Extraction of Rare-Earth Elements from Iron-Rich Rare-Earth Deposits

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Large iron-rich rare-earth-bearing deposits exist in China, Southern Africa, Canada and Australia. Although these carry significant reserves of rare-earth elements, many are not exploited because no economically viable process exists to do so. The mineralogy of the difficult deposits is complex, with rare-earth minerals of particle size of less than 20 microns disseminated in a matrix of iron oxide, such as hematite, magnetite or goethite. A process comprising fine milling for liberation of rare-earth minerals and physical upgrading of the resulting materials would be technically challenging, inefficient and not economically viable. Globally, research efforts are directed towards the development of novel methods and new reagents to overcome these difficulties.

PyEarth is a novel process currently under development at Mintek. The process aims to achieve efficient extraction of rare earths from mineralogically complex iron-rich, rare-earth bearing ores. Tests at laboratory scale on samples from deposits originating from Southern Africa have been conducted successfully, with positive results. It has been proven that the extraction of rare earths from this source is technically feasible, robust and viable. PyEarth incorporates pyrometallurgical pretreatment of these ores that allows efficient extraction of the rare earths by hydrometallurgical means.

INTRODUCTION

Rare earths are listed as strategic metals by the United Nations because of their properties and increasing usages (Long et al., 2010). Among many possible applications, rare earths are used to sustain today's lifestyle in the manufacture of magnets, catalysts, weaponry and production of green electricity. Currently, about 130 000 t/a of rare-earth oxide (REO) is produced globally, with about 90% of this production coming from China. Consumption of REO is projected to increase at a rate of 6–10% per annum and could reach a volume of 200 000 t/a in the very near future (Long et al., 2010; Haque et al, 2014; Li and Yang, 2014; Wang et al., 2015). More than 50% of today's world consumption of rare earths is currently located in China.

The proven global rare-earth reserves are estimated at about 110 million tons of REO (Wang et al, 2015). China's resources of rare earths are the largest and are estimated at about 48 million tons of REO or about 43% of the global reserves. Other rare-earth resources are located in Russia (17.3%), the United States of America (11.8%), Australia, India, Brazil and Greenland (Gupta and Krishnamurthy, 2004). As reported by Gupta (Gupta and Krishnamurthy, 2004), Namibia has the second-largest reserve worldwide, with about 20 million tons of REO, and South Africa has about 1 million tons. Despite the existence of large deposits, Africa's contribution to the global supply of rare earths is insignificant.

Production of rare earths from ores involves a complex process that is designed according to the nature and mineralogy of each deposit. Run-of-mine (ROM) rare-earth-bearing ores are beneficiated by physical methods into a concentrate that is further leached to extract the rare-earth species. In rare situations, an in-situ leaching process (also called solution mining) is carried out, in which the leaching reagent is injected into wells drilled in the orebody to selectively leach the rare-earth species and the leachate is collected, thus minimising environmental pollution (Li and Yang, 2014). Rare earths are extracted from the leachate and transformed into pure rare-earth oxides, chlorides or halides by hydrometallurgical means. Pure rare-earth metals and alloys are produced by direct reduction of pure rare-earth oