions (AL-7) and the residue from that acid baked and water leached (AB-6). The water leach PLS was then used in a preleach of fresh concentrate (AL-8).

Results for the tests performed using the flotation Conc2 as feed are presented in Table 12.

Table 12 Flotation Conc2 Extraction Results

Test ID	Sample	Avg. T, ºC	Avg. FA, g/L	Avg. ORP, mV	Added H_2SO_4, kg/t	Cons'd H_2SO_4, kg/t	$NaClO_3$, kg/t	Extraction, %													
								U	Th	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy	Ho	Er	Y
AL-7	Conc 1	83.7	14	345	105	74.2	0.9	72.8	58.0	15.1	16.3	16.0	17.8	24.8	31.1	31.9	50.0	56.5	67.7	70.4	63.1
+ AB-6[1]	AL-7 Res	90.0	9.4	541	600	466	--	99.0	92.6	93.8	95.2	95.5	96.3	95.2	92.3	95.0	94.9	94.7	95.1	94.8	96.2
AL-8[2]	Conc 1	81.6	4.7	468	35.6	26.4	2.8	91.4	58.8	28.2	28.6	27.3	29.5	39.0	42.9	47.9	64.6	75.1	84.2	87.2	83.8

[1] Extractions are combined overall extraction from AL-7 feed.

[2] Pre-leach using PLS from AB-6, extraction calculated by increase in metal units in solution.

Sulphuric acid addition for Test AL-7 was 105 kg/t on a dry feed basis, with 74 kg/t consumed. Sodium chlorate addition for Test AL-7 was 0.9 kg/t on a dry feed basis. Metal extractions were low with 16% LREE extraction and 53% HREE extraction.

Acid baking of the preleached residue of AL-7 resulted in an additional 466 kg/t sulphuric acid consumed on a dry feed basis. Overall metal extractions were significantly improved with the acid bake test and slightly improved over a similar direct acid bake test on a similar feed. Combined preleach and acid bake extractions were 95% for both LREE and HREE. This can be compared to the results obtained from AB-3 which achieved extractions of 67% for LREE and 83% for HREE.

Performing an acid leach using fresh Conc2 feed and the PLS from AB-6 required an additional 35.6 kg/t sulphuric acid addition on a dry feed basis resulting in a slurry pH of ~1.2. Sodium chlorate addition was 2.8 kg/t on a dry feed basis. Metal extractions in this test were higher than those found in AL-7, indicating the additional acid may not have been required and the residual acid after baking could be sufficient for the preleach, additional testing will be required to confirm this.

Figure 14 shows the final leach extraction of Tests AL-7 and AL-8 and includes the overall extraction of Test AL-7 with acid bake (AB-6) by element.

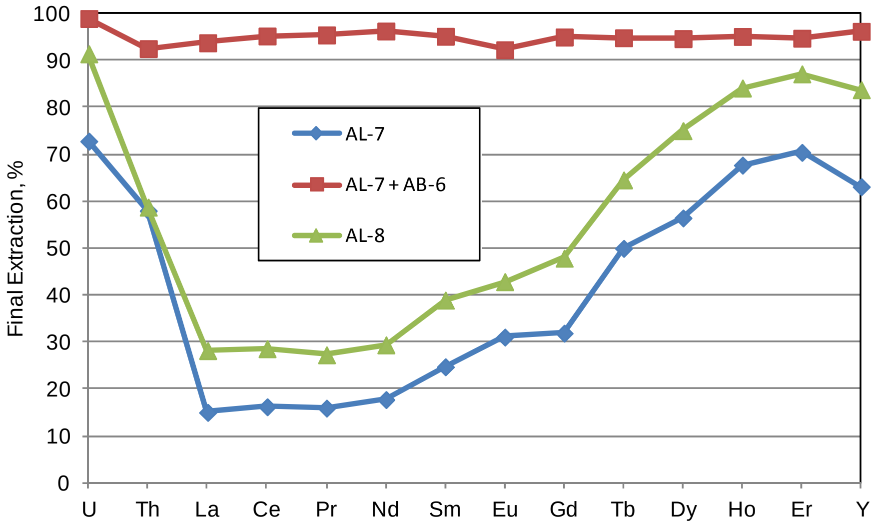


Figure 14 Flotation Conc2 Final Extractions

4.5. Flotation Tails2 Leach Results

Results for the tests performed using the flotation Tails2 as feed are presented in Table 13.

Table 13 Flotation Tails2 Extraction Results

Test ID	Sample	Avg. T, ºC	Avg. FA, g/L H_2SO_4	Avg. ORP, mV	Added H_2SO_4, kg/t	Cons'd H_2SO_4, kg/t	$NaClO_3$, kg/t	Extraction, %													
								U	Th	La	Ce	Pr	Nd	Sm	Eu[1]	Gd[1]	Tb[1]	Dy[1]	Ho[1]	Er[1]	Y[1]
AL-9	Tails 2	22.7	24	508	36.6	13.4	0.6	89.3	89.7	32.0	36.0	40.0	43.7	65.6	54.6	67.7	83.7	81.5	82.1	79.7	76.5
AL-10	Tails 2	51.4	19	518	37.7	20.4	0.0	94.1	94.9	39.4	43.6	48.4	51.2	69.2	60.0	79.2	89.0	95.1	90.5	93.4	88.6

[1] Extractions calculated based on PLS and direct head, all others based on PLS and calculated head due to assays below detection limits.

Sulphuric acid addition for Tests AL-9 and AL-10 were 36.6 kg/t and 37.7 kg/t on a dry feed basis, respectively and 13.4 kg/t and 20.4 kg/t acid was consumed in each test, respectively. Sodium chlorate addition for Test AL-9 was 0.6 kg/t on a dry feed basis while no addition was required for Test AL-10. Metal extractions were notably increased with AL-10 conditions (leach temperature of 50ºC).

Figure 15 shows the final leach extraction of Tests AL-9 and AL-10 by element.

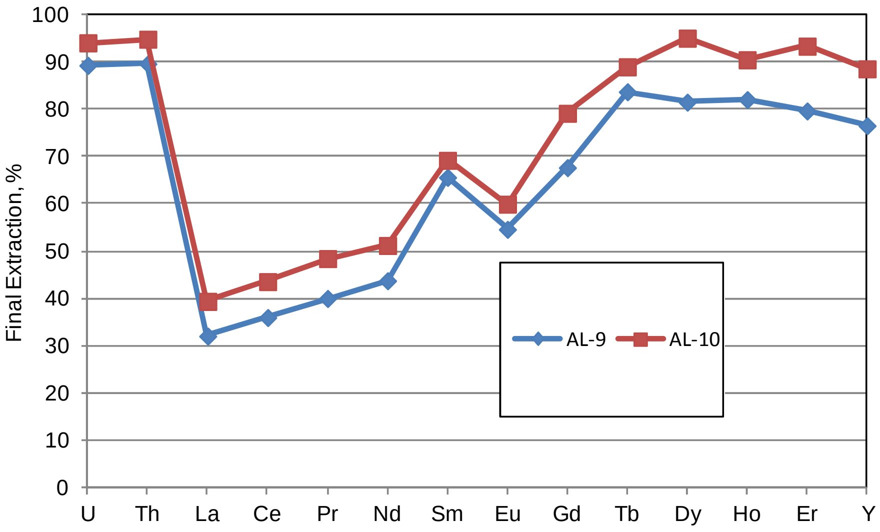


Figure 15 Flotation Tails2 Final Extractions

4.6. Liquor Neutralization and Ion Exchange Results

Results for the neutralization tests performed using Combined PLS (that was produced by combining liquors from AL-1, AL-2, AL-7, Al-8, POX-1 and AP-1) as feed are presented in Table 14 and Table 15. The composition of the PLS is listed in Table 16.

Table 14 Neutralization Reagent Addition Results

Test ID	Sample	Neutralizing Agent	Avg. T, ºC	Reagent Addition to Target pH, kg/t PLS							
				2.0	2.5	3.0	3.5	4.0	4.5	5.0	6.0
Neut-1	Combined PLS	20% Limestone	--	27.8	--	--	--	--	--	--	--
Neut-2	Combined PLS	25% MgO	46.8	3.5	6.3	7.8	11.1	14.2	14.8	16.7	18.1
Neut-3	Combined PLS	25% Limestone	45.6	19.5	21.6	25.7	31.7	37.4	39.5	45.3	69.8

Table 15 Neutralization Assay and Precipitation Results

Neut-1	La		Ce		Nd		Y		U		Th		Fe	
pH	mg/L	% ppt	mg/L	% ppt	mg/L	% ppt	mg/L	% ppt	mg/L	% ppt	mg/L	% ppt	mg/L	% ppt
Feed	111	--	221	--	83.5	--	62.2	--	282	--	268	--	5460	--
2.0	--	--	--	--	42	48	51.7[1]	8[1]	281	1	--	--	4630	1
Neut-2	La		Ce		Nd		Y		U		Th		Fe	
pH	mg/L	% ppt	mg/L	% ppt	mg/L	% ppt	mg/L	% ppt	mg/L	% ppt	mg/L	% ppt	mg/L	% ppt
Feed	111	--	221	--	83.5	--	62.2	--	282	--	268	--	5460	--
2.0	116	-16	230	-15	87.8	-16	64.0	-14	289	-14	276	-14	4750	4
2.5	122	-21	239	-19	91.2	-20	66.6	-17	299	-16	258	-6	3250	35
3.0	118	-19	231	-17	87.7	-18	65.0	-17	291	-15	177	26	2410	51
3.5	115	-17	226	-16	85.0	-15	62.8	-14	274	-10	65.5	72	2090	57
4.0	97.4	1	180	8	64.2	13	42.2	23	147	41	1.74	99	1720	64
4.5	85.2	13	143	26	46.4	37	24.8	55	57.4	77	0.12	100	70.8	99
5.0	58.3	39	80.4	58	22.2	69	8.05	85	9.91	96	0.05	100	55.6	99
6.0	3.20	97	3.29	98	0.77	99	0.25	100	0.56	100	0.03	100	1.21	100
Neut-3	La		Ce		Nd		Y		U		Th		Fe	
pH	mg/L	% ppt	mg/L	% ppt	mg/L	% ppt	mg/L	% ppt	mg/L	% ppt	mg/L	% ppt	mg/L	% ppt
Feed	111	--	221	--	83.5	--	62.2	--	282	--	268	--	5460	--
2.0	69.0	20	110	36	39.8	38	57.2	-19	277	-27	252	-22	4560	-8
2.5	68.7	20	109	36	39.4	39	56.7	-18	274	-26	252	-22	4500	-7
3.0	62.8	24	94.5	43	33.6	46	56.2	-21	270	-28	148	26	2320	43
3.5	57.4	32	81.2	52	28.4	55	55.7	-18	259	-21	58.4	71	2050	51
4.0	46.1	45	59.9	64	19.9	68	46.0	2	164	23	2.90	99	1850	55
4.5	22.0	74	24.0	86	7.18	89	15.5	67	27.6	87	0.15	100	1240	70
5.0	4.37	95	3.60	98	0.84	99	1.62	96	2.16	99	0.04	100	509	87
6.0	0.12	100	0.10	100	< 0.06	100	0.04	100	2.28	99	< 0.03	100	4.05	100

[1] Yttrium concentration calculated based on feed and residue values

Table 16 Combined PLS Composition

Sample ID	COMBINED PLS	Sample ID	COMBINED PLS
U mg/L	282	Ag mg/L	0.2
Th mg/L	268	Al mg/L	3430
La mg/L	111	As mg/L	3
Ce mg/L	221	Ba mg/L	0.25
Pr mg/L	24.3	Be mg/L	0.1
Nd mg/L	83.5	Bi mg/L	1
Sm mg/L	18	Ca mg/L	822
Eu mg/L	1.05	Cd mg/L	0.09
Gd mg/L	18	Co mg/L	16.7
Tb mg/L	2.81	Cr mg/L	35.4
Dy mg/L	15.2	Cu mg/L	39.6
Ho mg/L	2.52	Fe mg/L	5460
Er mg/L	6.28	Fe2+ mg/L	2260
Tm mg/L	0.81	K mg/L	1320
Yb mg/L	4.66	Li mg/L	6
Lu mg/L	0.61	Mg mg/L	241
Sc mg/L	0.73	Mn mg/L	97.2
Y mg/L	62.2	Mo mg/L	0.7
		Na mg/L	113
		Ni mg/L	30
		P mg/L	152
		Pb mg/L	6
		Sb mg/L	1
		Se mg/L	3
		Sn mg/L	2
		Sr mg/L	1.941
		Ti mg/L	51.4
		Tl mg/L	3
		V mg/L	2.1
		W mg/L	2
		Zn mg/L	14.8

The data for Neut-2 and Neut-3 are plotted in Figure 16 and Figure 17, respectively. The precipitation data was calculated using a sodium tie and the feed solution and filtrate assays. This calculation leads to some negative precipitation values due to the use of calculated concentrations of the elements under investigation. In order to compose a complete mass balance, additional testing without intermediate sampling at pH intervals will be required, which would be able to fully define the extent of precipitation at the target pH. The precipitation extents could be derived based on both solids and solution assays in such a test.

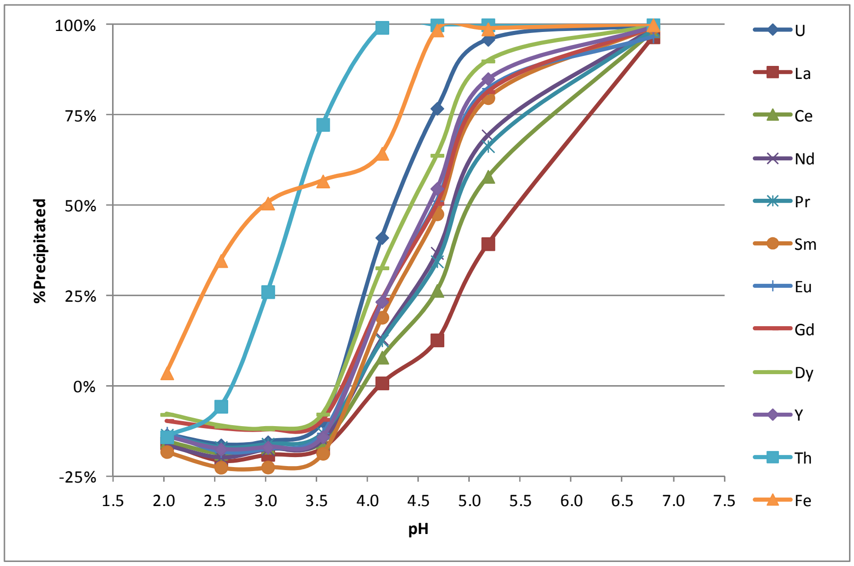


Figure 16 Neut2 Precipitation Curves (MgO)

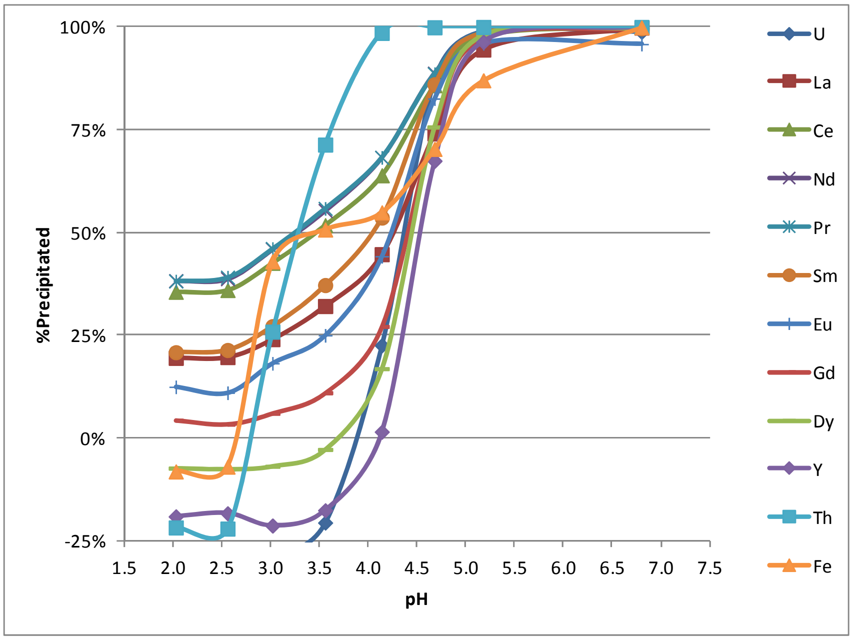


Figure 17 Neut3 Precipitation Curves (limestone)

As expected, a much greater amount of limestone (as in Neut-3) is required to achieve the same pH target when compared to using magnesium oxide (as in Neut-2) as the pH modifier with all other conditions remaining the same. Limestone resulted in greater losses of many elements of interest at all comparable pH points during the tests. Magnesium oxide proved to be more effective at iron and thorium removal with 57% of the iron and 72% of the thorium precipitated at pH 3.5 with limited losses of the other elements being tracked. At the same pH, limestone precipitated similar amounts of iron and thorium but also resulted in losses up to 55% of the REE.

The pH 2.0 neutralized PLS (Neut-1) was used for uranium ion-exchange testing. The PLS was contacted at various resin to solution ratios using Ambersep 920U SO4 strong base anion exchange resin. Results for the uranium ion exchange test are presented in Table 17.

Table 17 Uranium Ion Exchange Assay Results

	pH	U, mg/L	Fe, mg/L
Feed	2.15	281	4630
Contact 1	2.19	188	3750
Contact 2	2.20	122	3780
Contact 3	2.14	82	3820
Contact 4	2.14	47	3590
Contact 5	1.92	14	3980

Uranium in solution was steadily reduced from 281 mg/L in the feed to 14 mg/L in the barren solution after five contacts with fresh Ambersep 920U resin in sulphate form for 95% extraction in five stages. Iron was also reduced slightly from 4630 mg/L to 3980 mg/L implying some loading of anion ferric sulphate species. The pH remained fairly stable around pH 2. A uranium loading isotherm is shown in Figure 18, the results show that uranium may be extracted effectively from the PLS using ion-exchange. Additional testing will be required to optimize the process and select the most ideal resin.

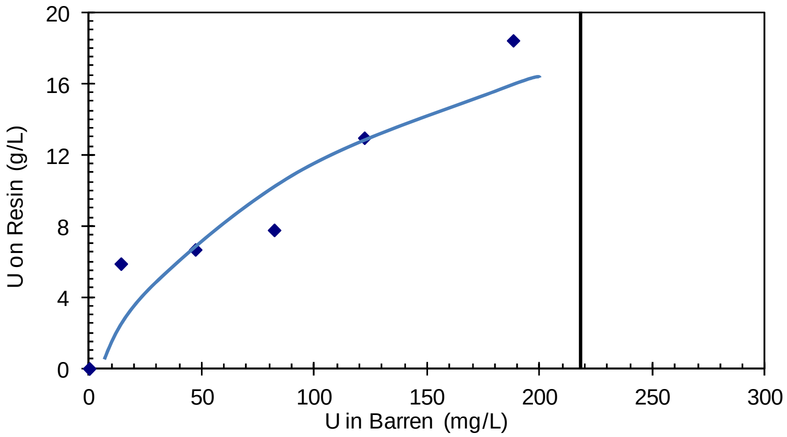


Figure 18 UIX Isotherm – Ambersep 920U

The UIX barren solution was then contacted with OC1026 resin (a DEHPA impregnated resin manufactured by Lewatit) to extract REE. Resin to solution ratios of 120, 60 and 20 mL/L were used in two-hour contacts on a vibratory shaker. The resin tended to float and stick to the sides of the vessels and thus generally had poor solution to resin contact. The test results are listed in Table 18 and plotted in Figure 19. Generally poor loading of the LREE was observed, but significant loading of HREE was measured. As expected, an increase in the resin:liquor ratio resulted in increased loading. Additional tests will be required to optimize IX extraction of REE from the PLS.

Table 18 REEIX Test Results

	Assays, mg/L				% Loading		
Element	Feed	120 mL/L	60 mL/L	20 mL/L	120 mL/L	60 mL/L	20 mL/L
TREE	336	239	261	308	29%	22%	8%
LREE	236	224	226	227	5%	4%	4%
HREE	100	15	34	81	85%	66%	19%
La	64.9	63.3	64.5	63.8	2%	1%	2%
Ce	107	103	102	102	4%	5%	5%
Pr	11.3	10.6	10.5	10.7	6%	7%	5%
Nd	42.4	38.9	40.3	40.5	8%	5%	4%
Sm	10.7	8.51	9.13	9.79	20%	15%	9%
Eu	0.68	0.48	0.55	0.62	29%	19%	9%
Gd	13.2	7.89	10.5	12.5	40%	20%	5%
Tb	2.34	0.65	1.42	2.03	72%	39%	13%
Dy	12.2	1.87	5.72	10.7	85%	53%	12%
Ho	2.25	0.2	0.79	1.88	91%	65%	16%
Er	5.48	0.21	1.08	4.21	96%	80%	23%
Tm	0.72	<0.04	0.05	0.43	>94%	93%	40%
Yb	4.3	0.03	0.13	1.74	99%	97%	60%
Lu	0.54	<0.03	<0.03	0.18	>94%	>94%	67%
Y	58.4	3.33	14.1	46.9	94%	76%	20%
Sc	0.54	<0.07	<0.07	<0.07	>87%	>87%	>87%
U	14.6	0.27	1.29	9.52	98%	91%	35%
Th	107	2.72	8.92	60.7	97%	92%	43%
Fe	2020	1700	1930	2090	16%	4%	-3%
Fe^{2+}	1900	1540	1690	1700			
Al	2690	2400	2520	2640	11%	6%	2%

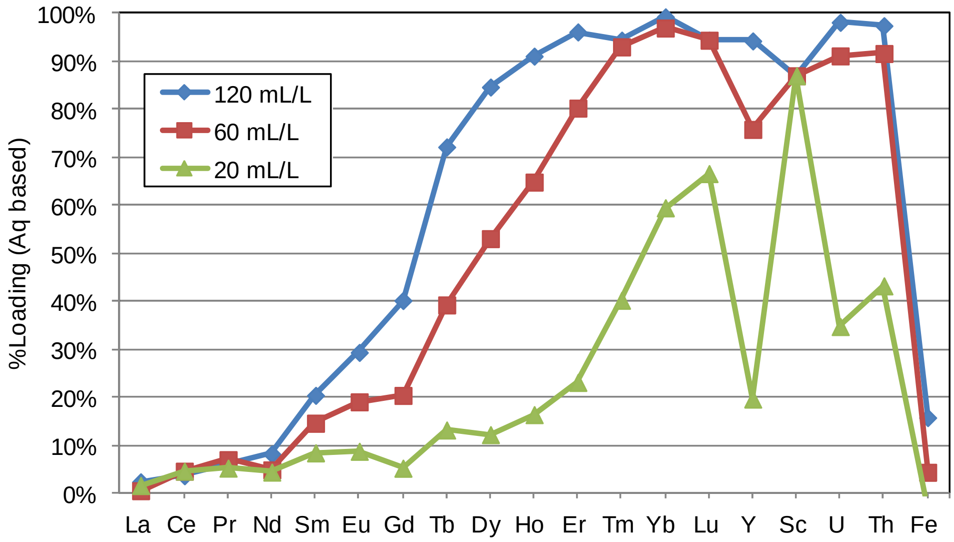


Figure 19 REE Loading on OC1026

Conclusions and Recommendations

Whole Ore Leaching

Two whole ore leach tests were conducted on ore ground to 80% passing 212 µm (test AL-1) and 75 µm (test AL-2). In both cases the leach was performed at 80ºC with 50 g/L free acid and over 72 hours with kinetic sampling. The test results indicate that LREE precipitation occurs with increased leach time. A possible mechanism is the formation of insoluble LREE double sulphates caused by the entry of K and Na into the system by the dissolution of K-feldspar and other minerals. An attempt to convert the hypothetical double sulphates to a soluble form failed and more testing is indicated. Since U leaching seemed to be essentially complete (~95% extraction) after 48 hours, such leach times, or shorter, would be appropriate for the maximization of LREE recovery. The acid consumptions for these tests were 46 kg/t and 35 kg/t for the coarse and fine grinds, respectively. LREE extraction was 14% and 16% for AL-1 and AL-2, respectively. HREE extraction was 71% and 74% for AL-1 and AL-2, respectively.

Flotation Concentrate Leaching

Several different methods of processing the flotation concentrate were investigated including simple atmospheric acid leaching, pressure oxidation in which the pyrite was oxidized to generate lixiviant solution and simultaneously promote metal dissolution, and sulphuric acid baking – a standard way of cracking monazite/bastnaesite minerals.

Flotation concentrate was subjected to a 50 g/L sulphuric acid leach procedure for 48 h at 80ºC with kinetic sampling. Not too surprisingly, the U and REE extraction data were very similar to those for the whole ore leaching with about 30% extraction for La and Ce and about 80% for Y and related HREE.

A sample of flotation concentrate was mixed as a paste with sulphuric acid and the resulting pellets held for 24 hours at a nominal 80ºC with moist air passing through the pellet bed. After 24 hours the ore was slurried with water, filtered and the products assayed. Data showed less than 30% U extraction, very low LREE extraction and about 30% HREE extraction.

A single pressure oxidation (POX) test on a flotation concentrate containing 814 g/t of U, 5877 g/t of TREE, and 9.2% S was performed. Test conditions included pre-acidification to pH 1.8 (39 kg/t acid added), a temperature of 210ºC and an oxygen overpressure of ~700 kPa (100 psi). The test was extended over four hours although full oxidation of pyrite and uranium extraction was expected in a shorter period of time. It was speculated that some cracking of refractory REE minerals might also occur given adequate time. In this test, 92% of the sulphide was oxidized, 98% of the U solubilized, along with 28% of the TREE, and 70% of the HREE. Y extraction was 80% reflecting the close association between U mineralization and Y. Clearly monazite was not cracked during POX.

Several acid bake tests were performed on Appia flotation concentrates. Initial tests indicated that a temperature of 200ºC and an acid addition of 1 t/t concentrate, and a three hour retention time was satisfactory with an acid consumption of 600 kg/t, U extraction of 98%, HREE extraction of 92%, and LREE extraction ranging from 56% for La to 90% for Sm. A pre-leach and acid bake process, simulating a counter-current leach system, was executed in tests AL-7 and AB-6 on a concentrate assaying 780 g/t U and 13636 g/t TREE. The flotation concentrate yielded 10.6% of the feed mass containing 92% of the S, 31% of the U, 79% of the LREE and 50% of the HREE. The pre-leach required 74 kg/t of acid (105 kg/t acid added). The acid bake on the pre-leach residue was done with a 600 kg/t acid addition and at a temperature of 250ºC for three hours. The subsequent water leach contained 133 kg/t of free acid which should, in a counter-current system, be more than sufficient for the pre-leach meaning that the fresh acid demand for the pre-leach and bake system is about 600 kg/t of concentrate.

Flotation Tailings Leaching

Samples of the flotation tailings corresponding to the concentrate used in the AL-7/AB-6 pre-leach and acid bake test were leached for 24 h at both room temperature and 50ºC in dilute sulphuric acid in tests AL-9 and AL-10. In both cases, the data showed metal extraction was substantially complete after just 12 h. Acid consumption was about 15-20 kg/t. The higher temperature leach offered 5% better U extraction (89% vs. 94%) and an average of 9% greater REE extraction (77% Y extraction at room temperature and 89% at 50ºC; and 44% Nd extraction at room temperature and 51% at 50ºC).

U and REE Recovery from Solution

A combined PLS was neutralized using both limestone and MgO. Limestone led to significant losses of REE as compared to MgO with similar levels of Fe (and Al, Th etc) removal. PLS was neutralized to pH 2.0 and then contacted with a strong base anion exchange resin to extract uranium and this led to 95% extraction of uranium in five successive contacts with fresh resin in each contact. The UIX barren was further neutralized to pH 3.0 and then contacted with a DEHPA impregnated resin to extract REE. LREE were not extracted to a high degree but HREE extraction was near 85%. Further IX work is needed to define conditions, probably counter-current extraction, where IX for all REE would be effective. Alternative resins should also be examined.

Recommendations

Additional testing should focus on the preleach and acid bake leach flowsheet on the flotation concentrate in order to further optimize the process and confirm the acid requirements. Additionally, testing to optimize the atmospheric leach of the flotation tailings is recommended. An additional test program examining uranium and REE recovery also needs be done to further optimize that part of the process and identify the most logical processing scheme (i.e. a combined resin-in-pulp recovery, separate resin-in-pulp recovery or liquid solid separation followed by SX recovery). Solid-liquid separation and rheology testing should also be done on relevant process streams – depending on the selected flowsheet – to provide related process design criteria.

The goal of the additional work will be to further define and optimize the pre-leach, bake, and tailings leach processes and identify an effective method for the recovery of U and REE from the leached slurry.

Appendix A – Leach Test Details

Atmospheric Leach Tests

Acid Bake Water Leach Tests

Roast / Leach Tests

Conc2 Pre-leach Acid Bake Tests

Project No.:	**13849-001**	**Project Technologist:**	Krystal Davis
Client:	**Appia Energy Corp**	**Date of Test:**	18-Dec-12

Test ID: **AL-2** **1kg**

Objective: To conduct a sulphuric acid leach test on a uranium ore
To study the kinetics of uranium dissolution under various conditions

Feed: Master Comp
Grind: 120 Stage 50% solids in lab ball mill
(note time/kg, weight charged, % solids in mill and mill size/type)

Procedure:

1. The sample was ground to the desired P80 and the mill discharge pan filtered
2. A small sample was cut for %H2O determination (~50 g)
3. The wet wt. of feed was recorded along with the kettle tare weight, slurry the solids and agitate kettle
4. The Slurry was brought to desired temperature (if required)
5. The pH or FA was adjusted to target level with H_2SO_4; all acid additions were recorded (beware of foaming, add acid sl
7. Small amounts of 200 g/L NaClO3 were added to achieve ORP target
8. Records of pH, emf, Temp were kept throughout the test, H_2SO_4 was added to keep constant pH (with acid requiremer
9. The reactor contents were sampled at 12, 24, 48 hrs using 60 mL syringe
10. The pulp sample weight was recorded, then filtered
11. Solids were washed with 60 mL pH 2.0 water, then 60 mL DI water (DO NOT LET CAKE CRACK or repulp)
12. The filtrate weight and SG was recorded, along w/ pH, FA and ORP if sufficient volume permitted
13. Dry solids and filtrate were submitted for analysis, the wash solution was rejected
14. After completion of the test the reactor was allowed to cool (if at T) and agitation was stopped
15. The final weight of vessel was recorded and the contents filtered and washed once with 300 mL pH 2.0 water, 3 times
16. The filtration rate was recorded along with notes on residue and liquor colour
17. The filtrate weight and SG was recorded, along w/ pH, FA and ORP
18. Dry solids and filtrate were submitted for analysis, the wash solution was rejected

All pulp samples must be labeled "NORM" for Naturally Occurring Radioactive Material

Conditions:

eactor Tare (incl. lid and baffles):	1753.41	g		
Wet Feed Weight:	1251.0	g		
Moisture:	20%			
Dry Feed Weight:	1000.8	g		
Target Slurry Density:	60	w/w%		
Water added:	417.0	mL		
Water in Feed:	250.2	mL		
Target FA:	50	g/L H_2SO_4		
Oxidant Type:	200 g/L NaClO3			
Target ORP:	500	mV *(vs. Ag|AgCl sat'd KCl electrode)*		
Fe3+	2	g/L		
Grind	75	µm	P80	41 µm
Temperature:	80	ºC		

Analytical Requirements:

Kinetic solutions	REE, Fe, Fe2+
Kinetic residues	REE
Final PLS	REE, Fe2+, ICP
Final residue	REE

Test Details:

Elapsed Time		Temp	pH	ORP	FA	Reagents Added		Comments
						H_2SO_4	200 g/L NaClO3	
(hh:mm)	(hrs)	(°C)		(mV)	(g/L H_2SO_4)	(g)	(mL)	
9:45	0.0	22.6	8.01	34		33.46		6.42g FeSO4 added
9:55		34.6	0.40	542				began heating to 80C
10:20		80.0	0.24	607				time 0
10:47		86.5	0.23	540				
14:07		80.2	0.14	576	31.86	14.01		FA check
14:50	5.1	80.1	-0.17	586				
16:16		82.8	-0.19	576				
22:27		81.0	-0.09	552	49.81	0.28		12h smp
7:50	22.1	77.7	-0.10	477	49.71	0.95		put sparger back, FA check
8:12		80.3	0.02	543				
10:20		82.8	-0.02	529	50.99			24h smp
15:32								25.4g DI added, stopped test
15:37		66.1	-0.11	479				restarted test, heating
20:51	35.1	35.6	-0.06	421				power tripped, heating mantle
21:04		70.7	0.00	398			2.00	heating, added chlorate
21:06		81.4	0.01	459				sparger and probes removed
8:09		80.0	0.02	520				
9:15		77.2	-0.03	528				FA Check, 155.2g DI added
9:39		58.2	0.04	528	71.1			heating
10:00	48.3	83.0	0.19	536	45.3	2.90		48h smp
14:00		79.1	-0.03	484				
7:30		76.8	0.72	504				277.4g DI added
8:21		83.7	0.56	402	29.6	13.00		FA check
8:30		81.9	0.10	428				
10:22	72.0	74.5	0.28	431				heating mantle quit. End test
TWA						64.6	2	mL or g
						62.0	0.4	kg/t

Project No.: **13849-001** **Project Technologist:** Krystal Davis
Client: **Appia Energy Corp** **Date of Test:** 18-Dec-12

Test ID: **AL-2** **1kg**

Objective: To conduct a sulphuric acid leach test on a uranium ore
To study the kinetics of uranium dissolution under various conditions

	Pulp Weight	**PLS Filtrate**	**Wash Filtrate**	**Wet Residue**	**Dry Residue**	**Pulp Density**	**Cake Moisture**	**PLS Filtrate**			
								pH	ORP¹	SG	Free Acid
Sample	g	g	g	(g)	(g)	w/w	(%)		(mV)		(g/L)
12 hr	74.2	18.5	180.01	55.5	50.7	68.2%	9			1.091	50
24 hr	98.0	24.3	172.8	75.5	66.2	67.5%	13			1.103	51
48 hr	102.0	24.2	117.1	72.3	69.2	67.8%	5			1.110	45
72 hr	108.5	16.4	152.3	79.8	71.8	66.2%	11			1.137	61
Final	1260.19	312.12	522.42	878.4	825.5	65.5%	7	0.04	446	1.113	68
							Avg.	**0.0**	**0.0**		**52.2**

¹(ORP measured against Ag|AgCl sat'd KCl electrode)
² Assuming SG of pore water in washed cake is 1.00.

Notes:

Filtration:

		enter filter paper type, size	Filtrate Colour:	green
10:37-12:49	132 min	enter final filtration time (time to no liquid on cake)	Residue Colour:	grey

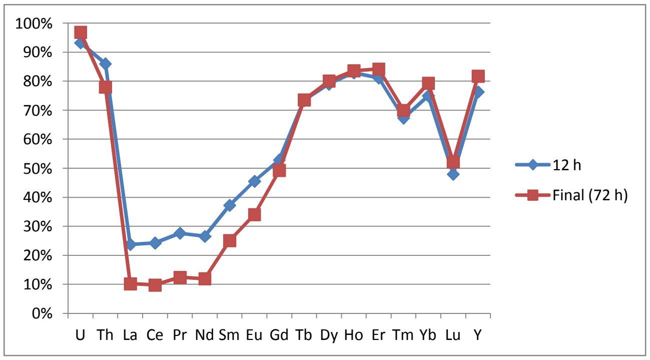
Project No.: **13849-001**
Client: **Appia Energy Corp**
Project Technologist: Krystal Davis
Date of Test: 18-Dec-12

Test **AL-2** **1000.8 g**

Metallurgical Balance

	Assays												Balance OUT/IN
	Head, g/t	PLS, mg/L					Residue, g/t						
		12 h	24 h	48 h	72 h	Final	12 h	24 h	48 h	72 h	Final	Calc'd Head	%
Amt, mL or g	1001	22	29	30	32	391	50.7	66.2	69.2	71.8	825.5	1083.4	
U	270	493	511	510	496	519	15.2	10.7	9.5	8.3	7.8	248	92%
Th	354	621	645	626	377	405	43.1	39.5	39.8	57.4	54.1	256	72%
Fe	19234	7120	7560	8660	8140	8650	--	--	--	--	20100	19256	100%
Fe++	--	4200	4810	4470	5160	5550	--	--	--	--	--	--	--
La	407	249	262	239	65.1	76.7	341	312	310	317	320	368	90%
Ce	767	459	490	445	109	131	612	555	569	577	574	659	86%
Pr	81	49.6	53.1	48.4	12.4	15.0	55.5	50.3	49.5	51.4	50.1	60	74%
Nd	254	171	180	171	46.7	55.8	202	182	185	192	195	228	90%
Sm	42.6	36.6	38.0	37.0	14.2	16.4	26.3	23.5	23.0	22.7	23.2	32.4	76%
Eu	1.9	1.96	2.04	1.99	0.97	1.09	1.0	0.9	0.8	0.7	1.0	1.5	81%
Gd	26.1	33.4	34.2	34.5	20.9	23.0	12.7	11.8	11.8	12.7	11.2	22.9	88%
Tb	3.3	5.17	5.34	5.29	3.77	4.12	0.8	0.7	0.7	0.7	0.7	2.7	82%
Dy	16.5	26.5	27.9	28.1	22.0	23.8	3.0	3.1	3.4	3.1	2.8	14.2	86%
Ho	2.6	4.56	4.77	4.77	4.02	4.32	0.4	0.4	0.4	0.4	0.4	2.4	93%
Er	6.7	11.1	11.8	11.8	10.5	11.3	1.1	1.0	1.0	0.9	1.0	6.2	93%
Tm	0.8	1.45	1.53	1.53	1.39	1.48	<0.3	<0.3	<0.3	<0.3	<0.3	1.0	123%
Yb	5.2	8.43	8.89	9.00	8.34	8.91	1.2	0.9	1.0	0.9	1.1	5.2	100%
Lu	0.7	1.08	1.13	1.16	1.08	1.15	<0.5	<0.5	<0.5	<0.5	<0.5	1.0	147%
Y	62	106	111	111	98.7	104	14	14	15	12	11	60	97%
Na	816	87	99	276	282	300	--	--	--	--	1020	906	111%
K	27400	1860	2560	3360	4560	4800	--	--	--	--	25400	21418	78%

Element / Sample	Extraction, %				
	12 h	24 h	48 h	72 h	Final
U	93%	95%	96%	96%	97%
Th	86%	88%	87%	75%	78%
La	24%	27%	25%	8%	10%
Ce	24%	28%	25%	8%	10%
Pr	28%	31%	29%	10%	12%
Nd	27%	30%	28%	10%	12%
Sm	37%	41%	41%	22%	25%
Eu	46%	50%	52%	38%	34%
Gd	53%	56%	56%	43%	49%
Tb	73%	77%	76%	71%	74%
Dy	79%	80%	78%	76%	80%
Ho	83%	84%	84%	82%	84%
Er	81%	84%	83%	84%	84%
Tm	67%	69%	69%	68%	70%
Yb	75%	81%	79%	81%	79%
Lu	48%	50%	50%	49%	52%
Y	76%	78%	76%	79%	82%
Na	4%	4%	11%	11%	12%
K	3%	4%	6%	8%	8%



Acid Consumption:

Acid In:	62.0	g
Free Acid Out:	26.6	g
Acid Consumption:	35.4	kg H2SO4 / t Ore

Extractions (PLS vs. Calc'd H

TREE:	16%
LREE:	11%
HREE:	74%

Weight Loss: -8%

Project No.:	**13849-001**	**Project Technologist:**	Krystal Davis
Client:	**Appia Energy Corp**	**Date of Test:**	18-Dec-12

Test ID: **AL-1** **1kg**

Objective: To conduct a sulphuric acid leach test on a uranium ore
To study the kinetics of uranium dissolution under various conditions

Feed: Master Comp
Grind: 30 50% solids in lab ball mill
(note time/kg, weight charged, % solids in mill and mill size/type)

Procedure:

1. The sample was ground to the desired P80 and the mill discharge pan filtered
2. A small sample was cut for %H2O determination (~50 g)
3. The wet wt. of feed was recorded along with the kettle tare weight, slurry the solids and agitate kettle
4. The Slurry was brought to desired temperature (if required)
5. The pH or FA was adjusted to target level with H_2SO_4; all acid additions were recorded (beware of foaming, add acid sl
7. Small amounts of 200 g/L NaClO3 were added to achieve ORP target
8. Records of pH, emf, Temp were kept throughout the test, H_2SO_4 was added to keep constant pH (with acid requiremer
9. The reactor contents were sampled at 12, 24, 48 hrs using 60 mL syringe
10. The pulp sample weight was recorded, then filtered
11. Solids were washed with 60 mL pH 2.0 water, then 60 mL DI water (DO NOT LET CAKE CRACK or repulp)
12. The filtrate weight and SG was recorded, along w/ pH, FA and ORP if sufficient volume permitted
13. Dry solids and filtrate were submitted for analysis, the wash solution was rejected
14. After completion of the test the reactor was allowed to cool (if at T) and agitation was stopped
15. The final weight of vessel was recorded and the contents filtered and washed once with 300 mL pH 2.0 water, 3 times
16. The filtration rate was recorded along with notes on residue and liquor colour
17. The filtrate weight and SG was recorded, along w/ pH, FA and ORP
18. Dry solids and filtrate were submitted for analysis, the wash solution was rejected

All pulp samples must be labeled "NORM" for Naturally Occurring Radioactive Material

Conditions:

eactor Tare (incl. lid and baffles):	1694.6	g
Wet Feed Weight:	1285.6	g
Moisture:	22%	
Dry Feed Weight:	1002.8	g
Target Slurry Density:	60	w/w%
Water added:	385.7	mL
Water in Feed:	282.8	mL
Target FA:	50	g/L H_2SO_4
Oxidant Type:	200 g/L NaClO3	
Target ORP:	500	mV *(vs. Ag|AgCl sat'd KCl electrode)*
Fe3+	2	g/L
Grind	212	μm
Temperature:	80	ºC

P80 152 μm

Analytical Requirements:

Kinetic solutions	REE, Fe, Fe2+
Kinetic residues	REE
Final PLS	REE, Fe2+, ICP
Final residue	REE

Test Details:

Elapsed Time		**Temp**	**pH**	**ORP**	**FA**	**Reagents Added**		**Comments**
						H_2SO_4	**200 g/L NaClO3**	
(hh:mm)	**(hrs)**	**(°C)**		**(mV)**	**(g/L H_2SO_4)**	**(g)**	**(mL)**	
12:00	0.0	17.9	6.35	35		35.16		6.43g FeSO4
12:05		26.1	0.47	517				heating
14:07		76.5	0.48	505	19.61	20.54		FA Check
14:50		79.3	-0.06	530				
16:16		81.8	-0.09	523				
22:27	10.5	78.7	-0.18	505	46.58	3.56		12h smp
7:50		79.2	0.05	486	55.2			put sparger back, FA Check
8:12		80.1	-0.04	498				
12:11	24.2	79.2	-0.02	492	49.61			
15:18								71.5g DI added, stopped test
15:30		63.8	-0.02	475				started test, heating
20:51		25.6	-0.14	444				power tripped, heating mantle
21:04		35.2	-0.09	405			2.00	heating, added chlorate
21:15	33.3	86.7	0.05	479			1.00	sparger and probes removed
8:09		81.3	0.27	453				
9:15		79.1	0.12	475	46.18	2.80		FA check, 30 g DI added
9:51		70.1	-0.10	433			1.00	heating back up
12:00	48.0	78.6	-0.16	466				48h smp
14:00		82.0	-0.29	525				87g DI added
7:30		77.1	-1.65	349				347.6g DI added
8:12		88.4	0.47	411	32.6	10.10		FA check
8:30		87.5	0.11	380				
10:19		88.1	0.03	358				
12:05	72.0	84.3	0.04	361	48.25			end test
TWA						72.2	4	mL or g
						69.1	0.8	kg/t

Project No.: **13849-001** **Project Technologist:** Krystal Davis
Client: **Appia Energy Corp** **Date of Test:** 18-Dec-12

Test ID: **AL-1** **1kg**

Objective: To conduct a sulphuric acid leach test on a uranium ore
To study the kinetics of uranium dissolution under various conditions

	Pulp Weight	**PLS Filtrate**	**Wash Filtrate**	**Wet Residue**	**Dry Residue**	**Pulp Density**	**Cake Moisture**	**PLS Filtrate**			
								pH	ORP[1]	SG	Free Acid
Sample	g	g	g	(g)	(g)	w/w	(%)		(mV)		(g/L)
12 hr	87.1	19.8	175.43	63.6	53.4	61.4%	18	🟩	🟩	1.116	47
24 hr	84.0	19.7	182.43	67.0	51.6	61.4%	25	🟩	🟩	1.089	50
48 hr	86.7	11.7	159.6	76.5	57.5	66.3%	27	🟩	🟩	1.146	66
72 hr	103.0	25.4	157.1	68.5	59.1	57.4%	15	🟩	🟩	1.100	48
Final	1313.6	361.3	597.9	943.7	767.5	58.4%	20	-0.03	433	1.097	47
							Avg.	**0.0**	**0.0**		**54.2**

[1] (ORP measured against Ag|AgCl sat'd KCl electrode)
[2] Assuming SG of pore water in washed cake is 1.00.

Notes:

Filtration: Whatman #3, 185mm — enter filter paper type, size — Filtrate Colour: green
>120 min — enter final filtration time (time to no liquid on cake) — Residue Colour: grey

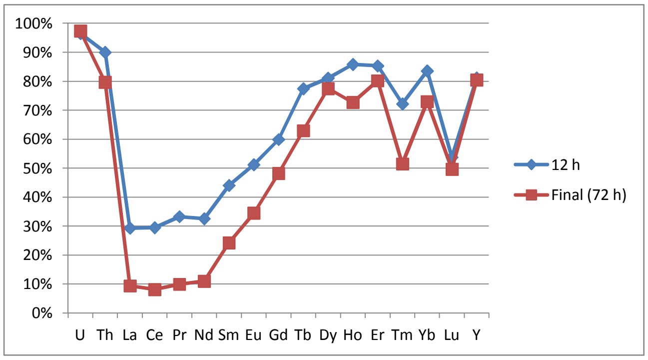
Project No.: **13849-001**
Client: **Appia Energy Corp**
Project Technologist: Krystal Davis
Date of Test: 18-Dec-12

Test **AL-1** **1002.8 g**

Metallurgical Balance

	Assays												Balance OUT/IN
	Head, g/t	PLS, mg/L					Residue, g/t					Calc'd Head	%
		12 h	24 h	48 h	72 h	Final	12 h	24 h	48 h	72 h	Final		
Amt, mL or g	1003	30	30	25	40	498	53.4	51.6	57.5	59.1	767.5	989.1	
U	270	463	485	679	380	365	9.5	8.3	7.4	6.7	6.5	252	93%
Th	354	576	605	815	315	305	36.2	35.9	37.5	69.3	50.5	272	77%
Fe	19234	13300	14600	13400	12200	12100	--	--	--	--	20100	23369	121%
Fe++	--	9800	10600	10700	9930	9610	--	--	--	--	--	--	--
La	407	240	257	319	56.5	55.3	327	322	320	332	348	396	97%
Ce	767	435	476	588	87.0	86.6	588	576	581	602	639	717	94%
Pr	81	46.8	51.1	63.1	9.84	9.74	53.0	53.0	52.4	55.3	57.5	66	82%
Nd	254	159	174	219	38.3	38.8	186	187	186	192	204	237	93%
Sm	42.6	33.9	36.9	47.8	12.7	12.0	24.3	24.0	22.8	25.2	24.4	34	80%
Eu	1.9	1.86	1.97	2.59	0.82	0.81	1.0	0.9	0.9	1.3	1.0	1.6	86%
Gd	26.1	32.0	33.5	44.1	17.9	16.9	12.1	10.8	10.9	12.4	11.8	24.1	92%
Tb	3.3	4.86	5.06	6.92	2.95	2.88	0.8	0.7	3.8	1.6	1.1	3.3	100%
Dy	16.5	24.3	26.9	36.4	16.5	15.9	3.2	3.0	5.6	4.1	3.0	14.4	87%
Ho	2.6	4.29	4.52	6.22	2.95	2.87	0.4	0.4	3.2	1.5	0.7	2.9	110%
Er	6.7	10.3	11.2	15.1	7.71	7.48	1.0	0.9	3.4	1.5	1.2	6.4	96%
Tm	0.8	1.38	1.45	2.03	1.01	0.98	<0.3	<0.3	2.5	1.1	0.6	1.4	173%
Yb	5.2	8.11	8.54	11.8	6.09	5.83	0.9	1.0	3.2	1.4	1.4	5.4	105%
Lu	0.7	1.03	1.09	1.54	0.78	0.76	<0.5	<0.5	<0.5	0.5	<0.5	1.0	145%
Y	62	99.2	106	146	72.4	69.9	13	13	13	12	11	59	96%
Na	820	103	110	210	309	295	--	--	--	--	1010	957	117%
K	27400	1240	2200	3040	3880	3740	--	--	--	--	23600	20534	75%

Element / Sample	Extraction, %				
	12 h	24 h	48 h	72 h	Final
U	96%	97%	98%	97%	97%
Th	90%	91%	91%	75%	80%
La	29%	32%	31%	10%	9%
Ce	29%	32%	31%	9%	8%
Pr	33%	36%	35%	11%	10%
Nd	33%	35%	34%	12%	11%
Sm	44%	47%	48%	25%	24%
Eu	51%	56%	56%	30%	34%
Gd	60%	64%	64%	49%	48%
Tb	77%	81%	45%	55%	63%
Dy	81%	84%	74%	73%	77%
Ho	86%	87%	46%	57%	73%
Er	85%	88%	66%	78%	80%
Tm	72%	74%	26%	38%	51%
Yb	84%	83%	62%	75%	73%
Lu	54%	56%	58%	51%	50%
Y	81%	82%	83%	80%	80%
Na	6%	6%	11%	17%	16%
K	3%	5%	8%	10%	9%



Acid Consumption:

Acid In:	69.3	g
Free Acid Out:	23.6	g
Acid Consumption:	45.6	kg H2SO4 / t Ore

Extractions (PLS vs. Calc'd H

TREE:	14%
LREE:	9%
HREE:	71%

Weight Loss: 1%

Project No.: **13849-001**
Client: **Appia Energy Corp**
Project Technologist: Krystal Davis
Date of Test: 22-Jan-13

Test ID: **AL1RL** **100 g**

Objective: To conduct a sulphuric acid leach test on a uranium ore
To study the kinetics of uranium dissolution under various conditions

Feed: AL1 Residue
Grind: #N/A 50% solids in lab ball mill
(note time/kg, weight charged, % solids in mill and mill size/type)

Procedure:

1. 100 g of AL1 Residue was pulp in 40 g/L NaOH at 40% solids.
2. The Slurry was brought to 70ºC.
3. Records of pH, emf, Temp were kept throughout the test. NaOH was added to maintain pH > 9.
4. After 1 hour the slurry was filtered and washed with 3 x 100 mL of DI water.
5. The wet filtercake was then added to 1 N HCl (36.5 g/L = 101 g of 36% HCl in 1 L solution)
6. The slurry was agitated for 2 h at room temperature
7. The final slurry was weighed, filtered and PLS retreived, volume measured and sent for assay.

Conditions:

eactor Tare (incl. lid and baffles):	1028.83	g
Wet Feed Weight:	100.0	g
Moisture:		
Dry Feed Weight:	100.0	g
NaOH Contact Density:	40.0	w/w%
40 g/L NaOH Added:	150	g
Temp. of NaOH Conctact:	70	ºC
HCl Leach Density:	40	w/w%
36.5 g/L HCl Added:	150	g

Analytical Requirements:
HCl PLS - REE Scan

Test Details:

					Reagents Added		
Elapsed Time (hh:mm) (hrs)	**Temp (°C)**	**pH**	**ORP (mV)**		**40 g/L NaOH (g)**	**1 N HCl (g)**	**Comments**
NaOH Contact	**70.0**	**>9**					
11:07	24.9	11.91	1069		20.00	-	Start Heating
11:18	73.0	10.61	1126		-	-	t = 0
11:33	74.0	10.80	1232		-	-	
11:48	72.4	10.81	1166		-	-	
12:00	73.3	10.78	1167		-	-	
12:18	69.1	10.85	1154		-	-	End test
HCl Leach	**RT**						
13:36	22.8	0.70	671		-	101.00	Hold for 2h
14:05	23.1	0.57	622		-	-	
14:34	23.2	0.32	668		-	-	
15:00	23.3	0.20	667		-	-	
15:30	23.4	0.11	663		-	-	End test
					20.0	101	mL or g 36.4606 g/L
					192.0	36.8	kg/t

Project No.: **13849-001** **Project Technologist:** Krystal Davis
Client: **Appia Energy Corp** **Date of Test:** 22-Jan-13

Test ID: **AL1RL** **100 g**

Objective: To conduct a sulphuric acid leach test on a uranium ore
To study the kinetics of uranium dissolution under various conditions

Sample	Pulp Weight	PLS Filtrate	Wash Filtrate	Wet Residue	Dry Residue	Pulp Density	Cake Moisture	PLS Filtrate			
								pH	ORP¹	SG	Free Acid
	g	g	g	(g)	(g)	w/w	(%)		(mV)		(g/L)
HCl Slurry	266.3	140.9	164.9	199.8	104.0	39.1%	48.0	0.49	663	1.016	30

[1](ORP measured against Ag|AgCl sat'd KCl electrode)
[2] Assuming SG of pore water in washed cake is 1.00.

Notes:

Filtration: enter filter paper type, size Filtrate Colour:
N/A min enter final filtration time (time to no liquid on cake) Residue Colour:

Project No.: **13849-001** **Project Technologist:** Krystal Davis
Client: **Appia Energy Corp** **Date of Test:** 22-Jan-13

Test **AL-1RL** **100 g**

Metallurgical Balance

	Assays			
	Head, g/t	Final PLS	Final Res	%Releached
Amt, mL or g	100	164	104	
U	6.5	0.73	6.8	18%
Th	50.5	11.8	33.6	38%
Fe	20100	--		
Fe++	--	--		
La	348	14.5	299	7%
Ce	639	19.3	539	5%
Pr	57.5	1.74	56.1	5%
Nd	204	5.52	176	4%
Sm	24.4	0.94	25.3	6%
Eu	1.0	0.05	1.1	8%
Gd	11.8	0.36	12.1	5%
Tb	1.1	0.05	1.1	7%
Dy	3.0	0.19	3.2	10%
Ho	0.7	0.02	0.7	5%
Er	1.2	0.05	1.2	7%
Tm	0.6	<0.04	0.4	11%
Yb	1.4	0.03	1	4%
Lu	<0.5	<0.03	<0.5	10%
Y	11	0.49	12	7%
Na	1010	--		
K	23600	--		

Acid Consumption:

Acid In:	3.7	g
Free Acid Out:	4.8	g
Acid Consumption:	-11.0	kg H2SO4 / t Ore

Extractions (PLS vs. Calc'd Hd):

TREE:	3%
LREE:	3%
HREE:	76%

Weight Loss: -4%

Project No.: **13849-001** **Project Technologist:** Krystal Davis
Client: **Appia Energy Corp** **Date of Test:** 26-Feb-13

Test ID: **AL-4**

Objective: To conduct a sulphuric acid leach test on a uranium ore
To study the kinetics of uranium dissolution under various conditions

Feed: Tails 1
Grind: -- 50% solids in lab ball mill
(note time/kg, weight charged, % solids in mill and mill size/type)

Procedure:

1. The sample was ground to the desired P80 and the mill discharge pan filtered
2. A small sample was cut for %H2O determination (~50 g)
3. The wet wt. of feed was recorded along with the kettle tare weight, slurry the solids and agitate kettle
4. The Slurry was brought to desired temperature (if required)
5. The pH or FA was adjusted to target level with H_2SO_4; all acid additions were recorded (beware of foaming, add acid slowly)
7. Small amounts of 200 g/L NaClO3 were added to achieve ORP target
8. Records of pH, emf, Temp were kept throughout the test, H_2SO_4 was added to keep constant pH (with acid requirement recorded)
9. The reactor contents were sampled at 12, 24, 48 hrs using 60 mL syringe
10. The pulp sample weight was recorded, then filtered
11. Solids were washed with 60 mL pH 2.0 water, then 60 mL DI water (DO NOT LET CAKE CRACK or repulp)
12. The filtrate weight and SG was recorded, along w/ pH, FA and ORP if sufficient volume permitted
13. Dry solids and filtrate were submitted for analysis, the wash solution was rejected
14. After completion of the test the reactor was allowed to cool (if at T) and agitation was stopped
15. The final weight of vessel was recorded and the contents filtered and washed once with 300 mL pH 2.0 water, 3 times with 300 mL DI
16. The filtration rate was recorded along with notes on residue and liquor colour
17. The filtrate weight and SG was recorded, along w/ pH, FA and ORP
18. Dry solids and filtrate were submitted for analysis, the wash solution was rejected

All pulp samples must be labeled "NORM" for Naturally Occurring Radioactive Material

Conditions:

Reactor Tare (incl. lid and baffles):	1245.1	g	
Wet Feed Weight:	400.0	g	
Moisture:	9%		
Dry Feed Weight:	364.0	g	
Target Slurry Density:	60	w/w%	
Water added:	258.3	mL	added too much water
Water in Feed:	36.0	mL	
Target FA:	20	g/L H_2SO_4	
Oxidant Type:	200 g/L NaClO3		
Target ORP:	500	mV *(vs. Ag|AgCl sat'd KCl electrode)*	
Fe3+	2	g/L	
Grind	~95	µm	
Temperature:	50	ºC	

Analytical Requirements:

Kinetic solutions	REE, Fe, Fe2+, SO4
Kinetic residues	REE, S(tot)
Final PLS	REE, Fe2+, ICP, SO4
Final residue	REE, Fe, Na, K, S(tot)

Test Details:

Elapsed Time		Temp	pH	ORP	FA	Reagents Added		Comments
						H_2SO_4	200 g/L NaClO3	
(hh:mm)	(hrs)	(°C)		(mV)	(g/L H_2SO_4)	(g)	(mL)	
9:40		24.2	0.93	437		6.10		2.89g FeSO4
9:49	0.0	53.4	1.15	504				time 0
10:06		53.8	1.13	507				water added, 30.6g
10:16		63.8	1.13	502				transferred to 1L kettle
11:04		51.3	1.09	491				
11:52	2.0	42.3	1.14	514	10.1	4.40		
12:10		59.5	0.81	512				
13:43		54.4	0.74	509				
14:45		51.0	0.73	509				
15:50		51.3	0.73	508				
20:00	10.0	49.0	0.65	505	27.16			10h sample
20:14		50.6	0.64	508				
8:00		50.9	6.23	512				pH meter recalibrated
9:49	24.0	49.9	0.85	507				added 40.9g DI
11:15		52.4	0.71	512				
21:23		51.8	0.60	510				
7:36		52.2	0.55	510				
9:49	48.0	52.1	0.55	510				
TWA						10.5	0	mL or g
						27.7	0.0	kg/t

Project No.: **13849-001**
Client: **Appia Energy Corp**
Project Technologist: Krystal Davis
Date of Test: 26-Feb-13

Test ID: **AL-4**

Objective: To conduct a sulphuric acid leach test on a uranium ore
To study the kinetics of uranium dissolution under various conditions

	Pulp Weight	**PLS Filtrate**	**Wash Filtrate**	**Wet Residue**	**Dry Residue**	**Pulp Density**	**Cake Moisture**	**PLS Filtrate**			
								pH	ORP[1]	SG	Free Acid
Sample	g	g	g	(g)	(g)	w/w	(%)		(mV)		(g/L)
12 hr	69.6	20.1	199.5	43.7	39.7	57.0%	9	🟩	🟩	1.035	27
24 hr	76.0	20.3	147.7	54.9	50.4	66.3%	9	🟩	🟩	1.042	33
Final	409.0	106.6	465.3	298.2	281.2	68.7%	6	1.8	420	1.050	37
							TWA:	**1.8**	**420**		**33.2**

[1] (ORP measured against Ag|AgCl sat'd KCl electrode)
[2] Assuming SG of pore water in washed cake is 1.00.

Notes:

* sub sample dried for assay 49g wet, 46.2 g dry

Filtration:

Whatman #3, 150mm enter filter paper type, size
10 min enter final filtration time (time to no liquid on cake)

Filtrate Colour: lt yellow
Residue Colour: lt grey

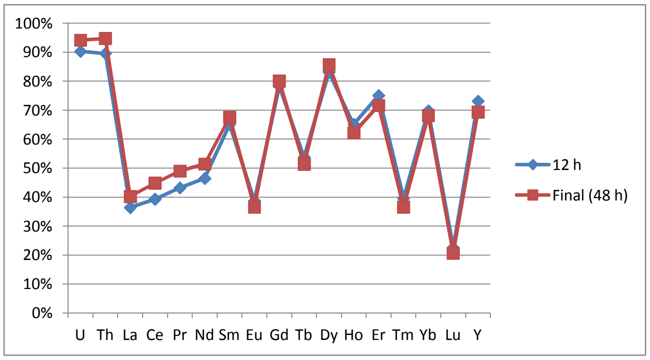
Project No.: **13849-001** **Project Technologist:** Krystal Davis
Client: **Appia Energy Corp** **Date of Test:** 26-Feb-13

Test **AL-4** **364 g**

Metallurgical Balance

	Assays								Balance
		PLS, mg/L			Residue, g/t				OUT/IN
	Head, g/t	12 h	24 h	Final	12 h	24 h	Final	Calc'd Head	%
Amt, mL or g	364	29	25	122	39.7	50.4	281.2	371.3	
U	86.4	127	148	176	9.9	6.9	4.7	83	96%
Th	81.7	103	118	142	8.7	4.3	3.4	66	81%
Fe	2870	4540	5580	6710	--	--	2200	4589	160%
Fe++	--	2400	2900	3500	--	--	--	--	--
La	232	17.3	19.8	28.0	22	15	18	30	13%
Ce	371	34.7	40.2	56.3	39	26	30	54	15%
Pr	35.5	3.97	4.62	6.21	3.8	2.6	2.8	6	16%
Nd	131	14.3	16.6	22.0	12	8	9	19	14%
Sm	18.5	4.35	4.97	6.26	1.7	1.3	1.3	4	22%
Eu	3.6	0.26	0.30	0.40	<0.3	<0.3	<0.3	0.5	13%
Gd	12.0	4.46	5.20	6.50	0.9	0.8	0.7	4	30%
Tb	1.1	0.80	0.98	1.22	<0.5	<0.5	<0.5	1.0	93%
Dy	6.6	4.76	5.54	6.92	0.7	0.7	0.5	4	54%
Ho	0.7	0.77	0.95	1.14	<0.3	<0.3	<0.3	0.8	114%
Er	2.4	2.07	2.38	2.92	<0.5	<0.5	<0.5	1.8	74%
Tm	0.4	0.27	0.32	0.40	<0.3	<0.3	<0.3	0.5	118%
Yb	1.8	1.59	1.93	2.47	0.5	0.6	<0.5	1.6	87%
Lu	<0.5	0.20	0.24	0.30	<0.5	<0.5	<0.5	0.6	126%
Y	14	18.7	21.3	26.2	<10	<10	<10	16.5	118%
S/SO_4	200	41000	49000	59000	400	400	500	204	102%
Na	890	--	--	191	--	--	1100	896	101%
K	23800	--	--	764	--	--	23700	18199	76%

Element / Sample	Extraction, % 12 h	24 h	Final
U	90%	91%	94%
Th	90%	93%	95%
La	36%	39%	40%
Ce	39%	43%	45%
Pr	43%	46%	49%
Nd	46%	50%	51%
Sm	65%	65%	68%
Eu	>39%	>33%	>37%
Gd	78%	76%	80%
Tb	>54%	>49%	>51%
Dy	83%	79%	86%
Ho	>65%	>61%	>62%
Er	>75%	>70%	>72%
Tm	>40%	>34%	>37%
Yb	70%	61%	>68%
Lu	>23%	>19%	>21%
Y	>73%	>67%	>69%
S	96%	95%	94%
Na	--	--	7%
K	--	--	1%



Acid Consumption:

Acid In:	10.1	g
Free Acid Out:	4.5	g
Acid Consumption:	15.5	kg H2SO4 / t Ore

Extractions (PLS vs. Calc'd Hd):

TREE:	51%
LREE:	46%
HREE:	70%

Weight Loss: -2%

Project No.:	**13849-001**	**Project Technologist:**	Krystal Davis
Client:	**Appia Energy Corp**	**Date of Test:**	26-Feb-13

Test ID: **AL-5**

Objective: To conduct a sulphuric acid leach test on a uranium ore
To study the kinetics of uranium dissolution under various conditions

Feed: Tails 1
Grind: -- 50% solids in lab ball mill
(note time/kg, weight charged, % solids in mill and mill size/type)

Procedure:

1. The sample was ground to the desired P80 and the mill discharge pan filtered
2. A small sample was cut for %H2O determination (~50 g)
3. The wet wt. of feed was recorded along with the kettle tare weight, slurry the solids and agitate kettle
4. The Slurry was brought to desired temperature (if required)
5. The pH or FA was adjusted to target level with H_2SO_4; all acid additions were recorded (beware of foaming, add acid slowly)
7. Small amounts of 200 g/L NaClO3 were added to achieve ORP target
8. Records of pH, emf, Temp were kept throughout the test, H_2SO_4 was added to keep constant pH (with acid requirement recorded)
9. The reactor contents were sampled at 12, 24, 48 hrs using 60 mL syringe
10. The pulp sample weight was recorded, then filtered
11. Solids were washed with 60 mL pH 2.0 water, then 60 mL DI water (DO NOT LET CAKE CRACK or repulp)
12. The filtrate weight and SG was recorded, along w/ pH, FA and ORP if sufficient volume permitted
13. Dry solids and filtrate were submitted for analysis, the wash solution was rejected
14. After completion of the test the reactor was allowed to cool (if at T) and agitation was stopped
15. The final weight of vessel was recorded and the contents filtered and washed once with 300 mL pH 2.0 water, 3 times with 300 mL DI
16. The filtration rate was recorded along with notes on residue and liquor colour
17. The filtrate weight and SG was recorded, along w/ pH, FA and ORP
18. Dry solids and filtrate were submitted for analysis, the wash solution was rejected

All pulp samples must be labeled "NORM" for Naturally Occurring Radioactive Material

Conditions:

Reactor Tare (incl. lid and baffles):	1418.5	g
Wet Feed Weight:	400.0	g
Moisture:	9%	
Dry Feed Weight:	364.0	g
Target Slurry Density:	60	w/w%
Water added:	258.3	mL
Water in Feed:	36.0	mL
Target FA:	20	g/L H_2SO_4
Oxidant Type:	200 g/L NaClO3	
Target ORP:	500	mV *(vs. Ag|AgCl sat'd KCl electrode)*
Fe3+	2	g/L
Grind	~95	µm
Temperature:	ambient	ºC

Analytical Requirements:

Kinetic solutions	REE, Fe, Fe2+, SO4
Kinetic residues	REE, S(tot)
Final PLS	REE, Fe2+, ICP, SO4
Final residue	REE, Fe, Na, K, S(tot)

Test Details:

Elapsed Time		Temp	pH	ORP	FA	Reagents Added		Comments
						H_2SO_4	200 g/L NaClO3	
(hh:mm)	(hrs)	(°C)		(mV)	(g/L H_2SO_4)	(g)	(mL)	
9:42	0.0	23.7	0.98	339		6.10		2.89g FeSO4 added
10:07		23.9	0.83	473				
10:16		23.2	0.84	473				Transferred to a 1L kettle
11:08		23.4	0.80	474				
11:52	2.0	23.5	0.78	474	8.72	3.40		
12:10		24.3	0.60	469				
13:46		24.3	0.45	475			0.60	
14:40		24.4	0.43	489			0.20	
15:52		24.0	0.43	503				
20:00	10.0	24.0	0.43	519	19.51			10h smp
8:00		23.5	0.44	519				
9:42	24.0	23.9	0.45	528		0.80		24h smp
11:15		24.3	0.42	528				
21:23		24.8	0.39	533				
7:36		24.4	0.46	528				
9:42	48.0	24.6	0.47	528				48h smp
TWA						10.3	0.8	mL or g
						27.2	0.4	kg/t

Project No.: **13849-001**
Client: **Appia Energy Corp**
Project Technologist: Krystal Davis
Date of Test: 26-Feb-13

Test ID: **AL-5**

Objective: To conduct a sulphuric acid leach test on a uranium ore
To study the kinetics of uranium dissolution under various conditions

	Pulp Weight	**PLS Filtrate**	**Wash Filtrate**	**Wet Residue**	**Dry Residue**	**Pulp Density**	**Cake Moisture**	**PLS Filtrate**			
								pH	ORP¹	SG	Free Acid
Sample	g	g	g	(g)	(g)	w/w	(%)		(mV)		(g/L)
12 hr	64.0	21.8	196.6	39.5	35.7	55.8%	10	🟩	🟩	1.027	20
24 hr	56.4	23.2	184.7	32.8	29.6	52.5%	10	🟩	🟩	1.024	18
Final	611.2	259.1	413.4	330.7	309.9	50.7%	6.4	1.26	496	1.024	17
							TWA:	**1.3**	**496.0**		**18.0**

[1] (ORP measured against Ag|AgCl sat'd KCl electrode)
[2] Assuming SG of pore water in washed cake is 1.00.

Notes:
Calculated values based on % moisture.
* sub sample submitted for assay 54.2g wet, 29.6 g dry

Filtration:
Whatman #3, 150mm — enter filter paper type, size
10 min — enter final filtration time (time to no liquid on cake)
Filtrate Colour: lt yellow
Residue Colour: lt grey

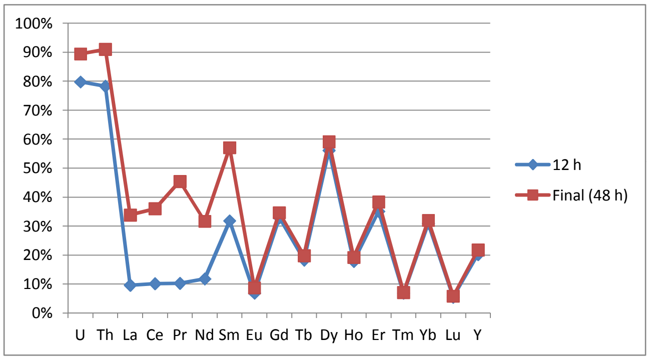
Project No.: **13849-001** **Project Technologist:** Krystal Davis
Client: **Appia Energy Corp** **Date of Test:** 26-Feb-13

Test **AL-5** **364 g**

Metallurgical Balance

	Assays								Balance
	Head, g/t	PLS, mg/L			Residue, g/t			Calc'd Head	OUT/IN
		12 h	24 h	Final	12 h	24 h	Final		%
Amt, mL or g	364	28	26	294	35.7	29.6	309.9	375.2	
U	86.4	92.0	84.9	80.3	18	12	9	86	99%
Th	81.7	74.7	69.0	64.0	16	7	6	68	83%
Fe	2870	3890	3270	2970	--	--	2200	4661	162%
Fe++	--	454	412	394	--	--	--	--	--
La	232	11.4	10.5	11.3	83	<20	21	37	16%
Ce	371	22.9	21.1	22.5	157	35	38	70	19%
Pr	35.5	2.66	2.46	2.63	18	3	3	7	19%
Nd	131	9.68	9.03	9.76	56	<20	<20	32	25%
Sm	18.5	3.02	2.80	2.80	5	<2	<2	5	26%
Eu	3.6	0.19	0.17	0.20	<2	<2	<2	2.2	61%
Gd	12.0	3.17	2.84	2.79	<5	<5	<5	8	63%
Tb	1.1	0.58	0.54	0.52	<2	<2	<2	2.5	226%
Dy	6.6	3.32	3.16	3.04	<2	<2	<2	5	73%
Ho	0.7	0.56	0.50	0.50	<2	<2	<2	2.5	353%
Er	2.4	1.40	1.31	1.31	<2	<2	<2	3.2	134%
Tm	0.4	0.19	0.17	0.16	<2	<2	<2	2.2	538%
Yb	1.8	1.16	1.06	0.99	<2	<2	<2	2.9	163%
Lu	<0.5	0.15	0.13	0.13	<2	<2	<2	2.1	424%
Y	14	13.1	12.1	11.7	<40	<40	<40	51.0	364%
S/SO_4	200	33000	28000	29000	400	400	400	837.3	419%
Na	890	--	--	159	--	--	1100	1033	116%
K	23800	--	--	139	--	--	23500	19519	82%

Element /	Extraction, %		
Sample	12 h	24 h	Final
U	80%	86%	89%
Th	78%	90%	91%
La	10%	32%	34%
Ce	10%	35%	36%
Pr	10%	42%	45%
Nd	12%	>29%	>32%
Sm	32%	>55%	>57%
Eu	>7%	>7%	>9%
Gd	>33%	>33%	>35%
Tb	>18%	>19%	>20%
Dy	>56%	>58%	>59%
Ho	>18%	>18%	>19%
Er	>35%	>37%	>38%
Tm	>7%	>7%	>7%
Yb	>31%	>32%	>32%
Lu	>5%	>5%	>6%
Y	20%	21%	22%
S	96%	95%	96%
Na	--	--	12%
K	--	--	1%



Acid Consumption:

Acid In:	9.9	g
Free Acid Out:	5.0	g
Acid Consumption:	13.4	kg H2SO4 / t Ore

Extractions (PLS vs. Calc'd Hd):

TREE:	32%
LREE:	36%
HREE:	25%

Weight Loss: -3%

Project No.:	**13849-001**	**Project Technologist:**	Krystal Davis
Client:	**Appia Energy Corp**	**Date of Test:**	17-Apr-13

Test ID: **AL-9** **500g**

Objective: To conduct a sulphuric acid leach test on a uranium ore
To study the kinetics of uranium dissolution under various conditions

Feed: Tails 2
Grind: N/A 50% solids in lab ball mill
(note time/kg, weight charged, % solids in mill and mill size/type)

Procedure:

1. The sample was ground to the desired P80 and the mill discharge pan filtered
2. A small sample was cut for %H2O determination (~50 g)
3. The wet wt. of feed was recorded along with the kettle tare weight, slurry the solids and agitate kettle
4. The Slurry was brought to desired temperature (if required)
5. The pH or FA was adjusted to target level with H_2SO_4; all acid additions were recorded (beware of foaming, add acid slowly)
7. Small amounts of 200 g/L NaClO3 were added to achieve ORP target
8. Records of pH, emf, Temp were kept throughout the test, H_2SO_4 was added to keep constant pH (with acid requirement recorded)
9. The reactor contents were sampled at 4, 12, 24 hrs using 60 mL syringe
10. The pulp sample weight was recorded, then filtered
11. Solids were washed with 60 mL pH 2.0 water, then 60 mL DI water (DO NOT LET CAKE CRACK or repulp)
12. The filtrate weight and SG was recorded, along w/ pH, FA and ORP if sufficient volume permitted
13. Dry solids and filtrate were submitted for analysis, the wash solution was rejected
14. After completion of the test the reactor was allowed to cool (if at T) and agitation was stopped
15. The final weight of vessel was recorded and the contents filtered and washed once with 300 mL pH 2.0 water, 3 times with 300 mL DI
16. The filtration rate was recorded along with notes on residue and liquor colour
17. The filtrate weight and SG was recorded, along w/ pH, FA and ORP
18. Dry solids and filtrate were submitted for analysis, the wash solution was rejected

All pulp samples must be labeled "NORM" for Naturally Occurring Radioactive Material

Conditions:

Reactor Tare (incl. lid and baffles):	1878.7	g
Wet Feed Weight:	650.3	g
Moisture:	23%	
Dry Feed Weight:	500.7	g
Target Slurry Density:	50	w/w%
Water added:	351.2	mL
Water in Feed:	149.6	mL
Target FA:	20	g/L H_2SO_4
Oxidant Type:	200 g/L NaClO3	
Target ORP:	500	mV *(vs. Ag|AgCl sat'd KCl electrode)*
Fe3+		g/L
Grind	~95	µm
Temperature:	ambient	ºC

Analytical Requirements:

Kinetic solutions	REE, Fe, Fe2+
Kinetic residues	REE
Final PLS	REE, Fe2+, ICP
Final residue	REE

Test Details:

Elapsed Time (hh:mm)	(hrs)	Temp (°C)	pH	ORP (mV)	FA (g/L H_2SO_4)	Reagents Added H_2SO_4 (g)	Reagents Added 200 g/L NaClO3 (mL)	Comments
8:00		20.2	7.75	-31		11.50		
8:25	0.0	22.7	1.06	471				Time 0
10:10		22.3	0.81	468	14.7	3.00		FA check
10:26	2.0	23.4	0.74	467			0.80	
11:18		23.2	0.71	486			0.70	
12:33	4.0	23.1	0.72	511	16.9	1.90		4h sample
14:55		22.8	0.70	532	15	2.70		FA check
15:02		23.0	0.67	531				
20:25	12.0	22.1	0.67	550	20.2			12h sample
8:21	24.0	21.9	0.69	554				end test, 24h sample
Avg.		**22.8**	**0.76**	**502**		19.1	1.5	mL or g
						36.6	0.6	kg/t

Sampling Data:

Sample	Pulp Weight g	PLS Filtrate g	Wash Filtrate g	Wet Residue (g)	Dry Residue (g)	Pulp Density w/w	Cake Moistur (%)	PLS Filtrate pH	PLS Filtrate ORP¹ (mV)	PLS Filtrate SG	PLS Filtrate Free Acid (g/L)
4 hr	78.0	32.7	153.9	43.8	39.2	50.3%	11	🟩	🟩	1.023	17
12 hr	56.7	20.8	160	30.3	27.7	48.9%	9	🟩	🟩	1.033	20
Final	896.2	296.7	902.4	553.7	434.4	48.5%	22	0.73	533	1.028	29
							TWA:				**24.0**

[1](ORP measured against Ag|AgCl sat'd KCl electrode)

[2] Assuming SG of pore water in washed cake is 1.00.

Project No.: **13849-001**
Client: **Appia Energy Corp**

Project Technologist: Krystal Davis
Date of Test: 17-Apr-13

Test ID: **AL-9** **500g**

Objective: To conduct a sulphuric acid leach test on a uranium ore
To study the kinetics of uranium dissolution under various conditions

Feed: Tails 2

Notes:

Filtration:

Whatman #3 150mm		enter filter paper type, size	Filtrate Colour:	light yellow
15	min	enter final filtration time (time to no liquid on cake)	Residue Colour:	grey

Metallurgical Balance:

Sample	4 h PLS	12 h PLS	24 h PLS	4 h Res	12 h Res	24 h Res
mL or g	37.9	28.1	449.2	39.2	27.7	434.4
Assays, mg/L or g/t						
U	161	204	170	48	25	21
Th	144	181	151	49	21	18
La	21.1	26.4	22.8	53	51	50
Ce	42.5	53	45.1	87	83	83
Pr	4.9	6.04	5.16	9	8	8
Nd	17.3	22	18.8	26	25	25
Sm	5.12	6.39	5.54	5	3	3
Eu	0.34	0.44	0.37	<2	<2	<2
Gd	5.81	7.28	6.07	2	<2	<2
Tb	1.18	1.45	1.21	<2	<2	<2
Dy	5.86	7.43	6.22	3	<2	<2
Ho	1.03	1.34	1.11	<2	<2	<2
Er	2.4	3.29	2.7	<2	<2	<2
Tm	0.34	0.43	0.38	<2	<2	<2
Yb	1.98	2.5	2.12	<2	<2	<2
Lu	0.23	0.31	0.27	<2	<2	<2
Y	23	29.9	25.2	<10	<10	<10
Sc	0.15	0.23	0.21	<25	<25	<25
Fe	2970	3610	2810			
Fe^{2+}	180	182	224			

Extraction		
4 h	12 h	24 h
76%	89%	89%
74%	90%	90%
28%	34%	32%
32%	39%	36%
35%	43%	40%
39%	47%	44%
50%	68%	66%
		55%
74%		68%
		84%
65%		82%
		82%
		80%
		65%
		66%
		55%
		77%
		1%

based on PLS and Direct Head

Metal Units, mg	4 h PLS	12 h PLS	24 h PLS	4 h Res	12 h Res	24 h Res	Sum	Calc'd Hd, g/t	Direct Hd, g/t	Balance Out/IN	Units In
U	6.11	5.73	76.37	1.88	0.69	9.12	100	200	187	107%	93.6
Th	5.46	5.08	67.83	1.92	0.58	7.82	89	177	182	97%	91.1
La	0.80	0.74	10.24	2.08	1.41	21.72	37	74	70	106%	35.1
Ce	1.61	1.49	20.26	3.41	2.30	36.06	65	130	130	100%	65.1
Pr	0.19	0.17	2.32	0.35	0.22	3.48	7	13	13.4	100%	6.7
Nd	0.66	0.62	8.45	1.02	0.69	10.86	22	45	45	99%	22.5
Sm	0.19	0.18	2.49	0.20	0.08	1.30	4	9	9.9	90%	5.0
Eu	0.01	0.01	0.17	0.08	0.06	0.87	1	2	0.7	341%	0.4
Gd	0.22	0.20	2.73	0.08	0.06	0.87	4	8	9.3	89%	4.7
Tb	0.04	0.04	0.54	0.08	0.06	0.87	2	3	1.5	217%	0.8
Dy	0.22	0.21	2.79	0.12	0.06	0.87	4	9	7.9	108%	4.0
Ho	0.04	0.04	0.50	0.08	0.06	0.87	2	3	1.4	225%	0.7
Er	0.09	0.09	1.21	0.08	0.06	0.87	2	5	3.5	137%	1.8
Tm	0.01	0.01	0.17	0.08	0.06	0.87	1	2	0.6	399%	0.3
Yb	0.08	0.07	0.95	0.08	0.06	0.87	2	4	3.3	127%	1.7
Lu	0.01	0.01	0.12	0.08	0.06	0.87	1	2	0.5	456%	0.3
Y	0.87	0.84	11.32	0.39	0.28	4.34	18	36	34	106%	17.0
Sc	0.01	0.01	0.09	0.98	0.69	10.86	13	25	<25	101%	12.5

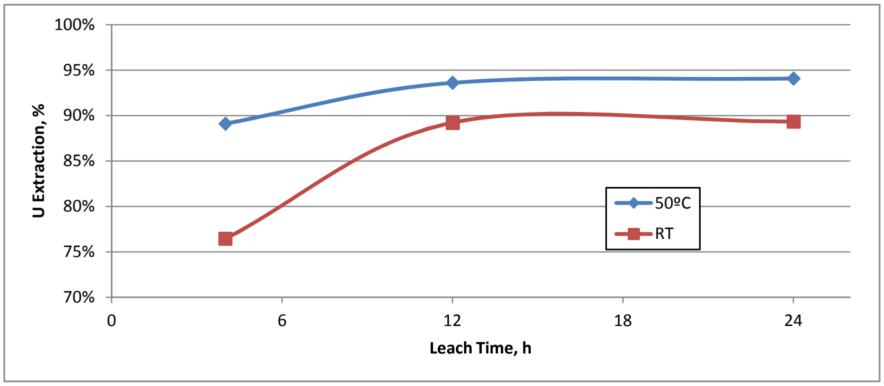
Project No.: **13849-001**
Client: **Appia Energy Corp**

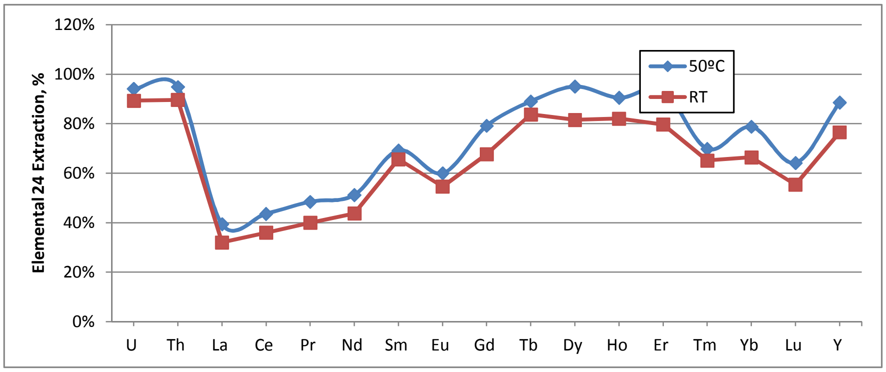
Project Technologist: Krystal Davis
Date of Test: 17-Apr-13

Test ID: **AL-9** **500g**

Objective: To conduct a sulphuric acid leach test on a uranium ore
To study the kinetics of uranium dissolution under various conditions

Feed: Tails 2
Trends:





Project No.:	**13849-001**	**Project Technologist:**	Krystal Davis
Client:	**Appia Energy Corp**	**Date of Test:**	17-Apr-13

Test ID: **AL-10** **500g**

Objective: To conduct a sulphuric acid leach test on a uranium ore
To study the kinetics of uranium dissolution under various conditions

Feed: Tails 2
Grind: N/A 50% solids in lab ball mill
(note time/kg, weight charged, % solids in mill and mill size/type)

Procedure:

1. The sample was ground to the desired P80 and the mill discharge pan filtered
2. A small sample was cut for %H2O determination (~50 g)
3. The wet wt. of feed was recorded along with the kettle tare weight, slurry the solids and agitate kettle
4. The Slurry was brought to desired temperature (if required)
5. The pH or FA was adjusted to target level with H_2SO_4; all acid additions were recorded (beware of foaming, add acid slowly)
7. Small amounts of 200 g/L NaClO3 were added to achieve ORP target
8. Records of pH, emf, Temp were kept throughout the test, H_2SO_4 was added to keep constant pH (with acid requirement recorded)
9. The reactor contents were sampled at 4, 12, 24 hrs using 60 mL syringe
10. The pulp sample weight was recorded, then filtered
11. Solids were washed with 60 mL pH 2.0 water, then 60 mL DI water (DO NOT LET CAKE CRACK or repulp)
12. The filtrate weight and SG was recorded, along w/ pH, FA and ORP if sufficient volume permitted
13. Dry solids and filtrate were submitted for analysis, the wash solution was rejected
14. After completion of the test the reactor was allowed to cool (if at T) and agitation was stopped
15. The final weight of vessel was recorded and the contents filtered and washed once with 300 mL pH 2.0 water, 3 times with 300 mL DI
16. The filtration rate was recorded along with notes on residue and liquor colour
17. The filtrate weight and SG was recorded, along w/ pH, FA and ORP
18. Dry solids and filtrate were submitted for analysis, the wash solution was rejected

All pulp samples must be labeled "NORM" for Naturally Occurring Radioactive Material

Conditions:

Reactor Tare (incl. lid and baffles):	1994.8	g
Wet Feed Weight:	650.8	g
Moisture:	23%	
Dry Feed Weight:	501.1	g
Target Slurry Density:	50	w/w%
Water added:	351.4	mL
Water in Feed:	149.7	mL
Target FA:	20	g/L H_2SO_4
Oxidant Type:	200 g/L NaClO3	
Target ORP:	500	mV *(vs. Ag|AgCl sat'd KCl electrode)*
Fe3+		g/L
Grind	~95	µm
Temperature:	50	ºC

Analytical Requirements:

Kinetic solutions	REE, Fe, Fe2+
Kinetic residues	REE
Final PLS	REE, Fe2+, ICP
Final residue	REE

Test Details:

Elapsed Time (hh:mm)	(hrs)	Temp (°C)	pH	ORP (mV)	FA (g/L H_2SO_4)	Reagents Added H_2SO_4 (g)	 200 g/L NaClO3 (mL)	Comments
8:09		22.4	7.91	-137		11.30		heating to 50 degrees
8:25		25.3	0.95	450				
8:38	0.0	50.0	0.93	499				Time 0
10:10		51.2	1.00	516	12.15	4.40		FA check
10:26	2.0	51.0	0.82	517				
12:35	4.0	50.1	0.74	520	20.8			4h sample
14:55		52.1	0.72	523	13.2	3.70		FA check
15:02		53.2	0.65	523				
20:38	12.0	52.6	0.54	525	18.9	0.30		12h sample
7:30		51.0	0.42	517				
8:30	24.0	51.8	0.47	522				end test, 24h sample
Avg.		**48.2**	**0.79**	**509**		19.7	0	mL or g
						37.7	0.0	kg/t

Sampling Data:

Sample	Pulp Weight g	PLS Filtrate g	Wash Filtrate g	Wet Residue (g)	Dry Residue (g)	Pulp Density w/w	Cake Moistur (%)	PLS Filtrate pH	 ORP¹ (mV)	 SG	 Free Acid (g/L)
4 hr	67.2	27.1	150.6	38.1	33.7	50.1%	12			1.025	21
12 hr	84.2	33.1	169.6	46.5	42.1	50.0%	10			1.032	19
Final	853.5	290.97	800.97	474.6	427.4	50.1%	10	0.71	458	1.028	19
							TWA:				**19.4**

¹(ORP measured against Ag|AgCl sat'd KCl electrode)
² Assuming SG of pore water in washed cake is 1.00.

Project No.: **13849-001**
Client: **Appia Energy Corp**

Project Technologist: Krystal Davis
Date of Test: 17-Apr-13

Test ID: **AL-10** **500g**

Objective: To conduct a sulphuric acid leach test on a uranium ore
To study the kinetics of uranium dissolution under various conditions

Feed: Tails 2

Notes:

Filtration: Whatman #3 150mm enter filter paper type, size
15 min enter final filtration time (time to no liquid on cake)

Filtrate Colour: light yellow
Residue Colour: grey

Metallurgical Balance:

Sample	4 h PLS	12 h PLS	24 h PLS	4 h Res	12 h Res	24 h Res
mL or g	32.7	40.8	414.5	33.7	42.1	427.4
Assays, mg/L or g/t						
U	194	212	197	23	14	12
Th	168	186	171	21	11	9
La	29.4	32.7	31.5	51	49	47
Ce	56.9	64	61.3	85	80	77
Pr	6.4	7.07	6.78	9	8	7
Nd	23.5	26.1	24.9	26	24	23
Sm	6.79	7.32	6.96	5	3	3
Eu	0.42	0.45	0.43	<2	<2	<2
Gd	7.17	7.89	7.56	3	<2	<2
Tb	1.3	1.44	1.37	<2	<2	<2
Dy	7.3	8.07	7.71	3	<2	<2
Ho	1.23	1.37	1.3	<2	<2	<2
Er	3.17	3.47	3.36	<2	<2	<2
Tm	0.41	0.45	0.43	<2	<2	<2
Yb	2.52	2.79	2.67	<2	<2	<2
Lu	0.31	0.34	0.33	<2	<2	<2
Y	30	33	30.8	<10	<10	<10
Sc	0.2	0.25	0.23	<25	<25	<25
Fe	3130	3250	3090			
Fe^{2+}	1890	2060	1950			

	Extraction	
4 h	12 h	24 h
89%	94%	94%
89%	94%	95%
36%	39%	39%
39%	44%	44%
41%	46%	48%
47%	51%	51%
57%	70%	69%
		60%
70%		79%
		89%
70%		95%
		91%
		93%
		70%
		79%
		64%
		89%
		1%

based on PLS and Direct Head

Metal Units, mg	4 h PLS	12 h PLS	24 h PLS	4 h Res	12 h Res	24 h Res	Sum	Calc'd Hd, g/t	Direct Hd, g/t	Balance Out/IN	Units In
U	6.34	8.65	81.66	0.78	0.59	5.13	103	206	187	110%	93.7
Th	5.49	7.59	70.88	0.71	0.46	3.85	89	178	182	98%	91.2
La	0.96	1.33	13.06	1.72	2.06	20.09	39	78	70	112%	35.1
Ce	1.86	2.61	25.41	2.86	3.37	32.91	69	138	130	106%	65.1
Pr	0.21	0.29	2.81	0.30	0.34	2.99	7	14	13.4	103%	6.7
Nd	0.77	1.06	10.32	0.88	1.01	9.83	24	48	45	106%	22.6
Sm	0.22	0.30	2.88	0.17	0.13	1.28	5	10	9.9	100%	5.0
Eu	0.01	0.02	0.18	0.07	0.08	0.85	1	2	0.7	347%	0.4
Gd	0.23	0.32	3.13	0.10	0.08	0.85	5	9	9.3	101%	4.7
Tb	0.04	0.06	0.57	0.07	0.08	0.85	2	3	1.5	223%	0.8
Dy	0.24	0.33	3.20	0.10	0.08	0.85	5	10	7.9	121%	4.0
Ho	0.04	0.06	0.54	0.07	0.08	0.85	2	3	1.4	234%	0.7
Er	0.10	0.14	1.39	0.07	0.08	0.85	3	5	3.5	151%	1.8
Tm	0.01	0.02	0.18	0.07	0.08	0.85	1	2	0.6	405%	0.3
Yb	0.08	0.11	1.11	0.07	0.08	0.85	2	5	3.3	140%	1.7
Lu	0.01	0.01	0.14	0.07	0.08	0.85	1	2	0.5	466%	0.3
Y	0.98	1.35	12.77	0.34	0.42	4.27	20	40	34	118%	17.0
Sc	0.01	0.01	0.10	0.84	1.05	10.69	13	25	<25	101%	12.5

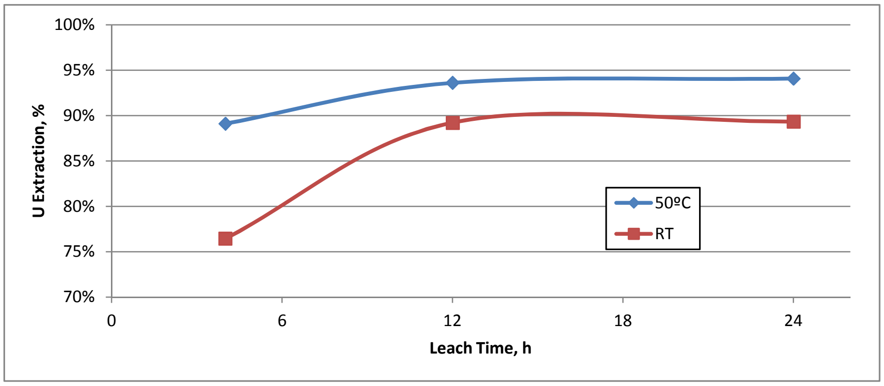
Project No.: **13849-001**
Client: **Appia Energy Corp**

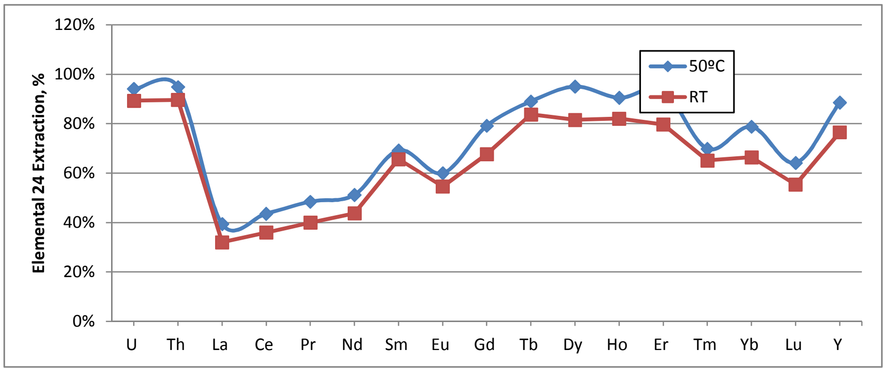
Project Technologist: Krystal Davis
Date of Test: 17-Apr-13

Test ID: **AL-10** **500g**

Objective: To conduct a sulphuric acid leach test on a uranium ore
To study the kinetics of uranium dissolution under various conditions

Feed: Tails 2
Trends:





Project No.:	**13849-001**	**Project Technologist:**	Krystal Davis
Client:	**Appia Energy Corp**	**Date of Test:**	27-Feb-13

Test ID: **AL-3** **1kg**

Objective: To conduct a sulphuric acid leach test on a uranium ore
To study the kinetics of uranium dissolution under various conditions

Feed: Conc 1
Grind: -- 50% solids in lab ball mill
(note time/kg, weight charged, % solids in mill and mill size/type)

Procedure:

1. The sample was ground to the desired P80 and the mill discharge pan filtered
2. A small sample was cut for %H2O determination (~50 g)
3. The wet wt. of feed was recorded along with the kettle tare weight, slurry the solids and agitate kettle
4. The Slurry was brought to desired temperature (if required)
5. The pH or FA was adjusted to target level with H_2SO_4; all acid additions were recorded (beware of foaming, add acid slowly)
7. Small amounts of 200 g/L NaClO3 were added to achieve ORP target
8. Records of pH, emf, Temp were kept throughout the test, H_2SO_4 was added to keep constant pH (with acid requirement recorded)
9. The reactor contents were sampled at 12, 24, 48 hrs using 60 mL syringe
10. The pulp sample weight was recorded, then filtered
11. Solids were washed with 60 mL pH 2.0 water, then 60 mL DI water (DO NOT LET CAKE CRACK or repulp)
12. The filtrate weight and SG was recorded, along w/ pH, FA and ORP if sufficient volume permitted
13. Dry solids and filtrate were submitted for analysis, the wash solution was rejected
14. After completion of the test the reactor was allowed to cool (if at T) and agitation was stopped
15. The final weight of vessel was recorded and the contents filtered and washed once with 300 mL pH 2.0 water, 3 times with 300 mL DI
16. The filtration rate was recorded along with notes on residue and liquor colour
17. The filtrate weight and SG was recorded, along w/ pH, FA and ORP
18. Dry solids and filtrate were submitted for analysis, the wash solution was rejected

All pulp samples must be labeled "NORM" for Naturally Occurring Radioactive Material

Conditions:

Reactor Tare (incl. lid and baffles):	1833.4	g	
Wet Feed Weight:	1100.0	g	
Moisture:	23%		
Dry Feed Weight:	849.2	g	
Target Slurry Density:	60	w/w%	
Water added:	315.3	mL	568.3
Water in Feed:	250.8	mL	
Target FA:	50	g/L H_2SO_4	
Oxidant Type:	200 g/L NaClO3		
Target ORP:	500	mV *(vs. Ag|AgCl sat'd KCl electrode)*	
Fe3+	2	g/L	
Grind	~95	µm	
Temperature:	80	ºC	

Analytical Requirements:

Kinetic solutions	REE, Fe, Fe2+, SO4
Kinetic residues	REE, S(tot)
Final PLS	REE, Fe2+, ICP, SO4
Final residue	REE, Fe, Na, K, S(tot)

Test Details:

Elapsed Time		Temp	pH	ORP	FA	Reagents Added		Comments
(hh:mm)	(hrs)	(°C)		(mV)	(g/L H_2SO_4)	H_2SO_4 (g)	200 g/L NaClO3 (mL)	
8:00	-	22.9	8.93	-397		48.90		Start Heating
8:27	0.0	80.9	2.52	-112				t = 0
10:50		79.3	2.44	268	0	10.00	2.00	
11:48		80.4	1.42	417	5.98	26.10	0.50	6.36g of FeSO4 added at 13:15
21:13	~12	83.2	0.60	498	20.04	24.00		12h Sample
21:28		83.0	0.21	530				
7:36		81.3	0.26	523				
8:27	24.0	81.3	0.25	525		8.60		24h Sample
12:52		81.1	0.21	533				
7:45		80.8	0.10	541	41.1	4.30		
8:10		82.4	0.14	548				
8:27	48.0	81.9	0.13	546				48h Sample, End Test
TWA						121.9	2.5	mL or g
						137.8	0.6	kg/t

Project No.: **13849-001**
Client: **Appia Energy Corp**
Project Technologist: Krystal Davis
Date of Test: 27-Feb-13

Test ID: **AL-3** **1kg**

Objective: To conduct a sulphuric acid leach test on a uranium ore
To study the kinetics of uranium dissolution under various conditions

	Pulp	PLS	Wash	Wet	Dry	Pulp	Cake	PLS Filtrate			
	Weight	Filtrate	Filtrate	Residue	Residue	Density	Moisture	pH	ORP[1]	SG	Free Acid
Sample	g	g	g	(g)	(g)	w/w	(%)		(mV)		(g/L)
12 hr	66.3	26.8	190.4	42.4	32.6	49.2%	25			1.100	20
24 hr	76.3	24.5	1130	53.2	36.8	48.2%	33			1.115	39
Final	1552.6	426.1	959.4	1021.1	789.2	50.8%	25	1.82	400	1.138	48
							TWA:	**1.8**	**400.0**		**38.9**

[2] Assuming SG of pore water in washed cake is 1.00.

Notes:

* A sub sample was cut for assay 70.9g wet, 54.8 g dried
Estimated values

Filtration:

Whatman 3, 185	enter filter paper type, size	Filtrate Colour:	yellow
overnight	enter final filtration time (time to no liquid on cake)	Residue Colour:	grey

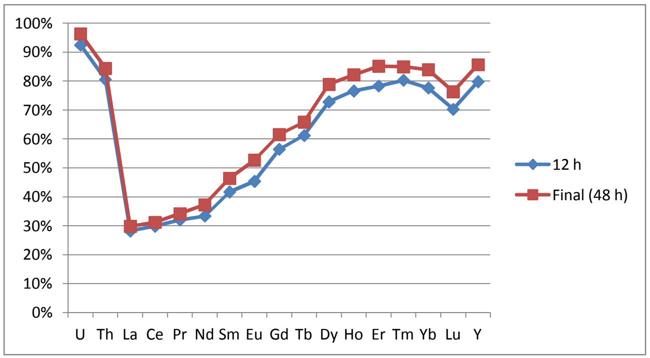
Project No.:	**13849-001**	**Project Technologist:**	Krystal Davis
Client:	**Appia Energy Corp**	**Date of Test:**	27-Feb-13

Test **AL-3** **849.2 g**

Metallurgical Balance

	Assays								Balance OUT/IN
	Head, g/t	PLS, mg/L			Residue, g/t			Calc'd Head	%
		12 h	24 h	Final	12 h	24 h	Final		
Amt, mL or g	849	31	35	671	32.6	36.8	789.2	858.6	
U	814	820	856	1000	62.4	39.8	32	880	108%
Th	1140	973	1030	1180	220	186	185	1186	104%
Fe	89500	17200	18400	21300	--	--	72100	84292	94%
Fe++	--	11600	11400	11400	--	--	--		
La	1550	451	466	500	1080	1020	1000	1430	92%
Ce	2580	842	889	923	1860	1780	1730	2525	98%
Pr	290	91.3	96.1	102	182	174	167	255	88%
Nd	903	311	337	368	584	551	527	843	93%
Sm	146	63.5	66.2	76.7	83.4	78.7	75.3	141	96%
Eu	6.4	3.27	3.46	4.06	3.7	3.2	3.1	6.6	102%
Gd	84.8	54.2	56.7	67.4	39.3	37	35.8	93	110%
Tb	12.0	8.23	8.66	10.2	4.9	4.6	4.5	13.1	110%
Dy	55.7	42.8	45.0	53.2	15	12.4	12.1	57	103%
Ho	8.5	6.97	7.30	8.68	2	1.6	1.6	8.9	105%
Er	20.7	17.3	18.0	21.6	4.5	3.4	3.2	21.5	104%
Tm	2.6	2.17	2.27	2.66	<0.5	<0.4	<0.4	2.7	102%
Yb	15.0	12.2	12.7	15.4	3.3	2.7	2.5	15.5	104%
Lu	2.0	1.51	1.59	1.90	<0.6	<0.5	<0.5	2.1	105%
Y	200	181	187	225	43	34	32	222.5	111%
S/SO_4	91600	70000	97000	130000	89100	90400	91800	83161.3	91%
Na	1480	--	--	265	--	--	1300	1402	95%
K	40300	--	--	1990	--	--	38400	36851	91%

Element / Sample	Extraction, %		
	12 h	24 h	Final
U	93%	95%	96%
Th	81%	84%	84%
La	28%	31%	30%
Ce	30%	32%	31%
Pr	32%	35%	34%
Nd	33%	37%	37%
Sm	42%	45%	46%
Eu	45%	51%	53%
Gd	56%	60%	62%
Tb	61%	64%	66%
Dy	73%	78%	79%
Ho	77%	81%	82%
Er	78%	84%	85%
Tm	80%	85%	85%
Yb	78%	82%	84%
Lu	70%	75%	76%
Y	80%	84%	86%
S	20%	26%	29%
Na	--	--	15%
K	--	--	4%



Acid Consumption:

Acid In:	117.0	g
Free Acid Out:	32.4	g
Acid Consumption:	99.6	kg H2SO4 / t Ore

Extractions (PLS vs. Calc'd Hd):

TREE:	36%
LREE:	32%
HREE:	78%

Weight Loss: -1%

Date: 04-Mar-13

Test: **POX 1** **13849-001** **Technologist:** D. Matthews

Purpose: **To determine the effect of grind size on kinetic oxidation.**

Sample: **Combined Conc** **Grind:** as received K80 ~ 40 micron
LIMS 03431 Feb

Procedure:
The feed pulp sample was placed into an autoclave vessel.
The pulp was mixed with a mechanical stirrer.
The pulp was adjusted to pH1.8 using concentrated sulphuric acid and maintained for 30 minutes.
The vessel was then sealed and heat up began.
Once the sample was at oxygen over pressure was applied.

Kinetic pulp samples were removed from the autoclave, via a bomb sampler, at 60 and 120 minutes.
The kinetic samples were weighed and immediately filtered.
The filtrates were collected and submitted for assays.
The filtercakes were displacement washed several times with fresh water.
The wash solutions were discarded. The filtercakes were dried, weighed, and submitted for assays.

After 240 minutes at temperature and pressure the autoclave was cooled to ~80°C.
The oxygen pressure was bleed off.
The autoclave was opened and the hot pulp was quickly filtered.
The filtrate was collected and a sample was submitted for assays.
The filtercake was displacement washed several times with DI water.
The wash solutions were discarded. The filtercake was dried, weighed, and submitted for assays.

Conditions:

POX Feed Wet Weight (g):	649		
POX Feed Dry Equiv. Weight (g):	**500**		
Solution Volume (mL):	**1166**		
Initial pH:	**8.7**		
98% H2SO4 Acid (g):	**20.1**	pH 1.8 for 30 minutes	kg/t = 39.3
Pulp Density (% solids **w/v**):	**30.0**	(w/w) before acid addition	
Temperature (°C):	**210**		
O_2 Over Pressure (psi):	**100**		
Time (at temperature) (min):	**240**		
Time (heat-up) (min):	**45**		

Autoclave Leach Data:

Elapsed Time min	D time	Temp °C	Pressure (psi) Total meas	Steam calculated	Over calculated	O_2 calculated	Off-Gas Flow mL/min	Off-Gas O_2 %	Remarks
0		22	35	-	-	-	-	-	Start Heating
45/0	0.0	210	365	**262**	**103**		250		Start Test
10	10.0	210	365	**262**	**103**	**74**	250	72	
20	10.0	210	365	**262**	**103**	**93**	250	90	
30	10.0	209	365	**256**	**109**	**99**	250	91	
40	10.0	210	365	**262**	**103**	**95**	250	92	
50	10.0	210	365	**262**	**103**	**95**	250	92	
60	10.0	210	365	**262**	**103**	**96**	250	93	remove sample
75	15.0	210	365	**262**	**103**	**97**	250	94	
90	15.0	209	365	**256**	**109**	**102**	250	94	
105	15.0	210	365	**262**	**103**	**98**	250	95	
120	15.0	210	365	**262**	**103**	**98**	250	95	remove sample
135	15.0	210	365	**262**	**103**	**98**	250	95	
150	15.0	209	365	**256**	**109**	**104**	250	96	
165	15.0	210	365	**262**	**103**	**99**	250	96	
180	15.0	210	365	**262**	**103**	**99**	250	96	
195	15.0	210	365	**262**	**103**	**99**	250	96	
210	15.0	210	365	**262**	**103**	**100**	250	97	
240	30.0	210	365	**262**	**103**	**100**	250	97	End Test
AVG. 0:120	**240**	**210**	**365**	**261**	**104**	**98**	**250**	**94**	

Date: 04-Mar-13

Test: **POX 1** **13849-001** **Technologist:** D. Matthews

Purpose: **To determine the effect of grind size on kinetic oxidation.**

Sample: **Combined Conc** **Grind:** as received K80 ~ 40 micron

Sample Data:

Time (h)	Pulp Sample (g)	Filtrate SG mg/mL	Dry Filtercake g	pH	Filtrate EMF (mV)	Free Acid g/L	Pulp Density (w/w)	Remarks
60	83.9	1.080	23.20	0.5	412	37.2	27.7	slow filtration
120	83.6	1.081	24.50	<0.5	501	48.9	29.3	medium filtration
Final	1547.6	1.082	443.30	<0.5	565	45.9	28.6	medium filtration
Purge	41.0	…	…	…	…	…	…	…
	1756.1		491.0					

Project No.: **13849-001**
Client: **Appia Energy Corp**

Project Technologist: D. Matthews
Date of Test: 04-Mar-13

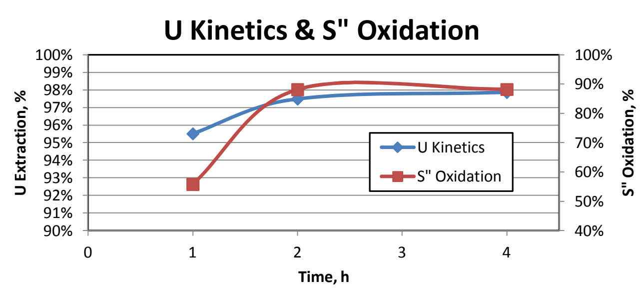
Test **POX-1** **500 g**

Metallurgical Balance

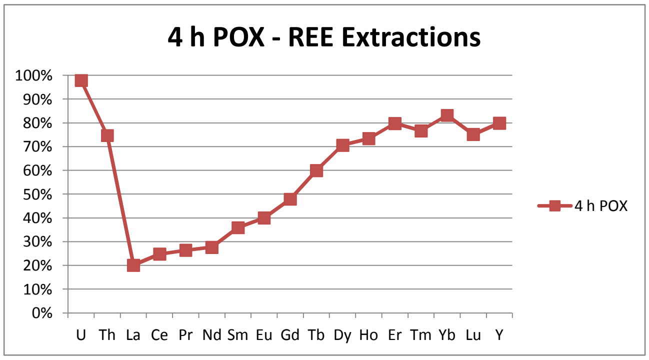
	Assays								Balance OUT/IN
	Head, g/t	PLS, mg/L			Residue, g/t			Calc'd Head	%
		60 min	120 min	Final	60 min	120 min	Final		
Amt, mL or g	500	56	55	1021	23.2	24.5	443.3	491.0	
U	814	376	386	378	42.8	22.2	19	892	110%
Th	1140	--	--	386			301	1074	94%
Fe	89500	16300	3870	4590	--	--	--		
Fe++	--	13000	433	20	--	--	--		
La	1550	--	--	137			1390	1571	101%
Ce	2580	--	--	295			2290	2747	106%
Pr	290	--	--	33.1			236	289	100%
Nd	903	--	--	110			735	917	102%
Sm	146	--	--	22.8			104	146	100%
Eu	6.4	--	--	1.20			4.60	6.9	108%
Gd	84.8	--	--	19.4			53.9	93	110%
Tb	12	--	--	3.05			5.2	11.7	98%
Dy	55.7	--	--	15.8			16.8	52	93%
Ho	8.5	--	--	2.70			2.50	8.5	100%
Er	20.7	--	--	7.25			4.7	21.0	101%
Tm	2.6	--	--	0.90			0.70	2.7	104%
Yb	15	--	--	5.42			2.8	15.0	100%
Lu	2	--	--	0.71			<0.6	2.2	109%
Y	200	--	--	73.2	--	--	47.0	211.1	106%
S''	88000	--	--	--	42600	10800	11000		
Na	1500	--	--	26	--	--	--		
K	39000	--	--	124	--	--	--		

Acid Balace

Acid IN, g	
Initial	19.7
Generatred	119.4
TOTAL IN	139.1
Acid OUT, g	
Free Acid	51.9
w MeSO$_4$	55.0
as SO$_4$ in Residue	28.7
TOTAL Out	135.7
OUT/IN	98%



Element / Sample	Extraction, %		
	60 min	120 min	Final
U	96%	97%	98%
Th			75%
La			20%
Ce			25%
Pr			26%
Nd			28%
Sm			36%
Eu			40%
Gd			48%
Tb			60%
Dy			71%
Ho			73%
Er			80%
Tm			77%
Yb			83%
Lu			75%
Y			80%
S			
Na			
K			



Extractions (PLS vs. Calc'd Hd):

TREE:	28%
LREE:	24%
HREE:	70%

Project No.:	**13849-001**	**Project Technologist:**	Krystal Davis
Client:	**Appia Energy Corp**	**Date of Test:**	07-Mar-13
Test ID:	**AP-1 24 h + 1 h**		

Objective: To conduct a sulphuric acid leach test on a uranium ore
To study the kinetics of uranium dissolution under various conditions

Feed: Conc 1
Grind: -- 50% solids in lab ball mill
(note time/kg, weight charged, % solids in mill and mill size/type)

Procedure:

1. The sample was ground to the desired P80 and the mill discharge pan filtered
2. A small sample was cut for %H2O determination (~50 g)
3. The wet wt. of feed was recorded and sulphuric acid was added to 40 kg/t with acid addition recorded.
4. The Slurry was charged to a column and brought to desired temperature (if required)
5. Low flow air was saturated with water vapour by sparging through a pair of flasks containing water before being injected into the column.
6. After 24 hours, the column was discharged and the total weight of the acidic solids was recorded.
7. the kettle tare weigh was recorded and the solids were slurried to target.
8. The slurry was mixed for one hour with pH, ORP, and temperature measurements recorded at regular intervals.
13. After completion of the test the reactor was allowed to cool (if at T) and agitation was stopped
14. The final weight of vessel was recorded and the contents filtered and washed once with 300 mL pH 2.0 water, 3 times with 300 mL DI
15. The filtration rate was recorded along with notes on residue and liquor colour
16. The filtrate weight and SG was recorded, along w/ pH, FA and ORP
17. Dry solids and filtrate were submitted for analysis, the wash solution was rejected

All pulp samples must be labeled "NORM" for Naturally Occurring Radioactive Material

Conditions:

Reactor Tare (incl. lid and baffles):		g
Wet Feed Weight:	518.1	g
Moisture:	23%	
Dry Feed Weight:	400.0	g
Acid Addition:	16.7	g H_2SO_4
Target Slurry Density:	50	w/w%
Water added:	281.8	mL
Water in Feed:	118.1	mL
Target FA:	--	g/L H_2SO_4
Saturated Air Flowrate:	0.67	mL/min
Target ORP:	--	mV *(vs. Ag|AgCl sat'd KCl electrode)*
Fe3+	--	g/L
Grind	~95	µm
Temperature:	70/ambient	ºC

Analytical Requirements:

Kinetic solutions	REE, Fe, Fe2+, SO4
Kinetic residues	REE, S(tot)
Final PLS	REE, Fe2+, ICP, SO4
Final residue	REE, Fe, Na, K, S(tot)

Test Details:

Elapsed Time (hh:mm)	(hrs)	Temp (°C)	pH	ORP (mV)	FA (g/L H_2SO_4)	Reagents Added H_2SO_4 (g)		Comments
9:00	-	50.7				16.70		Heat column up to 70˚
10:00	0.0	76.7						Hold for 24 h
10:23	24.0	75.8						Remove from column
11:42	0.00	20.9	3.95	244				Start mixing
12:00	0.25	21.0	3.96	233				
12:15	0.50	21.0	4.01	219				
12:30	0.75	21.0	4.05	209				
12:42	1.00	20.9	4.07	203				End mixing
TWA						16.7	0	mL or g
						40.1	0.0	kg/t

Project No.: **13849-001**
Client: **Appia Energy Corp**

Project Technologist: Krystal Davis
Date of Test: 07-Mar-13

Test ID: **AP-1** **24 h + 1 h**

Objective: To conduct a sulphuric acid leach test on a uranium ore
To study the kinetics of uranium dissolution under various conditions

	Pulp Weight	PLS Filtrate	Wash Filtrate	Wet Residue	Dry Residue	Pulp Density	Cake Moisture	PLS Filtrate			
								pH	ORP[1]	SG	Free Acid
Sample	g	g	g	(g)	(g)	w/w	(%)		(mV)		(g/L)
24 hr						#DIV/0!	#DIV/0!				
Final	1313.2	745.3	904.5	548.8	409.1	31.2%	26	4.17	208	1.008	0
							TWA:	**4.2**	**208.0**		**0.0**

[2] Assuming SG of pore water in washed cake is 1.00.

Notes: DI used to rinse: 770.48g

Filtration: Whatman 3, 185 — enter filter paper type, size
20 min — enter final filtration time (time to no liquid on cake)

Filtrate Colour: Clear
Residue Colour: Grey/Brown

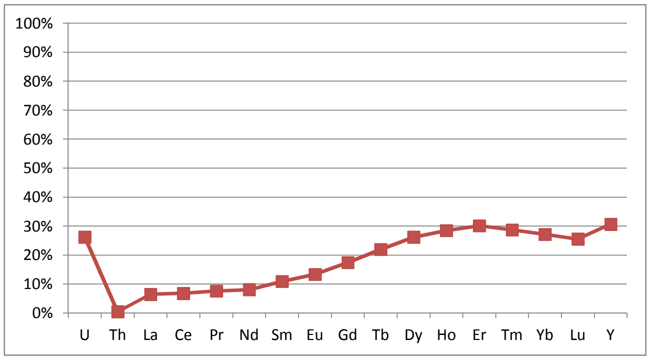
Project No.: **13849-001** **Project Technologist:** Krystal Davis
Client: **Appia Energy Corp** **Date of Test:** 07-Mar-13

Test **AP-1** **400 g**

Metallurgical Balance

	Assays				Balance OUT/IN %
	Head, g/t	Final PLS mg/L	Final Res, g/t	Calc'd Head	
Amt, mL or g	400	897	409.1	409.1	
U	814	106	655	887	109%
Th	1140	2.38	1200	1205	106%
Fe	89500	2680	85500	91373	102%
Fe++	--	2540	--		
La	1550	47.6	1520	1624	105%
Ce	2580	89.9	2710	2907	113%
Pr	290	9.72	260	281	97%
Nd	903	34.8	870	946	105%
Sm	146	7.64	137	154	105%
Eu	6.4	0.47	6.7	7.7	121%
Gd	84.8	8.08	84.1	102	120%
Tb	12.0	1.18	9.2	11.8	98%
Dy	55.7	6.27	38.8	53	94%
Ho	8.5	1.07	5.9	8.2	97%
Er	20.7	2.77	14.1	20.2	97%
Tm	2.6	0.33	<1.8	2.5	97%
Yb	15.0	1.85	10.9	15.0	100%
Lu	2.0	0.25	<1.6	2.1	107%
Y	200	30.0	149.0	214.7	107%
S/SO_4	91600	6600	91000	83016.0	91%
Na	1480	21	2200	2246	152%
K	40300	28	36800	36861	91%

Element / Sample	Final
U	26%
Th	0%
La	6%
Ce	7%
Pr	8%
Nd	8%
Sm	11%
Eu	13%
Gd	17%
Tb	22%
Dy	26%
Ho	28%
Er	30%
Tm	29%
Yb	27%
Lu	26%
Y	31%
S	5%
Na	2%
K	0%



Acid Consumption:

Acid In:	16.0	g
Free Acid Out:	0.0	g
Acid Consumption:	40.1	kg H2SO4 / t Ore

Weight Loss: -2%

Extractions (PLS vs. Calc'd Hd)

TREE:	8%
LREE:	7%
HREE:	26%

Project:	13849-001	**Date:**	1-Mar-13
Client:	Appia Energy	**Technologist:**	Krystal Davis

Test: AB-1

Purpose: To perform an acid bake and water leach on Appia Flotation Concentrate.

Sample: Flot Conc. ~200 g dry

Grind:
K80= µm

Procedure:

1. The conc. was ground at 50% solids in a lab rod mill for the indicated time. The mill discharge was filtered.
2. The filtered ground ore was combined with the specified amount of sulphuric acid in a crucible (record tare weight of crucible) inside a fumehood. The paste was made homogeneous.
3. The pasted was cured for 1 hour
4. The muffle furnace was heated to 200ºC
5. After curing, the ore/acid mixture was placed in a muffle furnace at temperature and baked for 3 hours.
6. After 3 hours at temperature, the furnace was turned off and allowed to cool to room temperature.
7. Weighed calcined sample in crucible, transfered calcine to mortar and pulverized if necessary.
8. Acid Baked solids were pulped to 10% solids in DI water in a 2L Pyrex reaction kettle.
9. A lid and condenser was attached to the kettle and a Teflon impellor/mechanical mixer setup was used to agitate the slurry. The slurry was heated to 90°C via heating mantle/temperature controller setup. pH was maintained below 1.5
11. After 4 hours at temperature, the pulp was weighed, filtered and the filtrate was collected. The solids were repulped in DI water, filtered, and displacement washed with 3 x 300 mL DI water.
12. Final solids were dried to constant mass. Final PLS, wash and soilds were assayed per the table below
13. Free acid and sg of the Final Solution was determined.

Conditions:

Stage 1

Crucible Tare Weight:			769.2	g
% Moisture of Feed:			22.84	%
Wet Weight Feed Added:			259.2	g
Dry Weight of Conc.:			200.0	g
Sulphuric Acid (96% H_2SO_4)	A/O	1000 kg/t	208.3	g
Total feed weight			408.3	g
Cure Time - acidic paste			1	h
Color of paste:			grey	
Bake time:			3	h
Acid Bake Temperature:			200	°C

Calcination Results:

Calcine Net wt.	393.2	g
Color:	dark grey	
Consistancy	solid	
Pulverized discharge (if necessary)		
Time		min

Samples for assay:

Final calcine solids :

Acid Bake Comments:

Water Leach Conditions:

Water Used to Wash Crucible (mL):	0	
Pyrex Reactor Tare (g):	1997.4	
Leach Feed - Acid Bake Product (g):	391.9	(as received)
Calculated Acid Bake Feed wt (g):	199.3	
DI Water to add (mL):	1794	**Total DI Added: 1794**
Pulp Density (%solids)	17.9	w/w Leach Feed (Acid Bake Product)
Pulp Density (%solids)	10.0	w/w Calculated Acid Bake Feed Should be: 10 w/w%
Time (h):	4	
Temperature (°C):	90	(temperature controlled with heating mantle)
Net Mass of slurry transferred to 4L Reactor		g
Initial gross mass		g

Leach Data:

Time	Elapsed Time hours	Temp °C	pH	ORP	Pulp colour	~~Kinetic Sample~~ ~~Pulp wt. g~~	~~PLS wt. g~~	~~PLS SG~~	Comments H_2SO_4 added, g (to maintain pH < 1.5)
8:30	-	25.7	1.01	484	g - b				grey - brown pulp, start heating
9:00	0	90.2	0.43	630	g - b				
10:21	1	91.3	0.48	579	g - b				
11:00	2	90.4	0.48	570	g - b				
12:00	3	90.3	0.48	553	g - b				
13:00	4	90.1	0.46	549	g - b				End Test

Results:

Final Pulp wt (g):	1867.0
Filtrate Collected wt (g):	1569.8
Filtrate SG:	1.0994
Filtrate Collected Vol (mL):	1427.9
Filtrate Calculated Vol (mL):	1553.7
Filtrate Free Acid (H2SO4 g/L):	51.7
Wash Solution (mL):	52
Solid Residue: Tare (g):	15.5
Wet wt (g):	257.8
Dry wt (g):	174.3
Net Dry Wt (g)	158.8

Notes:

Filter Paper:	#3, 150 mm	
PLS Filtration Time:	5	min
Repulp + Displacements:	15	min
Filtrate Colour:	yellow-green	
Residue Colour:	brown	
Wash Wt.:	1124.1	g

Filtrate Free Acid	51.68
pH	0.65
ORP	419

Assays:

Combined Wash Liquor: Ce, Y, P, Fe
Final Liquor: REE, Si, P, Fe, Al, Ca, Mn, Cl, F, FAT
Final Solids: REE (ICP-MS), Si, P, Fe, Al, Ca, Mn, Cl, F, S
(XRF) Ce, La, Nd, Pr

Metallurgical Balance:

	Assay, mg/L or %				Recovery	Out/In	Distribution %		
	Feed	4h PLS	Wash	Residue	%	%	4h PLS	Wash	Residue
g or mL	199	1554	52	159					
La	0.155	122		0.094	55.9	110	56		44
Ce	0.258	287		0.107	72.4	120	72		28
Pr	0.029	33.8		0.00720	82.1	111	82		18
Nd	0.0903	115		0.0163	87.3	114	87		13
Sm	0.0146	18.8		0.00208	89.8	112	90		10
Eu	0.00064	0.77		0.00013	85.3	110	85		15
Gd	0.00848	12.1		0.00113	91.3	122	91		9
Tb	0.0012	1.30		0.00014	90.1	94	90		10
Dy	0.00557	6.18		0.00068	89.9	96	90		10
Ho	0.00085	0.92		0.00010	90.0	94	90		10
Er	0.00207	2.32		0.00027	89.4	98	89		11
Tm	0.00026	0.28		<0.00003	90.1	93	90		10
Yb	0.00150	1.72		0.00019	89.9	99	90		10
Lu	0.00020	0.22		<0.00005	81.1	106	81		19
Y	0.02	24.7		0.0015	94.2	102	94		6
Sc	<0.0025	0.87		<0.0025	25.4	107	25		75
U	0.0814	111		0.00177	98.4	108	98		2
Th	0.114	138		0.0135	90.9	104	91		9
P	0.20	227			100.0	88	100		0
Fe	8.95	5080		7.55	39.7	111	40		60
Ca	1.19	925			100.0	61	100		0
Mn	0.02	34.5			100.0	116	100		0
Al	6.51	7350			100.0	88	100		0
Si	25.7	--			0.0	0	--		#DIV/0!

Acid Consumption:

Acid In:	200.0	g
Free Acid Out:	80.3	g
Acid Consumption:	598.5	kg H2SO4 / t Conc.

Weight Loss:	20%
TREE:	72%
LREE:	71%
HREE:	92%

Residue %S 10.1

Project: 13849-001
Client: Appia Energy
Date: 1-Mar-13
Technologist: Krystal Davis

Test: AB-2

Purpose: To perform an acid bake and water leach on Appia Flotation Concentrate.

Sample: Flot Conc. ~200 g dry
Grind:
K80= µm

Procedure:

1. The conc. was ground at 50% solids in a lab rod mill for the indicated time. The mill discharge was filtered.
2. The filtered ground ore was combined with the specified amount of sulphuric acid in a crucible (record tare weight of crucible) inside a fumehood. The paste was made homogeneous.
3. The pasted was cured for 1 hour
4. The muffle furnace was heated to 300ºC
5. After curing, the ore/acid mixture was placed in a muffle furnace at temperature and baked for 3 hours.
6. After 3 hours at temperature, the furnace was turned off and allowed to cool to room temperature.
7. Weighed calcined sample in crucible, transfered calcine to mortar and pulverized if necessary.
8. Acid Baked solids were pulped to 10% solids in DI water in a 2L Pyrex reaction kettle.
9. A lid and condenser was attached to the kettle and a Teflon impellor/mechanical mixer setup was used to agitate the slurry. The slurry was heated to 90°C via heating mantle/temperature controller setup. pH was maintained below 1.5
11. After 4 hours at temperature, the pulp was weighed, filtered and the filtrate was collected. The solids were repulped in DI water, filtered, and displacement washed with 3 x 300 mL DI water.
12. Final solids were dried to constant mass. Final PLS, wash and soilds were assayed per the table below
13. Free acid and sg of the Final Solution was determined.

Conditions:

Stage 1

Crucible Tare Weight:			871.6	g
% Moisture of Feed:			22.84	%
Wet Weight Feed Added:			259.2	g
Dry Weight of Conc.:			200.0	g
Sulphuric Acid (96% H_2SO_4)	A/O	1000 kg/t	208.3	g
Total feed weight			408.3	g
Cure Time - acidic paste			1	h
Color of paste:			grey	
Bake time:			3	h
Acid Bake Temperature:			300	°C

Calcination Results:

Calcine Net wt.	299.8	g
Color:	grey	
Consistancy	solid	
Pulverized discharge (if necessary)		
Time		min

Samples for assay:

Final calcine solids :

Acid Bake Comments:

Water Leach Conditions:

Water Used to Wash Crucible (mL):	0			
Pyrex Reactor Tare (g):	1835.12			
Leach Feed - Acid Bake Product (g):	292	(as received)		
Calculated Acid Bake Feed wt (g):	194.8			
DI Water to add (mL):	1753	**Total DI Added:**	**1753**	
Pulp Density (%solids)	14.3	w/w Leach Feed (Acid Bake Product)		
Pulp Density (%solids)	10.0	w/w Calculated Acid Bake Feed	Should be: 10	w/w%
Time (h):	4			
Temperature (°C):	90	(temperature controlled with heating mantle)		
Net Mass of slurry transferred to 4L Reactor		g		
Initial gross mass		g		

Leach Data:

Time	Elapsed Time hours	Temp °C	pH	ORP	Pulp colour	~~Kinetic Sample~~ ~~Pulp wt. g~~	~~PLS wt. g~~	~~PLS SG~~	Comments H_2SO_4 added, g (to maintain pH < 1.5)
8:25	-	20.9	2.13	518	Grey				Start Heating
9:05	0	90.1	1.69	608	"				t = 0
10:21	1	88.2	1.56	563	"				
11:00	2	88.9	1.32	569	"				
12:00	3	89.5	1.35	534	"				
13:00	4	91.8	1.44	533	"				End Test

Results:

	Final Pulp wt (g):	2187.4
	Filtrate Collected wt (g):	1846
	Filtrate SG:	1.0528
	Filtrate Collected Vol (mL):	1753.4
	Filtrate Calculated Vol (mL):	1906.1
	Filtrate Free Acid (H2SO4 g/L):	12.0
	Wash Solution (mL):	826
Solid Residue:	Tare (g):	13.1
	Wet wt (g):	302
	Dry wt (g):	193.7
	Net Dry Wt (g)	180.6

Notes:

Filter Paper:	#3, 150 mm	
PLS Filtration Time:	5	min
Repulp + Displacements:	15	min
Filtrate Colour:	yellow-green	
Residue Colour:	brown	
Wash Wt.:	826.3	g

Filtrate Free Acid	11.96
pH	1.11
ORP	455

Assays:

Combined Wash Liquor: Ce, Y, P, Fe
Final Liquor: REE, Si, P, Fe, Al, Ca, Mn, Cl, F, FAT
Final Solids: REE (ICP-MS), Si, P, Fe, Al, Ca, Mn, Cl, F, S
(XRF) Ce, La, Nd, Pr

Metallurgical Balance:

	Assay, mg/L or %				Recovery	Out/In	Distribution %		
	Feed	4h PLS	Wash	Residue	%	%	4h PLS	Wash	Residue
g or mL	195	1906	826	181					
La	0.155	120		0.0653	66.0	115	66		34
Ce	0.258	243		0.0917	73.7	125	74		26
Pr	0.029	26.1		0.00840	76.6	115	77		23
Nd	0.0903	88.6		0.0238	79.7	120	80		20
Sm	0.0146	14.3		0.00410	78.6	122	79		21
Eu	0.00064	0.58		0.00032	65.7	135	66		34
Gd	0.00848	8.76		0.00252	78.6	129	79		21
Tb	0.0012	0.95		0.00050	66.7	116	67		33
Dy	0.00557	4.49		0.00158	75.0	105	75		25
Ho	0.00085	0.67		0.00026	73.1	105	73		27
Er	0.00207	1.74		0.00058	76.0	108	76		24
Tm	0.00026	0.21		0.00010	68.9	115	69		31
Yb	0.00150	1.32		0.00041	77.3	111	77		23
Lu	0.00020	0.17		0.00010	64.2	130	64		36
Y	0.02	19.0		0.0028	87.7	106	88		12
Sc	<0.0025	0.59		<0.0025	19.9	116	20		80
U	0.0814	87.2		0.00276	97.1	108	97		3
Th	0.114	98.0		0.0283	78.5	107	79		21
P	0.20	64			100.0	31	100		0
Fe	8.95	2080		7.99	21.6	105	22		78
Ca	1.19	618			100.0	51	100		0
Mn	0.02	30.1			100.0	127	100		0
Al	6.51	5870			100.0	88	100		0
Si	25.7	--			0.0	0	--		#DIV/0!

Acid Consumption:

Acid In: 200.0 g
Free Acid Out: 22.8 g
Acid Consumption: 886.0 kg H2SO4 / t Conc.

Residue %S 7.17

Weight Loss: 7%

TREE: 73%
LREE: 73%
HREE: 81%

Project: 13849-001 **Date:** 3-Mar-13
Client: Appia Energy **Technologist:** Krystal Davis

Test: AB-3

Purpose: To perform an acid bake and water leach on Appia Flotation Concentrate.

Sample: Flot Conc. ~200 g dry **Grind:**
K80= µm

Procedure:

1. The conc. was ground at 50% solids in a lab rod mill for the indicated time. The mill discharge was filtered.
2. The filtered ground ore was combined with the specified amount of sulphuric acid in a crucible (record tare weight of crucible) inside a fumehood. The paste was made homogeneous.
3. The pasted was cured for 1 hour
4. The muffle furnace was heated to 250ºC
5. After curing, the ore/acid mixture was placed in a muffle furnace at temperature and baked for 3 hours.
6. After 3 hours at temperature, the furnace was turned off and allowed to cool to room temperature.
7. Weighed calcined sample in crucible, transfered calcine to mortar and pulverized if necessary.
8. Acid Baked solids were pulped to 10% solids in DI water in a 2L Pyrex reaction kettle.
9. A lid and condenser was attached to the kettle and a Teflon impellor/mechanical mixer setup was used to agitate the slurry. The slurry was heated to 90°C via heating mantle/temperature controller setup. pH was maintained below 1.5
11. After 4 hours at temperature, the pulp was weighed, filtered and the filtrate was collected. The solids were repulped in DI water, filtered, and displacement washed with 3 x 300 mL DI water.
12. Final solids were dried to constant mass. Final PLS, wash and soilds were assayed per the table below
13. Free acid and sg of the Final Solution was determined.

Conditions:

Stage 1

Crucible Tare Weight:			787.1	g
% Moisture of Feed:			22.84	%
Wet Weight Feed Added:			259.2	g
Dry Weight of Conc.:			200.0	g
Sulphuric Acid (96% H_2SO_4)	A/O	500 kg/t	104.2	g
Total feed weight			304.2	g
Cure Time - acidic paste			1	h
Color of paste:			grey	
Bake time:			3	h
Acid Bake Temperature:			250	°C

Calcination Results:

Calcine Net wt.	265.3	g
Color:	grey	
Consistancy		
Pulverized discharge (if necessary)		
Time		min

Samples for assay:

Final calcine solids :

Acid Bake Comments:

Water Leach Conditions:

Water Used to Wash Crucible (mL):		
Pyrex Reactor Tare (g):	1835.4	
Leach Feed - Acid Bake Product (g):	265.3	(as received)
Calculated Acid Bake Feed wt (g):	200.0	
DI Water to add (mL):	1800	**Total DI Added: 1800**
Pulp Density (%solids)	12.8	w/w Leach Feed (Acid Bake Product)
Pulp Density (%solids)	10.0	w/w Calculated Acid Bake Feed Should be: 10 w/w%
Time (h):	4	
Temperature (°C):	90	(temperature controlled with heating mantle)
Net Mass of slurry transferred to 4L Reactor		g
Initial gross mass		g

Leach Data:

Time	Elapsed Time hours	Temp °C	pH	ORP	Pulp colour	Kinetic Sample Pulp wt. g	PLS wt. g	PLS SG	Comments H_2SO_4 added, g (to maintain pH < 1.5)
8:30	-	22.7	1.97	495	Brown				Start Heating
10:10	0	92.0	1.89	527	"				t = 0, 17.7g H2SO4
11:10	1	92.5	1.40	522	"				
12:05	2	91.5	1.40	504	"				
13:00	3	92.8	1.37	499	"				
14:10	4	93.2	1.33	498	"				End Test

Results:

Final Pulp wt (g):	2533.3
Filtrate Collected wt (g):	2209.11
Filtrate SG:	1.0377
Filtrate Collected Vol (mL):	2128.9
Filtrate Calculated Vol (mL):	2280.8
Filtrate Free Acid (H2SO4 g/L):	13.9
Wash Solution (mL):	782
Solid Residue: Tare (g):	13.02
Wet wt (g):	240
Dry wt (g):	179.5
Net Dry Wt (g)	166.48

Notes:

Filter Paper:	#3, 150 mm	
PLS Filtration Time:		hrs
Repulp + Displacements:		hrs
Filtrate Colour:		
Residue Colour:		
Wash Wt.:	781.8	g
Wash SG:		g/mL

Filtrate Free Acid	13.92
pH	1.88
ORP	381

Assays:

Combined Wash Liquor: Ce, Y, P, Fe
Final Liquor: REE, Si, P, Fe, Al, Ca, Mn, Cl, F, FAT
Final Solids: REE (ICP-MS), Si, P, Fe, Al, Ca, Mn, Cl, F, S
(XRF) Ce, La, Nd, Pr

Metallurgical Balance:

	Assay, mg/L or %				Recovery	Out/In	Distribution %		
	Feed	4h PLS	Wash	Residue	%	%	4h PLS	Wash	Residue
g or mL	200	2281	782	166					
La	0.155	69.2		0.0748	55.9	91	56		44
Ce	0.258	153		0.0991	67.9	100	68		32
Pr	0.029	17.1		0.00841	73.6	91	74		26
Nd	0.0903	58.2		0.0254	75.8	97	76		24
Sm	0.0146	9.71		0.00383	77.6	98	78		22
Eu	0.00064	0.40		0.00021	72.3	99	72		28
Gd	0.00848	6.32		0.00203	81.0	105	81		19
Tb	0.0012	0.69		0.00029	76.5	86	77		23
Dy	0.00557	3.58		0.00114	81.1	90	81		19
Ho	0.00085	0.50		0.00016	81.1	83	81		19
Er	0.00207	1.38		0.00038	83.3	91	83		17
Tm	0.00026	0.15		0.00004	83.7	79	84		16
Yb	0.00150	0.97		0.00024	84.7	87	85		15
Lu	0.00020	0.13		<0.00005	78.1	95	78		22
Y	0.02	14.1		0.0031	86.2	93	86		14
Sc	<0.0025	0.34		<0.0025	15.7	99	16		84
U	0.0814	67.9		0.00256	97.3	98	97		3
Th	0.114	75.2		0.0341	75.1	100	75		25
P	0.20	87			100.0	49	100		0
Fe	8.95	2630		7.36	32.9	102	33		67
Ca	1.19	677			100.0	65	100		0
Mn	0.02	23.2			100.0	114	100		0
Al	6.51	3650			100.0	64	100		0
Si	25.7	--			0.0	0	--		#DIV/0!

Acid Consumption:

Acid In:	100.0	g
Free Acid Out:	31.7	g
Acid Consumption:	341.3	kg H2SO4 / t Conc.

Weight Loss:	17%
TREE:	67%
LREE:	67%
HREE:	83%

Project: 13849-001
Client: Appia Energy

Date: 3-Mar-13
Technologist: Krystal Davis

Test: AB-4

Purpose: To perform an acid bake and water leach on Appia Flotation Concentrate.

Sample: Flot Conc. ~200 g dry

Grind:
K80= µm

Procedure:

1. The conc. was ground at 50% solids in a lab rod mill for the indicated time. The mill discharge was filtered.
2. The filtered ground ore was combined with the specified amount of sulphuric acid in a crucible (record tare weight of crucible) inside a fumehood. The paste was made homogeneous.
3. The pasted was cured for 1 hour
4. The muffle furnace was heated to 250ºC
5. After curing, the ore/acid mixture was placed in a muffle furnace at temperature and baked for 3 hours.
6. After 3 hours at temperature, the furnace was turned off and allowed to cool to room temperature.
7. Weighed calcined sample in crucible, transfered calcine to mortar and pulverized if necessary.
8. Acid Baked solids were pulped to 10% solids in DI water in a 2L Pyrex reaction kettle.
9. A lid and condenser was attached to the kettle and a Teflon impellor/mechanical mixer setup was used to agitate the slurry. The slurry was heated to 90°C via heating mantle/temperature controller setup. pH was maintained below 1.5
11. After 4 hours at temperature, the pulp was weighed, filtered and the filtrate was collected. The solids were repulped in DI water, filtered, and displacement washed with 3 x 300 mL DI water.
12. Final solids were dried to constant mass. Final PLS, wash and soilds were assayed per the table below
13. Free acid and sg of the Final Solution was determined.

Conditions:

Stage 1

Crucible Tare Weight:			728.8	g	
% Moisture of Feed:			22.84	%	
Wet Weight Feed Added:			259.2	g	
Dry Weight of Conc.:			200.0	g	260 g?
Sulphuric Acid (96% H_2SO_4)	A/O	300 kg/t	62.5	g	
Total feed weight			262.5	g	
Cure Time - acidic paste			1	h	
Color of paste:			grey		
Bake time:			3	h	
Acid Bake Temperature:			250	°C	

Calcination Results:

Calcine Net wt.	296.9	g
Color:	grey	
Consistancy	solid	
Pulverized discharge (if necessary)		
Time		min

Samples for assay:

Final calcine solids :

Acid Bake Comments:

Water Leach Conditions:

Water Used to Wash Crucible (mL):		
Pyrex Reactor Tare (g):	1886.5	
Leach Feed - Acid Bake Product (g):	296.9	(as received)
Calculated Acid Bake Feed wt (g):	200.0	
DI Water to add (mL):	1800	**Total DI Added: 1800**
Pulp Density (%solids)	14.2	w/w Leach Feed (Acid Bake Product)
Pulp Density (%solids)	10.0	w/w Calculated Acid Bake Feed Should be: 10 w/w%
Time (h):	4	
Temperature (°C):	90	(temperature controlled with heating mantle)
Net Mass of slurry transferred to 4L Reactor		g
Initial gross mass		g

Leach Data:

Time	Elapsed Time hours	Temp °C	pH	ORP	Pulp colour	~~Kinetic Sample~~ ~~Pulp wt. g~~	~~PLS wt. g~~	~~PLS SG~~	Comments H_2SO_4 added, g (to maintain pH < 1.5)
8:30	-	22.1	2.15	449	Brown				Start Heating
10:10	0	87.6	1.57	446	"				t = 0, 3.4g H2SO4
11:10	1	87.4	1.41	434	"				
12:02	2	92.3	1.45	427	"				
13:00	3	88.7	1.42	421	"				
14:10	4	90.5	1.43	421	"				End Test

Results:

	Final Pulp wt (g):	2307.9
	Filtrate Collected wt (g):	1937.45
	Filtrate SG:	1.0244
	Filtrate Collected Vol (mL):	1891.3
	Filtrate Calculated Vol (mL):	2033.5
	Filtrate Free Acid (H2SO4 g/L):	4.5
	Wash Solution (mL):	825
Solid Residue:	Tare (g):	12.54
	Wet wt (g):	317.27
	Dry wt (g):	237.3
	Net Dry Wt (g)	224.76

Notes:

Filter Paper:	#3, 150 mm	
PLS Filtration Time:		hrs
Repulp + Displacements:		hrs
Filtrate Colour:		
Residue Colour:		
Wash Wt.:	824.5	g
Wash SG:		g/mL

Filtrate Free Acid	4.51
pH	2.02
ORP	355

Assays:

Combined Wash Liquor: Ce, Y, P, Fe
Final Liquor: REE, Si, P, Fe, Al, Ca, Mn, Cl, F, FAT
Final Solids: REE (ICP-MS), Si, P, Fe, Al, Ca, Mn, Cl, F, S
(XRF) Ce, La, Nd, Pr

Metallurgical Balance:

	Assay, mg/L or %				Recovery	Out/In	Distribution %		
	Feed	4h PLS	Wash	Residue	%	%	4h PLS	Wash	Residue
g or mL	200	2034	825	225					
La	0.155	81.3		0.0866	45.9	116	46		54
Ce	0.258	157		0.144	49.7	125	50		50
Pr	0.029	16.4		0.0138	51.8	111	52		48
Nd	0.0903	56.2		0.0439	53.7	118	54		46
Sm	0.0146	9.41		0.00664	56.2	117	56		44
Eu	0.00064	0.44		0.00030	57.0	123	57		43
Gd	0.00848	6.81		0.00351	63.7	128	64		36
Tb	0.0012	0.84		0.00053	58.9	121	59		41
Dy	0.00557	4.37		0.00186	68.0	117	68		32
Ho	0.00085	0.64		0.00028	67.4	114	67		33
Er	0.00207	1.73		0.00061	72.0	118	72		28
Tm	0.00026	0.20		0.00007	72.1	108	72		28
Yb	0.00150	1.25		0.00042	72.9	116	73		27
Lu	0.00020	0.16		0.00006	70.7	115	71		29
Y	0.02	17.9		0.0058	73.6	124	74		26
Sc	<0.0025	0.27		<0.0025	8.9	123	9		91
U	0.0814	97.9		0.00417	95.5	128	96		4
Th	0.114	76.1		0.0639	51.9	131	52		48
P	0.20	69			100.0	35	100		0
Fe	8.95	2470		7.86	22.1	127	22		78
Ca	1.19	775			100.0	66	100		0
Mn	0.02	30.3			100.0	133	100		0
Al	6.51	2530			100.0	40	100		0
Si	25.7	--			0.0	0	--		#DIV/0!

Acid Consumption:

Acid In:	60.0	g
Free Acid Out:	9.2	g
Acid Consumption:	254.1	kg H2SO4 / t Conc.

Weight Loss: -12%

TREE: 51%
LREE: 50%
HREE: 70%

Project: 13849-001 **Date:** 27-Mar-13

Client: Appia Energy **Technologist:** Krystal Davis

Test: Roast-WL

Purpose: To perform a roast on Appia concentrate sample. To produce calcine for leach testwork.

Sample: Flot Conc. ~400 g dry **Grind:**

K80= µm

Procedure:

1. ~~The conc. was ground at 50% solids in a lab rod mill for the indicated time. The mill discharge was filtered.~~
2. The muffle furnace was heated to 750ºC and the crucible tare weight was recorded.
3. The conc. was placed in a muffle furnace at temperature and baked for 2 hours. Sample was rabbled (mixed) every 30 minutes
4. After 2 hours at temperature, the furnace was turned off and the sample was cooled as quickly as possible.
5. Weighed calcined sample in crucible.
6. Roasted solids (calcine) were pulped to 10% solids in DI water in a 2L Pyrex reaction kettle.
7. A lid and condenser was attached to the kettle and a Teflon impellor/mechanical mixer setup was used to agitate the slurry. The slurry was heated to 90°C via heating mantle/temperature controller setup.
8. After 4 hours at temperature, the pulp was weighed, filtered and the filtrate was collected. The solids were repulped in DI water, filtered, and displacement washed with 3 x 300 mL DI water.
9. A sample of final solids were dried to constant mass. Final PLS, wash and soilds were assayed per the table below
10. Free acid and sg of the Final Solution was determined. Remaining residue split into two equal charges.

Conditions:

Stage 1

Crucible Tare Weight:			1567.9	g
% Moisture of Feed:			22.8	%
Wet Weight Feed Added:			518.4	g
Dry Weight of Conc.:			400.0	g
Sulphuric Acid (96% H_2SO_4)	A/O	0 kg/t	0.0	g
Total feed weight			400.0	g
Cure Time - acidic paste			0	h
Color of paste:			brown	
Roast time:			2	h
Roast Temperature:			750	°C

Calcination Results:

Calcine Net wt.	385.3	g
Color:	red brown	
Consistancy	powdery	
Pulverized discharge (if necessary)	n/a	
Time	-	min

Samples for assay:

Final calcine solids : N/A

Acid Bake Comments:

Water Leach Conditions:

Water Used to Wash Crucible (mL):	0	
Pyrex Reactor Tare (g):	2623.42	
Leach Feed - Acid Bake Product (g):	384.51	(as received)
Calculated Acid Bake Feed wt (g):	384.5	
DI Water to add (mL):	3461	**Total DI Added: 3461**
Pulp Density (%solids)	10.0	w/w Leach Feed (Acid Bake Product)
Pulp Density (%solids)	10.0	w/w Calculated Acid Bake Feed Should be: 10 w/w%
Time (h):	4	
Temperature (°C):	90	(temperature controlled with heating mantle)
Net Mass of slurry transferred to 4L Reactor		g
Initial gross mass		g

Leach Data:

Elapsed Time hours	Temp °C	pH	ORP	Pulp colour	~~Kinetic Sample~~ ~~Pulp wt. g~~	~~PLS wt. g~~	~~PLS SG~~	Comments
0	87.6	9.22	-77	red brown				t=0 solids added
1	88.5	8.88	-51	red brown				
2	87.9	8.72	-29	red brown				
3	89.3	8.64	-22	red brown				
4	89.7	8.61	-21	red brown				end test

Results:

Final Pulp wt (g):	3776.3
Filtrate Collected wt (g):	3076.5
Filtrate SG:	0.9957
Filtrate Collected Vol (mL):	3089.8
Filtrate Calculated Vol (mL):	3416.8
Filtrate Free Acid (H2SO4 g/L):	0.0
Wash Solution (mL):	1471
Solid Residue: Tare (g):	0
Wet wt (g):	559
Dry wt (g):	374.21
Net Dry Wt (g)	374.21

Notes:

Filter Paper:	#3, 150 mm	
PLS Filtration Time:	3 min	hrs
Repulp + Displacements:	10 min	hrs
Filtrate Colour:	clear	
Residue Colour:	red-brown	
Wash Wt.:	1467.3	g
Wash SG:	0.9978	g/mL

Filtrate Free Acid	0.00
pH	9.48
ORP	241

Assays:

Combined Wash Liquor: Ce, Y, Fe
Final Liquor: REE, Si, Fe, FAT
Final Solids: REE (ICP-MS), S, S=

Metallurgical Balance:

	Assay, mg/L or %				Recovery	Out/In	Distribution %		
	Feed	4h PLS	Wash	Residue	%	%	4h PLS	Wash	Residue
g or mL	385	3417	1471	374					
La	0.155	<0.03		0.172	0.0	108	0		100
Ce	0.258	<0.03	<0.03	0.313	0.0	118	0		100
Pr	0.029	<0.03		0.032	0.1	108	0		100
Nd	0.090	<0.06		0.107	0.1	115	0		100
Sm	0.015	<0.04		0.017	0.2	115	0		100
Eu	0.001	<0.03		0.001	3.2	129	3		97
Gd	0.008	<0.03		0.012	0.2	132	0		100
Tb	0.001	<0.03		0.001	2.0	110	2		98
Dy	0.006	<0.05		0.006	0.8	101	1		99
Ho	0.001	<0.02		0.001	2.0	105	2		98
Er	0.002	<0.04		0.002	1.7	100	2		98
Tm	0.000	<0.04		0.000	11.2	122	11		89
Yb	0.002	<0.02		0.002	1.1	110	1		99
Lu	0.000	<0.03		0.000	10.6	125	11		89
Y	0.020	<0.01	<0.01	0.022	0.0	106	0		100
Sc	0.003	<0.07		<0.00	2.5	100	2		98
U	0.081	<1		0.101	0.8	122	1		99
Th	0.114	<0.03		0.131	0.0	112	0		100
Fe	8.95			9.7	#DIV/0!	0	#DIV/0!		#DIV/0!
Si	25.66				#DIV/0!	0	#DIV/0!		#DIV/0!

Acid Consumption:

Acid In:	0.0	g
Free Acid Out:	0.0	g
Acid Consumption:	0.0	kg H2SO4 / t Ore

Weight Loss: 3%

TREE:	0%	WL Residue S:	0.52	%Stot
LREE:	0%		0.17	%S^{2-}
HREE:	1%	%S Removed:	94%	

Project:	13849-001	**Date:**	2-Apr-13
Client:	Appia Energy	**Technologist:**	Krystal Davis

Test: AB-5

Purpose: To perform an acid bake and water leach on Appia roasted concentrate.

Sample: Roast-WL Res ~200 g dry

Grind:
K80= µm

Procedure:

1. ~~The conc. was ground at 50% solids in a lab rod mill for the indicated time. The mill discharge was filtered.~~
2. The filtered ground ore res was combined with the specified amount of sulphuric acid in a crucible (record tare weight of crucible) inside a fumehood. The paste was made homogeneous.
3. The pasted was cured for 1 hour
4. The muffle furnace was heated to 250ºC
5. After curing, the ore/acid mixture was placed in a muffle furnace at temperature and baked for 3 hours.
6. After 3 hours at temperature, the furnace was turned off and allowed to cool to room temperature.
7. Weighed calcined sample in crucible, transfered calcine to mortar and pulverized if necessary.
8. Acid Baked solids were pulped to 10% solids in DI water in a 2L Pyrex reaction kettle.
9. A lid and condenser was attached to the kettle and a Teflon impellor/mechanical mixer setup was used to agitate the slurry. The slurry was heated to 90°C via heating mantle/temperature controller setup. pH was maintained below 1.5
11. After 4 hours at temperature, the pulp was weighed, filtered and the filtrate was collected. The solids were repulped in DI water, filtered, and displacement washed with 3 x 300 mL DI water.
12. Final solids were dried to constant mass. Final PLS, wash and soilds were assayed per the table below
13. Free acid and sg of the Final Solution was determined.

Conditions:

Stage 1

Crucible Tare Weight:			769.1	g
% Moisture of Feed:			0.0	%
Wet Weight Feed Added:			166.0	g
Dry Weight of Calcine.:			166.0	g
Sulphuric Acid (96% H_2SO_4)	A/O	300 kg/t	51.9	g
Total feed weight			217.9	g
Cure Time - acidic paste			1	h
Color of paste:			red brown	
Bake time:			3	h
Acid Bake Temperature:			250	°C

Calcination Results:

Calcine Net wt.	204.3	g
Color:	grey brown	
Consistancy	powdery	
Pulverized discharge (if necessary)		
Time		min

Samples for assay:

Final calcine solids :

Acid Bake Comments:

Water Leach Conditions:

Water Used to Wash Crucible (mL):		
Pyrex Reactor Tare (g):	2377.86	
Leach Feed - Acid Bake Product (g):	204.25	(as received)
Calculated Acid Bake Feed wt (g):	166.0	
DI Water to add (mL):	1494	**Total DI Added: 1494**
Pulp Density (%solids)	12.0	w/w Leach Feed (Acid Bake Product)
Pulp Density (%solids)	10.0	w/w Calculated Acid Bake Feed Should be: 10 w/w%
Time (h):	4	
Temperature (°C):	90	(temperature controlled with heating mantle)
Net Mass of slurry transferred to 4L Reactor		g
Initial gross mass		g

Leach Data:

Elapsed Time hours	Temp °C	pH	ORP	Pulp colour	~~Kinetic Sample~~			Comments
					~~Pulp wt. g~~	~~PLS wt. g~~	~~PLS SG~~	H_2SO_4 added, g (to maintain pH < 1.5)
0	79.5	1.71	721	brown				2.0 g H2SO4 added, t=0
1	82.6	1.47	629	brown				
2				brown				
3	81.5	1.26	654	brown				
4	81.5	1.22	662	brown				end test

Results:

	Final Pulp wt (g):	3084.8
	Filtrate Collected wt (g):	2794.9
	Filtrate SG:	1.0109
	Filtrate Collected Vol (mL):	2764.8
	Filtrate Calculated Vol (mL):	2897.1
	Filtrate Free Acid (H2SO4 g/L):	5.1
	Wash Solution (mL):	795
Solid Residue:	Tare (g):	12
	Wet wt (g):	251.7
	Dry wt (g):	168.1
	Net Dry Wt (g)	156.1

Notes:

Filter Paper:	#3, 150 mm	
PLS Filtration Time:	5min	hrs
Repulp + Displacements:	15min	hrs
Filtrate Colour:	pale yellow	
Residue Colour:	brown	
Wash Wt.:	791.8	g
Wash SG:	0.9956	g/mL

Filtrate Free Acid	5.10
pH	1.60
ORP	580

Assays:

Combined Wash Liquor: Ce, Y, Fe
Final Liquor: REE, ICP, Si, FAT
Final Solids: REE (ICP-MS), S

Metallurgical Balance:

	Assay, mg/L or %				Recovery	Out/In	Distribution %		
	Feed	4h PLS	Wash	Residue	%	%	4h PLS	Wash	Residue
g or mL	166	2897	795	156					
La	0.172	50.2		0.104	47.3	108	47		53
Ce	0.313	92.9	12.3	0.185	48.2	107	48		52
Pr	0.032	10.4		0.017	52.9	107	53		47
Nd	0.107	34.4		0.053	54.6	103	55		45
Sm	0.017	5.59		0.008	56.2	101	56		44
Eu	0.001	0.26		0.000	50.7	109	51		49
Gd	0.012	4.29		0.005	61.4	106	61		39
Tb	0.001	0.5		0.001	64.5	102	65		35
Dy	0.006	2.4		0.002	67.3	108	67		33
Ho	0.001	0.36		0.000	65.0	107	65		35
Er	0.002	0.9		0.001	67.9	111	68		32
Tm	0.000	0.11		0.000	56.1	118	56		44
Yb	0.002	0.62		0.001	62.2	104	62		38
Lu	0.000	0.08		0.000	51.5	118	51		49
Y	0.022	8.82	1.33	0.009	65.8	107	66		34
Sc	0.003	0.12		<0.00	8.2	102	8		92
U	0.101	47.1		0.023	79.2	103	79		21
Th	0.131	23.8		0.093	32.2	98	32		68
Fe	0.00	930	105	9.7	15.1	#DIV/0!	15		85
Si	0.00	73.1			100.0	#DIV/0!	100		0

Acid Consumption:

Acid In:	49.8	g
Free Acid Out:	14.8	g
Acid Consumption:	211.0	kg H2SO4 / t Calcine
Acid to Water Leach	11.6	kg H2SO4 / t Calcine

Weight Loss: 6%

Extractions:

TREE:	50%	U:	79%
LREE:	49%	Th:	32%
HREE:	64%		

Residue 1.48 %S

1.15 $\%S^{2-}$ (these don't match WL residue assays)

Project No.: **13849-001** **Project Technologist:** Krystal Davis
Client: **Appia Energy Corp** **Date of Test:** 03-Apr-13

Test ID: **AL-6** **~200 g**

Objective: To conduct a sulphuric acid leach test on a uranium ore
To study the kinetics of uranium dissolution under various conditions

Feed: Roast-WL Res
Grind: -- 50% solids in lab ball mill
(note time/kg, weight charged, % solids in mill and mill size/type)

Procedure:

1. The sample was ground to the desired P80 and the mill discharge pan filtered
2. A small sample was cut for %H2O determination (~50 g)
3. The wet wt. of feed was recorded along with the kettle tare weight, slurry the solids and agitate kettle
4. The Slurry was brought to desired temperature (if required)
5. The pH or FA was adjusted to target level with H2SO4; all acid additions were recorded (beware of foaming, add acid slowly)
7. Small amounts of 200 g/L NaClO3 were added to achieve ORP target
8. Records of pH, emf, Temp were kept throughout the test, H2SO4 was added to keep constant pH (with acid requirement recorded)
9. The reactor contents were sampled at 12, 24, 48 hrs using 60 mL syringe
10. The pulp sample weight was recorded, then filtered
11. Solids were washed with 60 mL pH 2.0 water, then 60 mL DI water (DO NOT LET CAKE CRACK or repulp)
12. The filtrate weight and SG was recorded, along w/ pH, FA and ORP if sufficient volume permitted
13. Dry solids and filtrate were submitted for analysis, the wash solution was rejected
14. After completion of the test the reactor was allowed to cool (if at T) and agitation was stopped
15. The final weight of vessel was recorded and the contents filtered and washed once with 300 mL pH 2.0 water, 3 times with 300 mL DI
16. The filtration rate was recorded along with notes on residue and liquor colour
17. The filtrate weight and SG was recorded, along w/ pH, FA and ORP
18. Dry solids and filtrate were submitted for analysis, the wash solution was rejected

All pulp samples must be labeled "NORM" for Naturally Occurring Radioactive Material

Conditions:

Reactor Tare (incl. lid and baffles):	1135.6	g
Wet Feed Weight:	166.0	g
Moisture:	0%	
Dry Feed Weight:	166.0	g
Target Slurry Density:	40	w/w%
Water added:	249.0	mL
Water in Feed:	0.0	mL
Target FA:	50	g/L H_2SO_4
Oxidant Type:	200 g/L NaClO3	
Target ORP:	500	mV *(vs. Ag|AgCl sat'd KCl electrode)*
Fe3+	2	g/L
Grind		µm
Temperature:	80	ºC

Analytical Requirements:

Kinetic solutions	REE, Fe, Fe2+, SO4
Kinetic residues	REE, S(tot)
Final PLS	REE, Fe2+, ICP, SO4
Final residue	REE, Fe, Na, K, S(tot)

Test Details:

Elapsed Time (hh:mm)	(hrs)	Temp (°C)	pH	ORP (mV)	FA (g/L H_2SO_4)	Reagents Added H_2SO_4 (g)	Reagents Added 200 g/L NaClO3 (mL)	Comments
8:44	-	21.3	7.01	120		13.50		Initial acid addition, heating to 80˚C
8:46	-	31.6	0.84	1159				2.5g $Fe_2(SO_4)_3$ added
9:00	0.0	87.0	0.59	697	40.7			t = 0
10:00	1.0	82.4	0.46	712		2.30		
12:45		81.1	0.38	728	50.01			
19:46	12.0	84.0	0.30	736	46.5	2.60		12h sample
8:30		77.4	0.58	714	33.44	4.20		23g DI added
9:00	24.0	75.5	0.35	718				24h sample
15:07		79.2	0.38	728				89.2g Di added, probes removed to avoid evap
15:45					43.05	2.10		FA check
7:45		24.1	0.73	568	47.66			Test was off, GFI tripped? 73g DI added, reheating
8:05		65.8	0.40	691				
9:00	48.0	81.7	0.36	722				End Test
TWA						24.7	0	mL or g
						142.8	0.0	kg/t

Sample	Pulp Weight g	PLS Filtrate g	Wash Filtrate g	Wet Residue (g)	Dry Residue (g)	Pulp Density w/w	Cake Moisture (%)	PLS Filtrate pH	PLS Filtrate ORP¹ (mV)	PLS Filtrate SG	PLS Filtrate Free Acid (g/L)
12 hr	49.1	17.3	168.8	21.0	21.1	43.0%	-1			1.055	47
24 hr	47.0	19.1	193.2	27.2	17.8	37.9%	36			1.085	56
Final	280.7	89.8	813.1	152.8	113.4	40.4%	28	1.17	546	1.104	59
							TWA:	**1.2**	**546.0**		**55.3**

² Assuming SG of pore water in washed cake is 1.00.

Project No.:	**13849-001**	**Project Technologist:**	Krystal Davis
Client:	**Appia Energy Corp**	**Date of Test:**	03-Apr-13

Test ID: **AL-6** **~200 g**

Objective: To conduct a sulphuric acid leach test on a uranium ore
To study the kinetics of uranium dissolution under various conditions

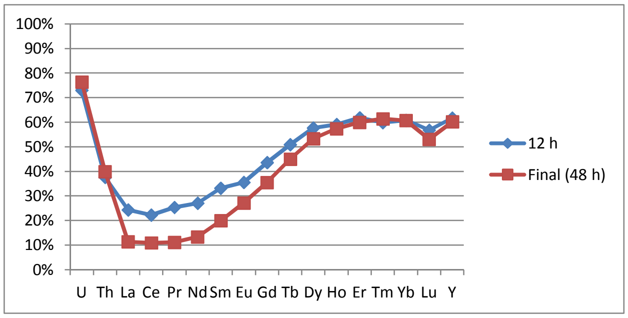
Notes:

Filtration: Whatman 3, 185 — enter filter paper type, size — Filtrate Colour:
N/A — enter final filtration time (time to no liquid on cake) — Residue Colour:

Metallurgical Balance

	Assays Head, g/t	PLS, mg/L 12 h	24 h	Final	Residue, g/t 12 h	24 h	Final	Calc'd Head	Balance OUT/IN %
Amt, mL or g	166	27	27	152	21.1	17.8	113.4		
U	1010	611	487	541	284	238	224	885	88%
Th	1310	378	292	341	790	744	688	1069	82%
Fe		5350	4970	9220					
Fe++		59	87	97					
La	1720	363	249	151	1420	1410	1580	1647	96%
Ce	3130	585	425	253	2570	2550	2760	2879	92%
Pr	322	69.3	47.7	26.9	257	256	287	300	93%
Nd	1070	236	167	103	800	811	895	959	90%
Sm	172	47.7	33.3	24.8	121	122	133	155	90%
Eu	8.2	2.59	1.95	1.62	5.9	5.3	5.8	7.5	91%
Gd	115	39.8	29.7	27.4	64.7	66.2	66.6	97	84%
Tb	13.3	5.52	4.16	4.04	6.7	6.4	6.6	11.3	85%
Dy	57.6	28.5	21.6	22	26.3	25.9	25.8	52	90%
Ho	9	4.6	3.53	3.62	4	3.8	3.6	8.0	89%
Er	20.9	11.7	8.8	9.19	9.1	9.1	8.2	19.4	93%
Tm	2.9	1.42	1.13	1.19	1.2	1.2	1	2.5	85%
Yb	16.8	8.37	6.47	7.16	6.8	6.8	6.2	14.8	88%
Lu	2.3	1.15	0.84	0.93	1.1	1.2	1.1	2.2	95%
Y	218	118	89.5	94.9	92	94	84	199.2	91%
S/SO_4				100000					
Na				123					
K				4840					

Element / Sample	Extraction, % 12 h	24 h	Final
U	73%	76%	76%
Th	38%	37%	40%
La	24%	21%	11%
Ce	22%	20%	11%
Pr	25%	22%	11%
Nd	27%	24%	13%
Sm	33%	29%	20%
Eu	36%	36%	27%
Gd	44%	40%	35%
Tb	51%	50%	45%
Dy	58%	56%	53%
Ho	59%	58%	57%
Er	62%	59%	60%
Tm	60%	59%	61%
Yb	61%	59%	61%
Lu	57%	51%	53%
Y	62%	59%	60%
S	#DIV/0!	#DIV/0!	100%
Na	--	--	100%
K	--	--	100%



Acid Consumption:

Acid In:	23.7	g
Free Acid Out:	11.3	g
Acid Consumption:	74.5	kg H_2SO_4 / t Ore

Weight Loss: 8%

Extractions (PLS vs. Calc'd H

TREE:	14%
LREE:	12%
HREE:	52%

Project No.:	**13849-001**	**Project Technologist:**	Krystal Davis
Client:	**Appia Energy Corp**	**Date of Test:**	08-Apr-13

Test ID: **AL-7** **200g**

Objective: To conduct a sulphuric acid preleach test on a 9% mass pull Appia Concentrate
To produced preleach residue for an acid bake test

Feed: Conc 2 (F19, F20, F21 Combined Concentrate)
Grind: N/A 50% solids in lab ball mill
(note time/kg, weight charged, % solids in mill and mill size/type)

Procedure:

1. ~~The sample was ground to the desired P80 and the mill discharge pan filtered~~
2. ~~A small sample was cut for %H2O determination (~50 g)~~
3. The wet wt. of feed was recorded along with the kettle tare weight, slurry the solids and agitate kettle
4. The Slurry was brought to desired temperature (if required)
5. The pH or FA was adjusted to target level with H_2SO_4; all acid additions were recorded (beware of foaming, add acid slowly)
7. Small amounts of 200 g/L NaClO3 were added to achieve ORP target
8. Records of pH, emf, Temp were kept throughout the test, H_2SO_4 was added to keep constant pH (with acid requirement recorded)
9. The reactor contents were sampled at 6 hrs using 60 mL syringe
10. The pulp sample weight was recorded, then filtered
11. Solids were washed with 60 mL pH 2.0 water, then 60 mL DI water (DO NOT LET CAKE CRACK or repulp)
12. The filtrate weight and SG was recorded, along w/ pH, FA and ORP if sufficient volume permitted
13. Dry solids and filtrate were submitted for analysis, the wash solution was rejected
14. After completion of the test the reactor was allowed to cool (if at T) and agitation was stopped
15. The final weight of vessel was recorded and the contents filtered
16. The filtration rate was recorded along with notes on residue and liquor colour
17. The filtrate weight and SG was recorded, along w/ pH, FA and ORP
18. Store solids as a wet cake

All pulp samples must be labeled "NORM" for Naturally Occurring Radioactive Material

Conditions:

Reactor Tare (incl. lid and baffles):	1886.15	g
Wet Feed Weight:	220.1	g
Moisture:	9%	
Dry Feed Weight:	199.6	g
Target Slurry Density:	30	w/w%
Water added:	445.3	mL
Water in Feed:	20.5	mL
Target FA:	15	g/L H_2SO_4
Oxidant Type:	200 g/L NaClO3	
Target ORP:	500	mV *(vs. Ag|AgCl sat'd KCl electrode)*
Fe3+		g/L
Grind	~95	µm
Temperature:	80	ºC

Analytical Requirements:

Kinetic solutions	
Kinetic residues	
Final PLS	REE, Fe2+, ICP
Final residue	

Test Details:

Elapsed Time (hh:mm)	(hrs)	Temp (°C)	pH	ORP (mV)	FA (g/L H_2SO_4)	Reagents Added H_2SO_4 (g)	Reagents Added 200 g/L NaClO3 (mL)	Comments
8:50		21.0	5.74	328		7.50		acid addition
8:52		24.0	2.06	487				heating
9:00	0.0	80.0	4.06	10				time 0
9:50		88.0	4.52	-6		7.50		acid addition
9:53	1.0	90.0	1.54	139				
10:23		91.9	1.77	384	4.4	5.10		FA check, adid and chlorate additions
10:32		89.3	1.21	421			0.90	
11:30		76.1	1.21	510	13.7	0.70		FA check
11:55	3.0	79.9	1.24	510	15.6			
13:05	4.0	80.4	1.23	481	14.5	1.00		
14:15	5.0	78.5	1.27	499				
15:00	6.0	82.6	1.31	502				
TWA						21.8	0.9	mL or g
						104.8	0.9	kg/t

Sample	Pulp Weight	PLS Filtrate	Wash Filtrate	Wet Residue	Dry Residue	Pulp Density	Cake Moisture	PLS Filtrate pH	PLS Filtrate ORP[1]	PLS Filtrate SG	PLS Filtrate Free Acid
	g	g	g	(g)	(g)	w/w	(%)		(mV)		(g/L)

Project No.: **13849-001** **Project Technologist:** Krystal Davis
Client: **Appia Energy Corp** **Date of Test:** 08-Apr-13

Test ID: **AL-7** **200g**

Objective: To conduct a sulphuric acid preleach test on a 9% mass pull Appia Concentrate
To produced preleach residue for an acid bake test

Final	620.2	361.35	n/a	200.9	n/a	#VALUE!	#VALUE!	2.16	410	1.026	14
							TWA:				

[1](ORP measured against Ag|AgCl sat'd KCl electrode)
[2] Assuming SG of pore water in washed cake is 1.00.

Notes: Wet, unwashed solids proceeded to test AB6

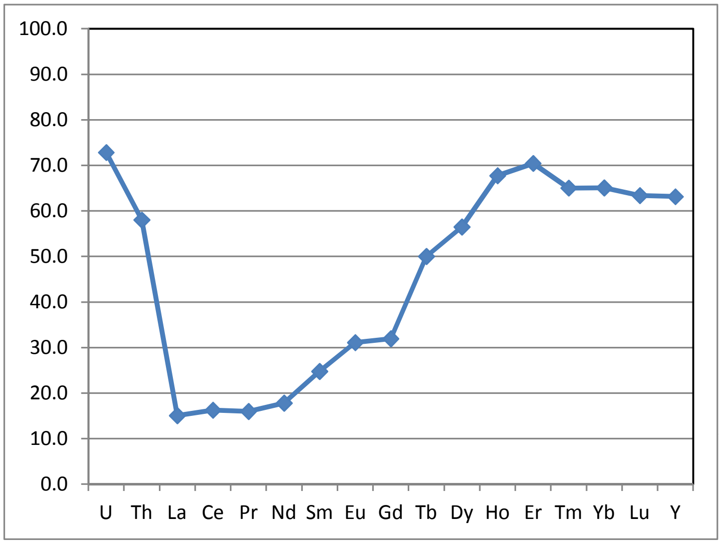
Filtration:

Whatman #3, 150mm		enter filter paper type, size	Filtrate Colour:	yellow-green
5min	min	enter final filtration time (time to no liquid on cake)	Residue Colour:	grey

Metallurgical Balance:

	Assays		Metal Units		
Product	Head g, g/t	Filtrate mL, mg/L	Head mg	Filtrate mg	Filtrate %
Amt, g, mL	199.6	352			
U	780	322	156	113	72.8
Th	1700	559	339	197	58.0
Fe	112600	3320	22475	1169	
Fe++		2480			
La	3500	299	699	105	15.1
Ce	6300	581	1258	205	16.3
Pr	700	63.4	140	22	16.0
Nd	2200	222	439	78	17.8
Sm	299	42	60	15	24.8
Eu	13.1	2.31	3	1	31.1
Gd	190	34.4	38	12	31.9
Tb	18.7	5.3	4	2	50.0
Dy	75.9	24.3	15	9	56.5
Ho	11.2	4.3	2	2	67.7
Er	26.3	10.5	5	4	70.4
Tm	3.5	1.29	1	0	65.0
Yb	20.5	7.56	4	3	65.1
Lu	2.7	0.97	1	0	63.4
Y	285	102	57	36	63.1

Distribution to Filtrate by Element:



excludes metal units in unwashed cake

Project: 13849-001
Client: Appia Energy
Date: 9-Apr-13
Technologist: Sarah Power
Victoria Vanderbyl

Test: AB-6

Purpose: To perform an acid bake and water leach on an 8-9% Appia Concentrate Preleach Residue

Sample: AL-7 Unwashed Wet Cake Residue ~200 g dry
Grind:
K80= µm

Procedure:

1. The filtered unwashed wet cake from AL-7 was combined with the specified amount of sulphuric acid in a crucible (record tare weight of crucible) inside a fumehood. The paste was made homogeneous.
2. The paste was cured for 1 hour
3. The muffle furnace was heated to 250ºC
4. After curing, the ore/acid mixture was placed in a muffle furnace at temperature and baked for 3 hours.
5. After 3 hours at temperature, the furnace was turned off and allowed to cool to room temperature.
6. Weighed calcined sample in crucible, transfered calcine to mortar and pulverized if necessary.
7. Acid Baked solids were pulped to 10% solids in DI water in a 2L Pyrex reaction kettle.
8. A lid and condenser was attached to the kettle and a Teflon impellor/mechanical mixer setup was used to agitate the slurry. The slurry was heated to 90°C via heating mantle/temperature controller setup. pH was maintained below 1.5
9. After 4 hours at temperature, the pulp was weighed, filtered. The solids were washed with 60mL of DI water, the PLS and first wash were stored and a sample sent for assay. The cake was then repulped once and displacement washed with 3 x 300 mL DI water.
10. Final solids were dried to constant mass. Final PLS, wash and soilds were assayed per the table below
11. Free acid and sg of the Final Solution was determined.

Conditions:
Stage 1

Crucible Tare Weight:			769.0	g
% Moisture of Feed:				%
Wet Weight Feed Added:				g
Assumed Dry Weight:			194.5	g
Sulphuric Acid (96% H_2SO_4)	A/O	600 kg/t	121.6	g
Total feed weight			316.1	g
Cure Time - acidic paste			1	h
Color of paste:				
Bake time:			3	h
Acid Bake Temperature:			250	°C

Calcination Results:

Calcine Net wt.	214.6	g
Color:	grey brown	
Consistancy		
Pulverized discharge (if necessary)		
Time		min

Samples for assay:

Final calcine solids :

Acid Bake Comments:

Project: 13849-001
Client: Appia Energy

Date: 9-Apr-13
Technologist: Sarah Power
Victoria Vanderbyl

Test: AB-6

Purpose: To perform an acid bake and water leach on an 8-9% Appia Concentrate Preleach Residue

Water Leach Conditions:

Water Used to Wash Crucible (mL):		
Pyrex Reactor Tare (g):	2573.9	
Leach Feed - Acid Bake Product (g):	214.6	(as received)
Calculated Acid Bake Feed wt (g):	194.5	
DI Water to add (mL):	1751	**Total DI Added: 1751**
Pulp Density (%solids)	10.9	w/w Leach Feed (Acid Bake Product)
Pulp Density (%solids)	10.0	w/w Calculated Acid Bake Feed Should be: 10 w/w%
Time (h):	4	
Temperature (°C):	90	(temperature controlled with heating mantle)
Net Mass of slurry transferred to 4L Reactor		g
Initial gross mass		g

Leach Data:

Elapsed Time hours	Temp °C	pH	ORP	Pulp colour	~~Kinetic Sample~~			Comments
					~~Pulp wt, g~~	~~PLS wt, g~~	~~PLS SG~~	H_2SO_4 added, g (to maintain pH < 1.5)
0	89.1	1.97	551	grey				brought pulp to pH 1.44 with 18.3g
1	90.1	1.43	554	grey				
2	90.0	1.40	539	grey				
3	90.0	1.41	533	grey				
4	90.7	1.41	528	grey				

Results:

Final Pulp wt (g):	2981.3
Filtrate Collected wt (g):	2815.1
Filtrate SG:	1.0314
Filtrate Collected Vol (mL):	2729.4
Filtrate Calculated Vol (mL):	2776.7
Filtrate Free Acid (H2SO4 g/L):	9.4
Wash Solution (mL):	1276
Solid Residue: Tare (g):	
Wet wt (g):	
Dry wt (g):	
Net Dry Wt (g)	117.4

Notes:

Filter Paper:	#3, 185 mm	
PLS Filtration Time:	7	min
Repulp + Displacements:	22	min
Filtrate Colour:	yellow/green	
Residue Colour:	brown/grey	
Wash Wt.:	1277	g
Wash SG:	1.0005	g/mL

Filtrate Free Acid	9.36
pH	1.40
ORP	433

Assays:

Combined Wash Liquor: Store for AL-8
Final Liquor and First Wash: REE, ICP
Final Solids: REE (ICP-MS),Fe

Project: 13849-001
Client: Appia Energy

Date: 9-Apr-13
Technologist: Sarah Power
Victoria Vanderbyl

Test: AB-6

Purpose: To perform an acid bake and water leach on an 8-9% Appia Concentrate Preleach Residue

Metallurgical Balance:

	Assay, mg/L or g/t (Fe in %)				Recovery	Out/In	Distribution %		
	Feed	4h PLS	Wash	Residue	%	%	4h PLS	Wash	Residue
g or mL	195	2777	1276	117					
La	3050	170		325	92.5	86	93		7
Ce	5413	295		436	94.1	83	94		6
Pr	604	29.8		43	94.3	75	94		6
Nd	1856	97		114	95.3	78	95		5
Sm	231	13		22	93.4	86	93		7
Eu	9	0.47		2	88.1	82	88		12
Gd	133	6.21		13	91.8	73	92		8
Tb	10	0.5		2	88.7	84	89		11
Dy	34	1.66		6	86.2	81	86		14
Ho	4	0.2		1	84.0	92	84		16
Er	8	0.43		2	81.6	94	82		18
Tm	1	0.05		0	79.8	71	80		20
Yb	7	0.33		1	84.8	76	85		15
Lu	1	0.04		1	65.4	86	65		35
Y	108	5.44		17	88.3	82	88		12
Sc		0.23		25	17.9	#DIV/0!	18		82
U	217	6.3		12	92.8	45	93		7
Th	732	28.5		189	78.1	71	78		22
P		145			100.0	#DIV/0!	100		0
Fe	11.0	3410		13.3	37.7	118	38		62
Ca		609			100.0	#DIV/0!	100		0
Al		1900			100.0	#DIV/0!	100		0
Si					#DIV/0!	#DIV/0!	#DIV/0!		#DIV/0!

Acid Consumption:

Acid In:	116.7	g
Free Acid Out:	26.0	g
Acid Consumption:	466.4	kg H2SO4 / t Ore
	133.6	kg/t acid in WL PLS

Weight Loss: 40%

TREE: 94%
LREE: 94%
HREE: 89%

Project No.: **13849-001**
Client: **Appia Energy Corp**

Project Technologist:
Date of Test:

Test ID: **AL-8** **200g**

Objective: To conduct a sulphuric acid leach test on a uranium ore
To study the kinetics of uranium dissolution under various conditions

Feed: AB6-WL
Grind: -- 50% solids in lab ball mill
(note time/kg, weight charged, % solids in mill and mill size/type)

Procedure:

1. ~~The sample was ground to the desired P80 and the mill discharge pan filtered~~
2. ~~A small sample was cut for %H20 determination (~50 g)~~
3. The wet wt. of feed was recorded along with the kettle tare weight, slurry the solids and agitate kettle
4. The Slurry was brought to desired temperature (if required)
5. The pH or FA was adjusted to target level with H_2SO_4; all acid additions were recorded (beware of foaming, add acid slowly)
7. Small amounts of 200 g/L NaClO3 were added to achieve ORP target
8. Records of pH, emf, Temp were kept throughout the test, H_2SO_4 was added to keep constant pH (with acid requirement recorded)
9. ~~The reactor contents were sampled at 6 hrs using 60 mL syringe~~
10. ~~The pulp sample weight was recorded, then filtered~~
11. ~~Solids were washed with 60 mL pH 2.0 water, then 60 mL DI water (DO NOT LET CAKE CRACK or repulp)~~
12. ~~The filtrate weight and SG was recorded, along w/ pH, FA and ORP if sufficient volume permitted~~
13. ~~Filtrate was submitted for analysis, the wash solution was stored~~
14. After completion of the test the reactor was allowed to cool (if at T) and agitation was stopped
15. The final weight of vessel was recorded and the contents filtered and not washed
16. The filtration rate was recorded along with notes on residue and liquor colour
17. The filtrate weight and SG was recorded, along w/ pH, FA and ORP
18. Wet cake solids were stored and filtrate was submitted for analysis

All pulp samples must be labeled "NORM" for Naturally Occurring Radioactive Material

Conditions:

Reactor Tare (incl. lid and baffles):	1880.6	g
Wet Feed Weight:	220.0	g
Moisture:	9%	
Dry Feed Weight:	199.5	g
Target Slurry Density:	30	w/w%
AB-6 PLS	445.1	mL
Water in Feed:	20.5	mL
Target FA:	pH 1.6	g/L H_2SO_4
Oxidant Type:	200 g/L NaClO3	
Target ORP:	500	mV *(vs. Ag|AgCl sat'd KCl electrode)*
Fe3+		g/L
Grind	~95	µm
Temperature:	80	ºC

Analytical Requirements:

Kinetic solutions	
Kinetic residues	
Final PLS	REE, Fe2+, ICP
Final residue	Store

Test Details:

Elapsed Time (hh:mm)	(hrs)	Temp (°C)	pH	ORP (mV)	FA (g/L H_2SO_4)	Reagents Added H_2SO_4 (g)	Reagents Added 200 g/L NaClO3 (mL)	Comments
7:52		20.7	3.67	364		3.70		
8:36		75.9	1.61	463		1.70		
9:05	0.0	80.0	1.62	454				
9:39		82.8	2.02	382		2.00	0.60	
11:22		81.2	1.53	470			1.70	
11:32		81.6	1.39	512				
14:11		79.0	1.15	460			0.50	
14:15		85.1	1.29	508				
14:48		81.2	1.20	482				
15:05	6.0	81.7	1.20	477				end test
TWA						7.4	2.8	mL or g
						35.6	2.8	kg/t

Project No.: **13849-001** **Project Technologist:**
Client: **Appia Energy Corp** **Date of Test:**

Test ID: **AL-8** **200g**

Objective: To conduct a sulphuric acid leach test on a uranium ore
To study the kinetics of uranium dissolution under various conditions

	Pulp Weight	PLS Filtrate	Wash Filtrate	Wet Residue	Dry Residue	Pulp Density	Cake Moisture	PLS Filtrate			
								pH	ORP[1]	SG	Free Acid
Sample	g	g	g	(g)	(g)	w/w	(%)		(mV)		(g/L)
Final	662.4	384.4	n/a	240.0	199.5	30.1%	17	1.77	388	1.026	5
							Avg				

[1](ORP measured against Ag|AgCl sat'd KCl electrode)
[2] Assuming SG of pore water in washed cake is 1.00.

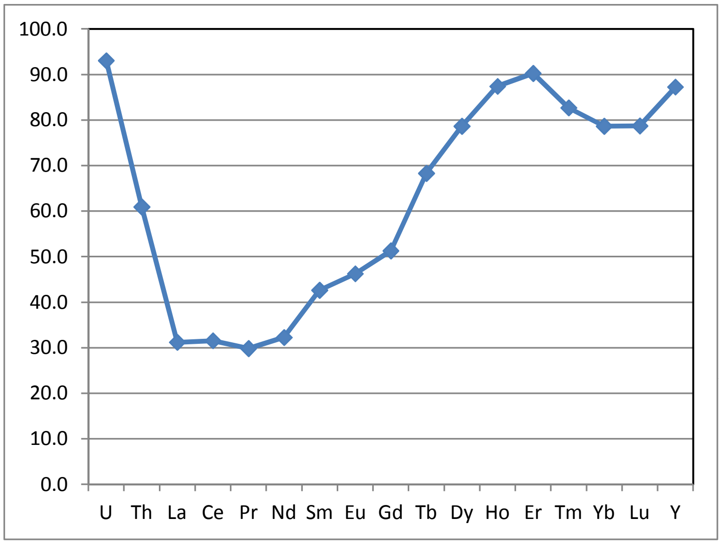
Notes:

Filtration: enter filter paper type, size Filtrate Colour:
min enter final filtration time (time to no liquid on cake) Residue Colour:

Metallurgical Balance:

Product	Assays		Metal Units		Filtrate %
	Head g, g/t	Filtrate mL, mg/L	Head mg	Filtrate mg	
Amt, g, mL	199.5	451			
U	780	321	156	145	93.0
Th	1700	458	339	207	60.9
Fe		1470			
Fe++		1250			
La	3500	483	698	218	31.2
Ce	6300	879	1257	397	31.5
Pr	700	92.4	140	42	29.8
Nd	2200	314	439	142	32.3
Sm	299	56.4	60	25	42.6
Eu	13.1	2.68	3	1	46.3
Gd	190	43.1	38	19	51.3
Tb	18.7	5.65	4	3	68.3
Dy	75.9	26.4	15	12	78.6
Ho	11.2	4.33	2	2	87.4
Er	26.3	10.5	5	5	90.3
Tm	3.5	1.28	1	1	82.7
Yb	20.5	7.13	4	3	78.6
Lu	2.7	0.94	1	0	78.7
Y	285	110	57	50	87.3

Distribution to Filtrate by Element:



Appendix B – Neutralization & IX Test Details

Project: 13849-001
Client: Appia Energy
Test: **Neut-1**

Project Technologist: Krystal Davis
Date of Test: 30-Apr-13

Objective: To prepare solution for IX test work.

Feed: **Combined PLS**

H&S: Refer to H_2SO_4, MgO, MSDS's
Feed solution is corrosive, avoid contact with skin and clean all spills accordingly

Procedure:

1. 600 mL of Solution was placed in a 1000 mL Pyrex reactor equipped with at least 1 baffle. The solution was agitated with a Teflon impellor and mechanical mixer. The pulp was agitated at ambient temperature.
2. Limestone slurry (20 % solids) was added stepwise to get to pH 2.0 and held for 1 hour. The pulp was then filtered and a drop H2SO4 was added to the filtrate.
3. A liquor sample was taken for U, Nd, Y, Fe and the solids sampled Fe, U, Nd, Y, Ca, and S.
4. Reagent additions were accurately recorded. (Reagent added slowly to mixing vortex).
5. Assay all liquors for REE, Fe, Na. Assay final solids for REE, Fe.

Test Observations:

Time hh:mm	Time min	Temp °C	pH	emf mV	Limestone Pulp Wt. Reading (20% solids), g	Limestone (20%) added, g	Notes
14:15			1.13		469.1		
14:25			2		380.7	88.4	
15:25			2.15				
						Sum	
						88.4	

Sampling INFO

			Sample					Sample		
Sample #	pH Target	PLS, g	SG	PLS, mL	Sample #	pH Target	PLS, g	SG	PLS, mL	Comments
1	2.0	646.3								

Final Pulp Details:

Feed Solution vol.	601.32	mL	Final Pulp wt.	713.3	g
Feed Solution wt.	635.6	g	Filtrate Collected	646.3	g
Reactor Tare.	515.3	g	Filtrate SG	1.049	
LS pulp added	88.4	g	Filtrate Collected		mL
LS pulp density	20%	solids	Calc Filtrate vol		mL
100% LS added	17.7	g	Residue Tare	9.7	g
100% LS added	27.8	kg / t PLS	Wet Residue	63.4	g
			Dry Residue	28.57	

Final Filtration:

Diameter of filtration paper:	110	mm	Washing time:		min
type of paper (Whatman ##):	0.45 um		Clarity of wash:		
Filtration time:	5	min	Volume of wash:	252	g
Clarity of filtrate:	clear		Colour of wash:		
Colour of filtrate:	clear				
Cake thickness:	20	mm			

Project: 13849-001
Client: Appia Energy
Test: **Neut-1**

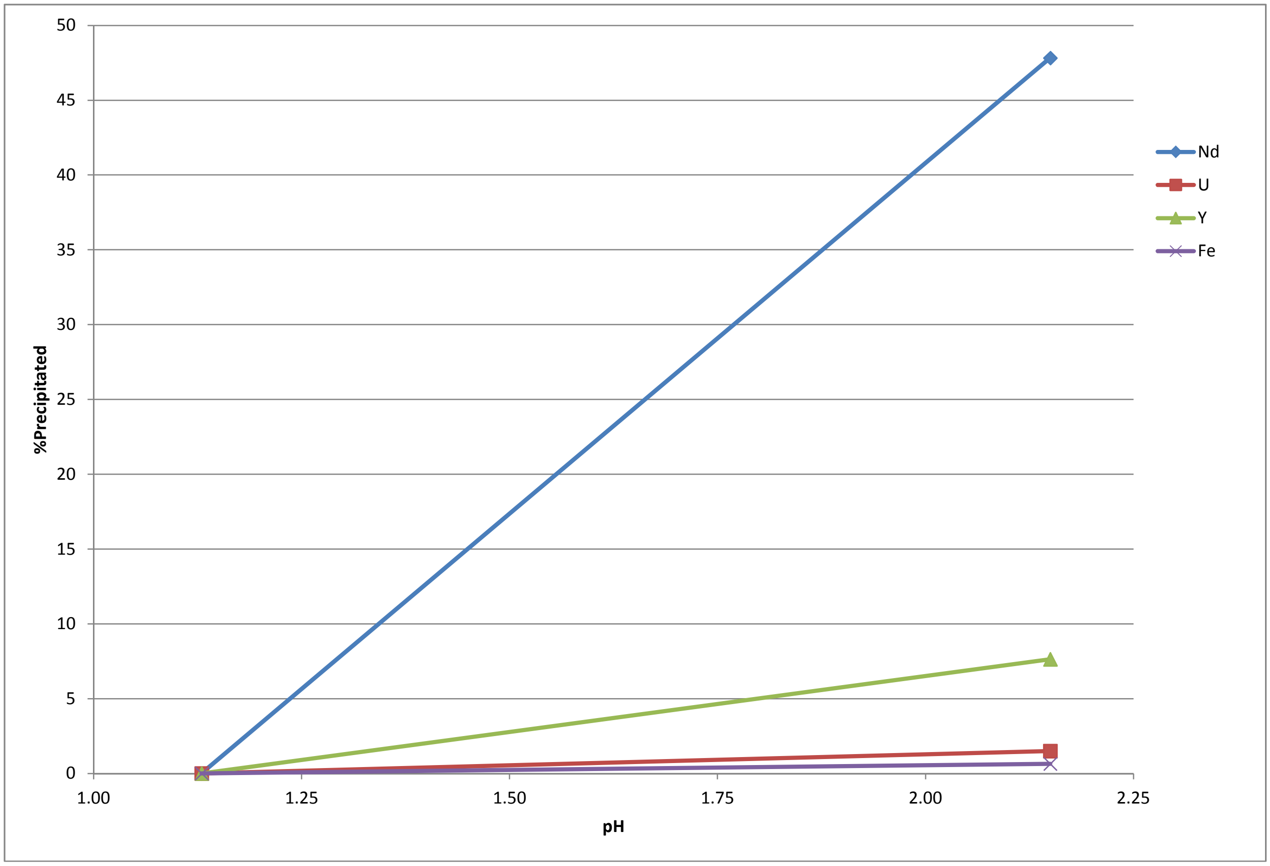
Project Technologist: Krystal Davis
Date of Test: 30-Apr-13

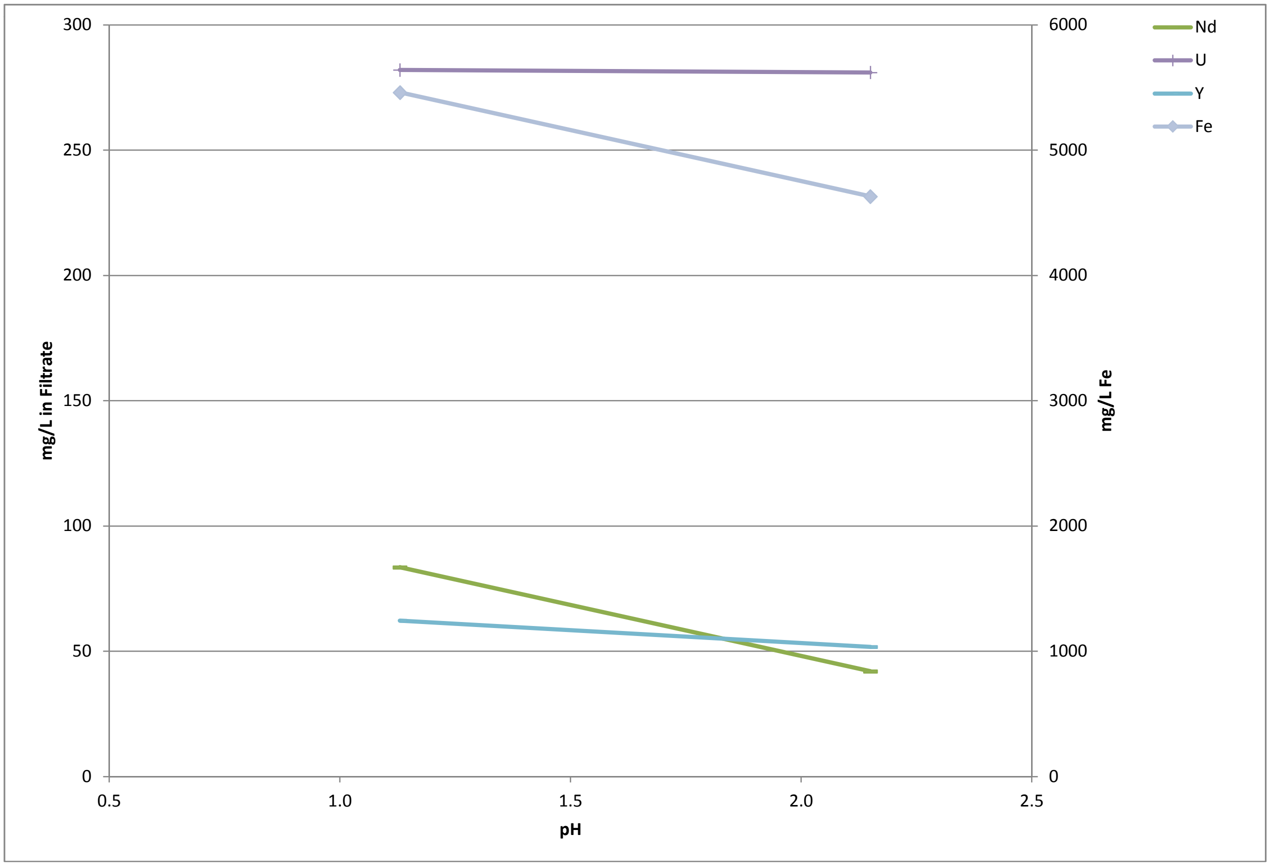
Metallurgical Balance:

Sample #	Fd	pH 2	Res		
Amt g or mL	601.32	667.9	28.6		
Assays, mg/L or g/t					
Nd	83.5	42	900		
U	282	281	<100		
Y	62.2	51.7	100	calculated concentration	
Fe	5460	4630	700		
Metal Units, mg				Sum	Balance
Nd	50.21	28.05	26	54	107%
U	169.57	187.69	3	191	112%
Y	37.40	34.55	3	37	100%
Fe	3283	3092.55	20	3113	95%
Precipitated, %		1.13	2.15	Sum	
Nd		0	48	48	
U		0	1	1	
Y		0	8	8	
Fe		0	1	1	

Project: 13849-001
Client: Appia Energy
Test: **Neut-1**

Project Technologist: Krystal Davis
Date of Test: 30-Apr-13





Project:	13849-001	**Project Technologist:**	Krystal Davis
Client:	Appia Energy	**Date of Test:**	02-May-13
Test:	**Neut-2**		

Objective: To determine precipitation S curve on combined PLS with MgO

Feed: **Combined PLS**

H&S: Refer to H_2SO_4, MgO, MSDS's
Feed solution is corrosive, avoid contact with skin and clean all spills accordingly

Procedure:

1. 350 mL of Solution was placed in a 500 mL Pyrex reactor equipped with at least 1 baffle. The solution was agitated with a Teflon impellor and mechanical mixer. The pulp was heated to **40 deg C**.
2. MgO slurry (25 % solids) was added stepwise to get to pH levels of 2.0, 2.5, 3.0, 3.5, 4.0, 4.5, 5.0, 6.0. Slurry was held at each pH for 30 min.
3. Liquor samples were taken at each pH step. (25 or 30 mL syringe, filter & collect PLS, return unwashed residue to reactor). A drop of conc. H2SO4 was added to each filtrate.
4. Reagent additions were accurately recorded. (Reagent added slowly to mixing vortex).
5. The pulp was filtered and the filtration time was noted. Wash the solids with DI water. Save sample of wash solution. Save washed wet cake for further testwork (do not submit for assay).
6. Assay all liquors for REE, Fe, Na. Assay final solids for REE, Fe.

Test Observations:

Time hh:mm	Time min	Temp °C	pH	emf mV	MgO Pulp Wt. Reading (25% solids), g	MgO (100%) added, kg/t	Notes
8:16	0	47.8	0.51	513	553.29		Time 0
8:32	16	57.6	2.01	464	548.29	3.5	hold
9:02	46	53.9	2.03	464			pH 2 sample
9:06	50	53	2.56	433	544.13	6.3	hold
9:36	80	48.4	2.56	407	543.61		pH 2.5 smaple
9:40	84	47.2	3.05	326	542.01	7.8	hold
10:10	114	42.9	3.02	310			pH 3.0 sample
10:15	119	41.7	3.5	237	537.16	11.1	hold
10:45	149	41.6	3.56	230			pH 3.5 sample
10:50	154	42.1	4.01	155	532.79	14.2	hold
11:15	179	42.3	4.14	147			pH 4.0 sample
11:30	194	47.2	4.75	90	531.93	14.8	hold
12:00	224	45.6	4.68	108			pH 4.5 sample
12:02	226	45.8	5.10	26	529.18	16.7	hold
12:42	266	47.1	5.18	77			pH 5.0 sample
12:50	274	45.7	6.04	24	527.12	18.1	hold
13:20	304	45.4	6.80	84			pH 6.0 sample, end test

Sampling INFO

			Sample					Sample		
Sample #	pH Target	PLS, g	SG	PLS, mL	Sample #	pH Target	PLS, g	SG	PLS, mL	Comments
1	2.0	16.8	1.075	16						
2	2.5	21.9	1.076	20						
3	3.0	50.4	1.075	47						
4	3.5	21.7	1.071	20						
5	4.0	20.5	1.069	19						
6	4.5	22.9	1.066	21						
7	5.0	20.9	1.065	20						
8	6.0	16.2	1.065	15						

Final Pulp Details:

Feed Solution vol.	349.08	mL	Final Pulp wt.	218.27	g
Feed Solution wt.	362.0	g	Filtrate Collected	176.1	g
Reactor Tare.	1213.8	g	Filtrate SG	1.062	
MgO pulp added	26.2	g	Filtrate Collected		mL
MgO pulp density	25%	solids	Calc Filtrate vol		mL
100% MgO added	6.5	g	Residue Tare	5.06	g
100% MgO added	18.1	kg / t PLS	Wet Residue	33.97	g
			Dry Residue	8.22	

Final Filtration:

Diameter of filtration paper:	110	mm	Washing time:	5	min
type of paper (Whatman ##):	0.45 um		Clarity of wash:	clear	
Filtration time:	5	min	Volume of wash:	278	g
Clarity of filtrate:	clear		Colour of wash:	clear	
Colour of filtrate:	clear				
Cake thickness:	20	mm			

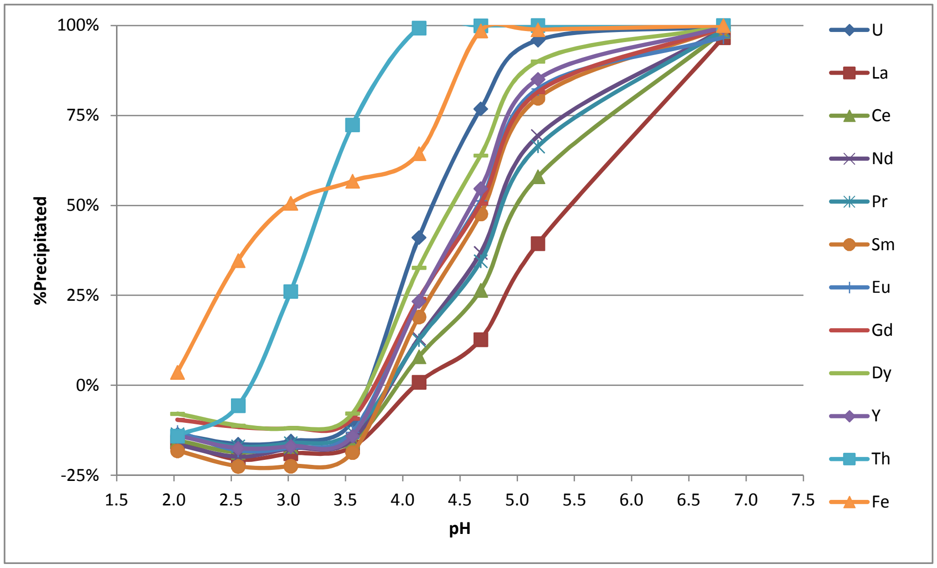
Project: 13849-001
Client: Appia Energy
Test: **Neut-2**

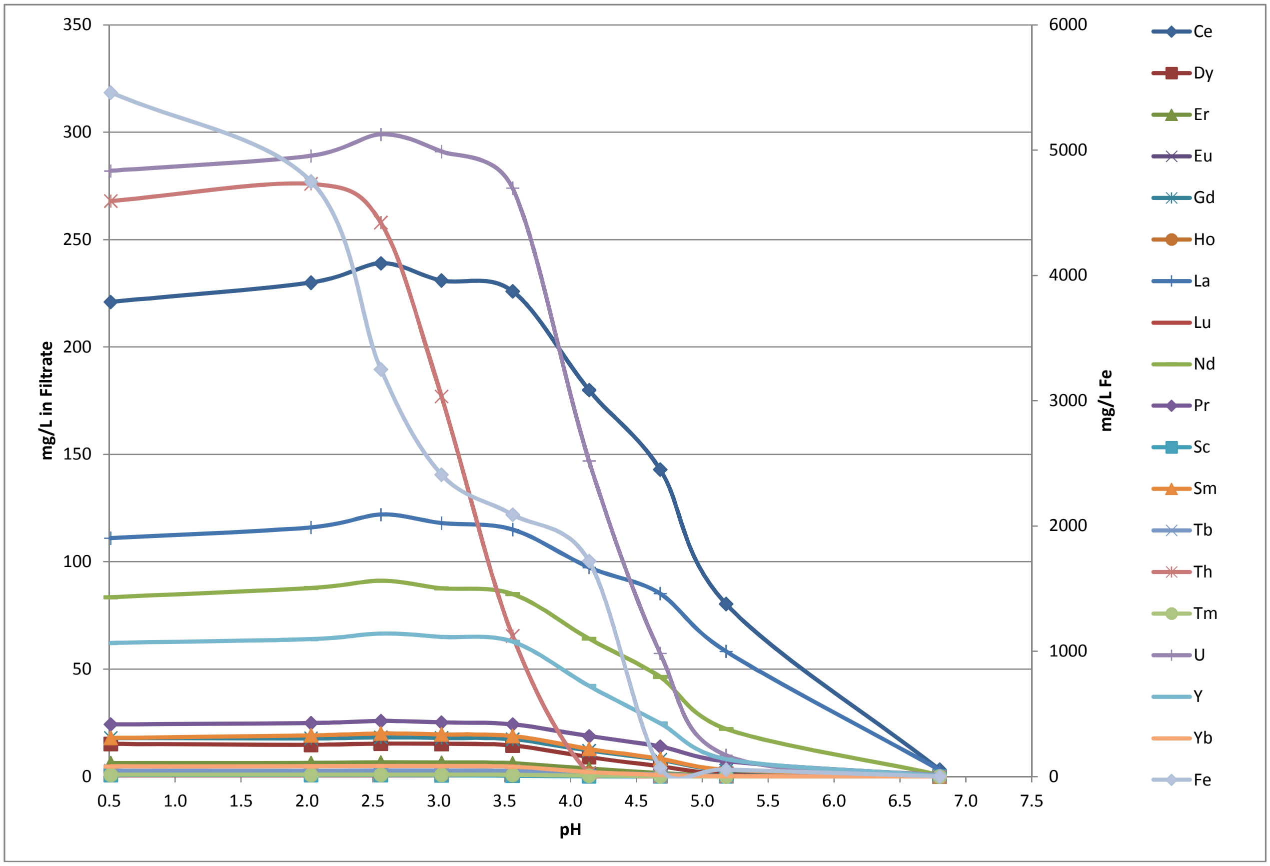
Project Technologist: Krystal Davis
Date of Test: 02-May-13

Metallurgical Balance:

Sample #	Fd	1	2	3	4	5	6	7	8	Res		
Amt g or mL	349.08	15.7	20.3	46.9	20.2	19.2	21.5	19.7	213.0	8.2		
Assays, mg/L or g/t												
Ce	221	230	239	231	226	180	143	80.4	3.29	6180		
Dy	15.2	14.8	15.4	15.2	14.5	9.05	4.83	1.32	0.05	451		
Er	6.28	6.31	6.61	6.47	6.2	3.6	1.71	0.42	0.04	193		
Eu	1.05	1.07	1.13	1.1	1.05	0.71	0.45	0.16	0.03	30.9		
Gd	18	17.8	18.3	18	17.4	12.1	7.83	2.88	0.09	524		
Ho	2.52	2.5	2.59	2.56	2.45	1.51	0.78	0.21	0.02	68.1		
La	111	116	122	118	115	97.4	85.2	58.3	3.2	3240		
Lu	0.61	0.62	0.65	0.63	0.6	0.28	0.11	0.03	0.03	19.5		
Nd	83.5	87.8	91.2	87.7	85	64.2	46.4	22.2	0.77	2390		
Pr	24.3	24.9	25.9	25.2	24.3	18.8	14	7.07	0.26	681		
Sc	0.73	0.75	0.77	0.62	0.39	0.07	0.07	0.07	0.07	25		
Sm	18	19.2	20.1	19.7	18.9	12.9	8.29	3.15	0.1	531		
Tb	2.81	2.74	2.86	2.81	2.71	1.77	1.03	0.32	0.03	85.5		
Th	268	276	258	177	65.5	1.74	0.12	0.05	0.03	8920		
Tm	0.81	0.81	0.85	0.82	0.78	0.41	0.17	0.04	0.04	25.7		
U	282	289	299	291	274	147	57.4	9.91	0.56	7070		
Y	62.2	64	66.6	65	62.8	42.2	24.8	8.05	0.25	1940		
Yb	4.66	4.74	4.93	4.84	4.56	2.09	0.77	0.16	0.02	146		
Fe	5460	4750	3250	2410	2090	1720	70.8	55.6	1.21	148000		
Na	113	102	103	101	100	100	99.4	97.9	97.6			
Metal Units, mg											Sum	Balance
Ce	77.15	3.60	4.86	10.83	4.57	3.45	3.07	1.58	0.70	51	83	108%
Dy	5.31	0.23	0.31	0.71	0.29	0.17	0.10	0.03	0.01	4	6	105%
Er	2.19	0.10	0.13	0.30	0.13	0.07	0.04	0.01	0.01	2	2	108%
Eu	0.37	0.02	0.02	0.05	0.02	0.01	0.01	0.00	0.01	0	0	109%
Gd	6.28	0.28	0.37	0.84	0.35	0.23	0.17	0.06	0.02	4	7	106%
Ho	0.88	0.04	0.05	0.12	0.05	0.03	0.02	0.00	0.00	1	1	99%
La	38.75	1.82	2.48	5.53	2.32	1.87	1.83	1.15	0.68	27	44	114%
Lu	0.21	0.01	0.01	0.03	0.01	0.01	0.00	0.00	0.01	0	0	113%
Nd	29.15	1.37	1.85	4.11	1.72	1.23	1.00	0.44	0.16	20	32	108%
Pr	8.48	0.39	0.53	1.18	0.49	0.36	0.30	0.14	0.06	6	9	107%
Sc	0.25	0.01	0.02	0.03	0.01	0.00	0.00	0.00	0.01	0	0	113%
Sm	6.28	0.30	0.41	0.92	0.38	0.25	0.18	0.06	0.02	4	7	110%
Tb	0.98	0.04	0.06	0.13	0.05	0.03	0.02	0.01	0.01	1	1	108%
Th	93.55	4.32	5.24	8.30	1.32	0.03	0.00	0.00	0.01	73	93	99%
Tm	0.28	0.01	0.02	0.04	0.02	0.01	0.00	0.00	0.01	0	0	112%
U	98.44	4.52	6.08	13.64	5.54	2.82	1.23	0.19	0.12	58	92	94%
Y	21.71	1.00	1.35	3.05	1.27	0.81	0.53	0.16	0.05	16	24	111%
Yb	1.63	0.07	0.10	0.23	0.09	0.04	0.02	0.00	0.00	1	2	108%
Fe	1906	74.37	66.06	112.99	42.25	33.00	1.52	1.09	0.26	1217	1548	81%
Na	39	1.60	2.09	4.74	2.02	1.92	2.14	1.92	20.79		37	94%
Dilution (Na-tie)		1.11	1.10	1.12	1.13	1.13	1.14	1.15	1.16			
pH		2.03	2.56	3.02	3.56	4.14	4.68	5.18	6.8			
U		-14%	-16%	-15%	-10%	41%	77%	96%	100%			
La		-16%	-21%	-19%	-17%	1%	13%	39%	97%			
Ce		-15%	-19%	-17%	-16%	8%	26%	58%	98%			
Nd		-16%	-20%	-18%	-15%	13%	37%	69%	99%			
Pr		-14%	-17%	-16%	-13%	13%	35%	66%	99%			
Sm		-18%	-23%	-22%	-19%	19%	48%	80%	99%			
Eu		-13%	-18%	-17%	-13%	24%	51%	82%	97%			
Gd		-10%	-12%	-12%	-9%	24%	51%	82%	99%			
Dy		-8%	-11%	-12%	-8%	33%	64%	90%	100%			
Y		-14%	-17%	-17%	-14%	23%	55%	85%	100%			
Th		-14%	-6%	26%	72%	99%	100%	100%	100%			
Fe		4%	35%	51%	57%	64%	99%	99%	100%			
Er		-11%	-15%	-15%	-12%	35%	69%	92%	99%			
Ho		-10%	-13%	-14%	-10%	32%	65%	90%	99%			
Lu		-13%	-17%	-16%	-11%	48%	79%	94%	94%			
Sc		-14%	-16%	5%	40%	89%	89%	89%	89%			
Tb		-8%	-12%	-12%	-9%	29%	58%	87%	99%			
Tm		-11%	-15%	-13%	-9%	43%	76%	94%	94%			
Yb		-13%	-16%	-16%	-11%	49%	81%	96%	100%			

Project:	13849-001	**Project Technologist:**	Krystal Davis
Client:	Appia Energy	**Date of Test:**	02-May-13
Test:	**Neut-2**		





Project: 13849-001
Client: Appia Energy
Test: **Neut-3**

Project Technologist: Krystal Davis
Date of Test: 02-May-13

Objective: To determine precipitation S curve on combined PLS with Limestone

Feed: Combined PLS

H&S: Refer to H_2SO_4,CaCO3, MSDS's
Feed solution is corrosive, avoid contact with skin and clean all spills accordingly

Procedure:

1. 350 mL of Solution was placed in a 500 mL Pyrex reactor equipped with at least 1 baffle. The solution was agitated with a Teflon impellor and mechanical mixer. The pulp was heated to **40 deg C**.
2. Limestone slurry (25 % solids) was added stepwise to get to pH levels of 2.0, 2.5, 3.0, 3.5, 4.0, 4.5, 5.0, 6.0. Slurry was held at each pH for 30 min.
3. Liquor samples were taken at each pH step. (25 or 30 mL syringe, filter & collect PLS, return unwashed residue to reactor). A drop of conc. H2SO4 was added to each filtrate.
4. Reagent additions were accurately recorded. (Reagent added slowly to mixing vortex).
5. The pulp was filtered and the filtration time was noted. Wash the solids with DI water. Save sample of wash solution. Save washed wet cake for further testwork (do not submit for assay).
6. Assay all liquors for REE, Fe, Na. Assay final solids for REE, Fe.

Test Observations:

Time hh:mm	Time min	Temp °C	pH	emf mV	LS Pulp Wt. Reading (25% solids), g	LS (100%) added, kg/t	Notes
8:16	0	47.1	0.88	506	555.11		Time 0
8:32	16	50.8	2.06	469	526.52	19.5	hold
9:02	46	47.8	2.16	468			pH 2 sample
9:06	50	47	2.55	447	523.5	21.6	hold
9:36	80	44	2.50	448			pH 2.5 smaple
9:40	84	42.7	3.01	374	517.48	25.7	hold
10:10	114	46.4	3.02	352			pH 3.0 sample
10:15	119	45.2	3.5	266	508.74	31.7	hold
10:45	149	42.1	3.52	260			pH 3.5 sample
10:50	154	42.1	4.01	160	500.43	37.4	hold
11:30	194	46.2	4.09	174			pH 4.0 sample
11:32	196	45.5	4.5	118	497.25	39.5	hold
12:00	224	42.3	4.64	112			pH 4.5 sample
12:06	230	45.8	5.00	55	491.76	43.3	hold
12:42	266	43.6	5.18	16			pH 5.0 sample
12:50	274	48.8	5.96	-95	452.98	69.8	hold
13:20	304	48.0	6.18	10			pH 6.0 sample, end test
						Sum	

Sampling INFO

Sample #	pH Target	Sample			Sample #	pH Target	Sample			Comments
		PLS, g	SG	PLS, mL			PLS, g	SG	PLS, mL	
1	2.0	20.5	1.041	20						
2	2.5	19.4	1.041	19						
3	3.0	19.0	1.033	18						
4	3.5	21.6	1.026	21						
5	4.0	18.5	1.016	18						
6	4.5	27.7	1.012	27						
7	5.0	15.7	1.006	16						
8	6.0	19.4	1.006	19						

Final Pulp Details:

Feed Solution vol.	352.87	mL	Final Pulp wt.	287.64	g
Feed Solution wt.	365.9	g	Filtrate Collected	199.51	g
Reactor Tare.	1158.3	g	Filtrate SG	1.003	
LS pulp added	555.1	g	Filtrate Collected		mL
LS pulp density	25%	solids	Calc Filtrate vol		mL
100% LS added	138.8	g	Residue Tare	4.12	g
100% LS added	379.2	kg/t PLS	Wet Residue	78.83	g
			Dry Residue	41.45	

Final Filtration:

Diameter of filtration paper:	110	mm	Washing time:	10	min
type of paper (Whatman ##):	0.45 um		Clarity of wash:	clear	
Filtration time:	5	min	Volume of wash:	299	g
Clarity of filtrate:	clear		Colour of wash:	clear	
Colour of filtrate:	clear				
Cake thickness:	2	"			

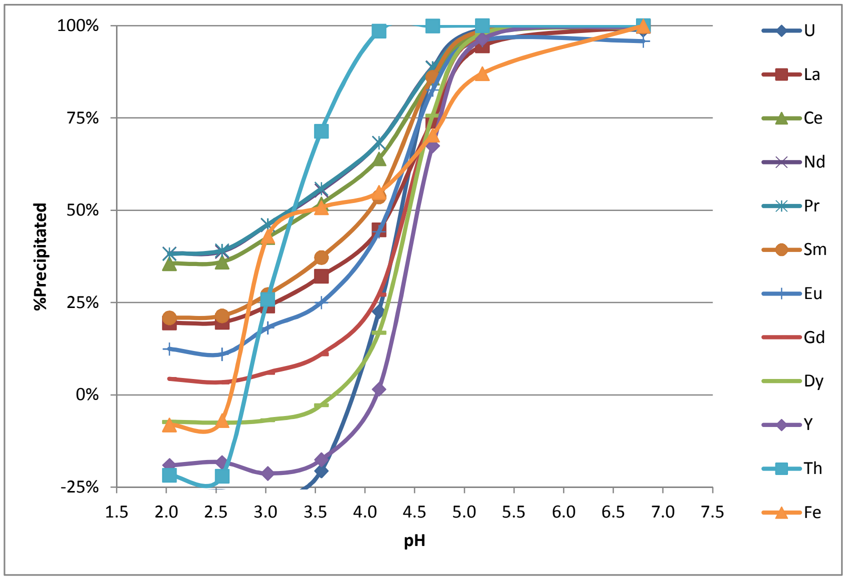
Project: 13849-001
Client: Appia Energy
Test: **Neut-3**

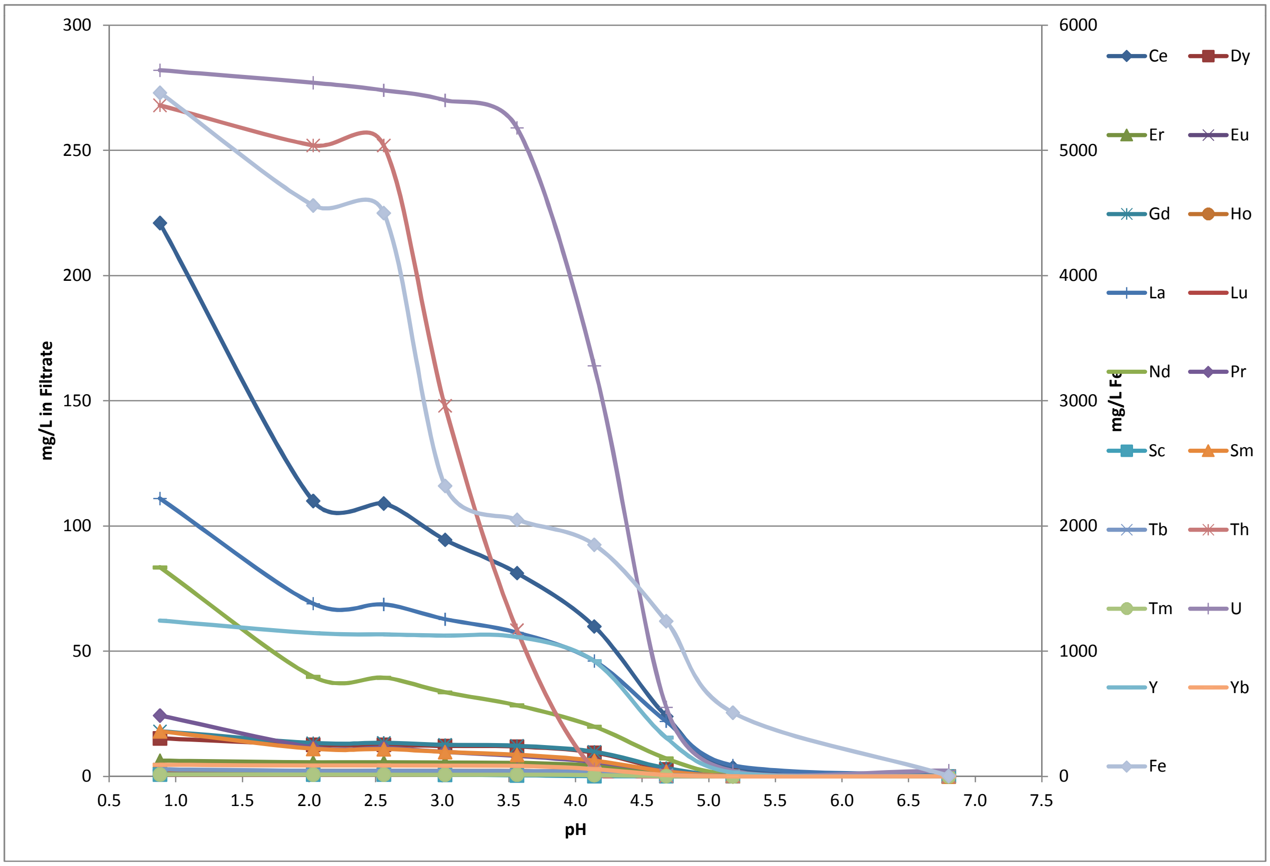
Project Technologist: Krystal Davis
Date of Test: 02-May-13

Metallurgical Balance:

Sample #	Fd	1	2	3	4	5	6	7	8	Res		
Amt g or mL	352.87	19.6	18.7	18.4	21.1	18.2	27.4	15.6	264.7	41.5		
					Assays, mg/L or g/t							
Ce	221	110	109	94.5	81.2	59.9	24	3.6	0.1	1720		
Dy	15.2	12.6	12.6	12.1	11.9	9.49	2.83	0.24	0.05	103		
Er	6.28	5.62	5.67	5.53	5.45	4.27	1.18	0.09	0.04	43.3		
Eu	1.05	0.71	0.72	0.64	0.6	0.44	0.14	0.03	0.03	7.8		
Gd	18	13.3	13.4	12.6	12.2	9.85	3.36	0.34	0.03	126		
Ho	2.52	2.18	2.22	2.14	2.13	1.68	0.51	0.04	0.02	15.3		
La	111	69	68.7	62.8	57.4	46.1	22	4.37	0.12	860		
Lu	0.61	0.58	0.58	0.57	0.56	0.39	0.08	0.03	0.03	4.2		
Nd	83.5	39.8	39.4	33.6	28.4	19.9	7.18	0.84	0.06	669		
Pr	24.3	11.6	11.4	9.76	8.17	5.79	2.12	0.26	0.03	191		
Sc	0.73	0.72	0.72	0.62	0.4	0.07	0.07	0.07	0.07	25		
Sm	18	11	10.9	9.76	8.61	6.27	1.92	0.19	0.04	138		
Tb	2.81	2.2	2.24	2.12	2.09	1.65	0.53	0.05	0.03	19.8		
Th	268	252	252	148	58.4	2.9	0.15	0.04	0.03	2070		
Tm	0.81	0.74	0.74	0.72	0.72	0.53	0.13	0.04	0.04	5.8		
U	282	277	274	270	259	164	27.6	2.16	2.28	1590		
Y	62.2	57.2	56.7	56.2	55.7	46	15.5	1.62	0.04	435		
Yb	4.66	4.36	4.42	4.31	4.25	2.96	0.61	0.03	0.02	31.9		
Fe	5460	4560	4500	2320	2050	1850	1240	509	4.05	30600		
Na	113	87.3	87.1	84.2	86.1	84.9	86.5	81.1	77.4			
					Metal Units, mg						Sum	Balance
Ce	77.99	2.16	2.04	1.74	1.71	1.09	0.66	0.06	0.03	71	81	104%
Dy	5.36	0.25	0.24	0.22	0.25	0.17	0.08	0.00	0.01	4	5	102%
Er	2.22	0.11	0.11	0.10	0.11	0.08	0.03	0.00	0.01	2	2	106%
Eu	0.37	0.01	0.01	0.01	0.01	0.01	0.00	0.00	0.01	0	0	107%
Gd	6.35	0.26	0.25	0.23	0.26	0.18	0.09	0.01	0.01	5	7	102%
Ho	0.89	0.04	0.04	0.04	0.04	0.03	0.01	0.00	0.01	1	1	96%
La	39.17	1.36	1.28	1.15	1.21	0.84	0.60	0.07	0.03	36	42	108%
Lu	0.22	0.01	0.01	0.01	0.01	0.01	0.00	0.00	0.01	0	0	110%
Nd	29.46	0.78	0.74	0.62	0.60	0.36	0.20	0.01	0.02	28	31	105%
Pr	8.57	0.23	0.21	0.18	0.17	0.11	0.06	0.00	0.01	8	9	104%
Sc	0.26	0.01	0.01	0.01	0.01	0.00	0.00	0.00	0.02	1	1	430%
Sm	6.35	0.22	0.20	0.18	0.18	0.11	0.05	0.00	0.01	6	7	105%
Tb	0.99	0.04	0.04	0.04	0.04	0.03	0.01	0.00	0.01	1	1	105%
Th	94.57	4.95	4.71	2.72	1.23	0.05	0.00	0.00	0.01	86	99	105%
Tm	0.29	0.01	0.01	0.01	0.02	0.01	0.00	0.00	0.01	0	0	113%
U	99.51	5.44	5.12	4.96	5.45	2.99	0.76	0.03	0.60	66	91	92%
Y	21.95	1.12	1.06	1.03	1.17	0.84	0.42	0.03	0.01	18	24	108%
Yb	1.64	0.09	0.08	0.08	0.09	0.05	0.02	0.00	0.01	1	2	106%
Fe	1927	89.58	84.03	42.63	43.16	33.74	33.98	7.92	1.07	1268	1604	83%
Na	40	1.71	1.63	1.55	1.81	1.55	2.37	1.26	20.49		32	81%
Dilution (Na-tie)		1.29	1.30	1.34	1.31	1.33	1.31	1.39	1.46			
pH		2.03	2.56	3.02	3.56	4.14	4.68	5.18	6.8			
U		-27%	-26%	-28%	-21%	23%	87%	99%	99%			
La		20%	20%	24%	32%	45%	74%	95%	100%			
Ce		36%	36%	43%	52%	64%	86%	98%	100%			
Nd		38%	39%	46%	55%	68%	89%	99%	100%			
Pr		38%	39%	46%	56%	68%	89%	99%	100%			
Sm		21%	21%	27%	37%	54%	86%	99%	100%			
Eu		12%	11%	18%	25%	44%	83%	96%	96%			
Gd		4%	3%	6%	11%	27%	76%	97%	100%			
Dy		-7%	-8%	-7%	-3%	17%	76%	98%	100%			
Y		-19%	-18%	-21%	-18%	2%	67%	96%	100%			
Th		-22%	-22%	26%	71%	99%	100%	100%	100%			
Fe		-8%	-7%	43%	51%	55%	70%	87%	100%			
Er		-16%	-17%	-18%	-14%	10%	75%	98%	99%			
Ho		-12%	-14%	-14%	-11%	11%	74%	98%	99%			
Lu		-23%	-23%	-25%	-20%	15%	83%	93%	93%			
Sc		-28%	-28%	-14%	28%	87%	87%	87%	86%			
Tb		-1%	-3%	-1%	2%	22%	75%	98%	98%			
Tm		-18%	-19%	-19%	-17%	13%	79%	93%	93%			
Yb		-21%	-23%	-24%	-20%	15%	83%	99%	99%			

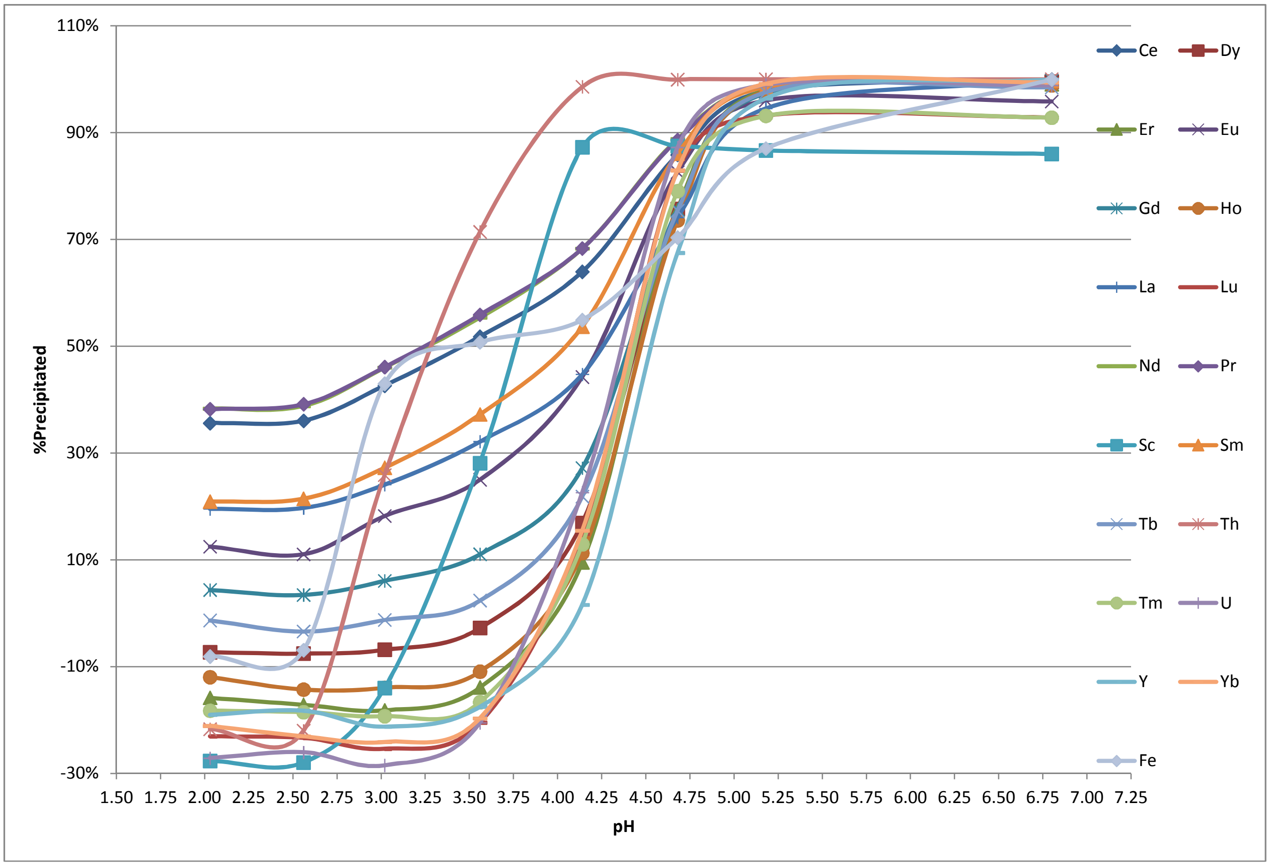
Project:	13849-001	**Project Technologist:**	Krystal Davis
Client:	Appia Energy	**Date of Test:**	02-May-13
Test:	**Neut-3**		





Project: 13849-001
Client: Appia Energy
Test: **Neut-3**

Project Technologist: Krystal Davis
Date of Test: 02-May-13



Project: 13849-001
Client: Appia Energy
Test: **Neut-4**

Project Technologist: Krystal Davis
Date of Test: May 16. 2013

Objective: To neutralize the UIX Barren to pH 3.0 and remove ferric Fe prior to REE IX

Feed: **UIX-1 Barren**

H&S: Refer to H_2SO_4, MgO, MSDS's
Feed solution is corrosive, avoid contact with skin and clean all spills accordingly

Procedure

1. Take the UIX barren in a suitable (500 mL) reaction flask and heat to 40ºC.
2. MgO slurry (25 % solids) was added stepwise to get to pH levels of 3.0. MgO was added slowly over 1 hour and held at pH 3 for 1 h. MgO addition was recorded.
3. The slurry was filtered and solids washed. Wash solution was discarded, PLS was recovered and stored for REE IX
4. A 15 mL sample of the filtrate was submitted for REE and ICP and Fe2+ assays.
5. The cake was dried and submited for REE and FE assays.
6.

Test Observations:

Time hh:mm	Time min	Temp °C	pH	emf mV	MgO Pulp Wt. Reading (25% solids), g	MgO (25%) added, g	Notes
11:20		40.1	1.92	466	567.83		start MgO additions
11:27		45	2.1	459			hold
11:34		47.2	2.31	447	563.48		hold
11:40		47.7	2.5	427	561.62		hold
11:48		47.8	2.71	393	559.26		hold
11:52		47.8	2.85	360	558.72		hold
11:56		47.8	2.96	341	558.37		hold for 1h
13:00		41.3	3.1	318			end test

Final Pulp Details:

Start Volume:	460	mL
Final Slurry Weight:	493.75	g
Cake Wet Wt.:	11.29	g
Cake Dry Wt.:	2.89	g
Filtrate SG:	1.036	

reactor tare: 1202.26g

Project No.	13849-001	**Project Technologist:**	Krystal Davis
Client:	Appia Energy	**Date of Test:**	03-May-13

Test No. **UIX-1**

Objective: To remove U prior to REE-IX

Procedure:

1. Take the pH 2.0 PLS from Neut-1 test, agitate in a 1 L beaker or kettle.
2. Add 5 mL of wsr per L of solution and agitate for 1.5 h.
3. After 1.5 h, filter out the resin. Measure wsr and dry and record weight.
4. Submit dried resins for U and Fe.
5. Take a 15 mL aquesous sample and submit for U and Fe assays.
6. Using the solution from the first contact, add an additional 5 mL/L of fresh resin.
7. Repeat for total of 4 cross-current contacts.
8. Store the remaining solution after the 4th contact for REE IX tests.

Reactants:

Aqueous Feed	Resin Phase
pH 2.0 Combined PLS	Ambersep 920U SO4

Conditions:

Phase Ratios	5 mL/L	Agitation	agitated flask
Contact Time	1.5 h	Temperature	(ambient)
Equilibrium pH	natural	Atmosphere	open

Data:

Starting Volume of Feed Solution: 600 mL

	Vol. Aq., mL	Clr of Aq	wsr in, mL	wsr out, mL	dry resin, g	Aq. Sample, mL	Clr of Barren	pH of barren
Contact 1	595	golden	3.0	3	1.21	15	clear-golden	2.19
Contact 2	570	golden	2.9	2.9	1.08	15	clear-golden	2.2
Contact 3	545	golden	2.8	2.8	1.02	15	clear-golden	2.14
Contact 4	535	golden	2.8	2.8	0.98	15	clear-golden	2.14
Contact 5	500	golden	2.8	2.8	0.95	15	clear-golden	1.92

Final Volume of Barren:	480	mL
Final pH:	1.92	
Final ORP:	466	mV

Project No.	13849-001	**Project Technologist:**	Krystal Davis
Client:	Appia Energy	**Date of Test:**	16-May-13

Test No. **REEIX-1**

Objective: To load REE on OC1026 Resin

Procedure:

1. Retreive the pH 3.0 filtrate from Neut-4
2. Conduct resin contacts on a vibrator shaker at the indicated volumes and raios below.
3. After 2 h, filter out the resin. Measure wsr and dry and record weight.
4. Submit dried resins for REE-MS and ICP-OES scans (REE priority)
5. Take a 15 mL aquesous sample and submit for REE and ICP Scans.

Reactants:

Aqueous Feed	Resin Phase
Neut-4 Filtrate (pH 3.0 after UIX)	OC1026

Conditions:

Phase Ratios	various	Agitation	agitated flask
Contact Time	2	Temperature	(ambient)
Equilibrium pH	natural	Atmosphere	open

Data:

Starting Volume of Feed Solution: 460 mL

	Vol. Aq., mL	Resin Ratio, mL/L	wsr in, mL	wsr out, mL	dry resin, g	Aq. Sample, mL	Clr of Barren	pH of barren
Contact 1	100	120	12.0	7.37	3.94	15	clear	2.36
Contact 2	180	60.0	10.8	6.41	3.6	30	yellow	3.1
Contact 3	180	20.0	3.6	2.06	1.2	30	orange	3.5

Notes:

Contact 1: Lots of resin floating and stuck to walls. Solution went from orange to clear.

Contact 2: Some resin floating and on walls (~40%) floating, some (~40%) the rest in solution or settled. PLS colour went from orange to yellow.

Contact 3: Some resin (~50%) floating, the rest settled on the bottom nothing suspended. PLS colour remained orange.

Project No. 13849-001 **Project Technologist:** Krystal Davis
Client: Appia Energy **Date of Test:** 16-May-13

Test No. **REEIX-1**

Test Results:

	Assays, mg/L				% Loading		
Element	Feed	120 mL/L	60 mL/L	20 mL/L	120 mL/L	60 mL/L	20 mL/L
TREE	336	239	261	308	29%	22%	8%
LREE	236	224	226	227	5%	4%	4%
HREE	100	15	34	81	85%	66%	19%
La	64.9	63.3	64.5	63.8	2%	1%	2%
Ce	107	103	102	102	4%	5%	5%
Pr	11.3	10.6	10.5	10.7	6%	7%	5%
Nd	42.4	38.9	40.3	40.5	8%	5%	4%
Sm	10.7	8.51	9.13	9.79	20%	15%	9%
Eu	0.68	0.48	0.55	0.62	29%	19%	9%
Gd	13.2	7.89	10.5	12.5	40%	20%	5%
Tb	2.34	0.65	1.42	2.03	72%	39%	13%
Dy	12.2	1.87	5.72	10.7	85%	53%	12%
Ho	2.25	0.2	0.79	1.88	91%	65%	16%
Er	5.48	0.21	1.08	4.21	96%	80%	23%
Tm	0.72	<0.04	0.05	0.43	>94%	93%	40%
Yb	4.3	0.03	0.13	1.74	99%	97%	60%
Lu	0.54	<0.03	<0.03	0.18	>94%	>94%	67%
Y	58.4	3.33	14.1	46.9	94%	76%	20%
Sc	0.54	<0.07	<0.07	<0.07	>87%	>87%	>87%
U	14.6	0.27	1.29	9.52	98%	91%	35%
Th	107	2.72	8.92	60.7	97%	92%	43%
Fe	2020	1700	1930	2090	16%	4%	-3%
Fe^{2+}	1900	1540	1690	1700			
Al	2690	2400	2520	2640	11%	6%	2%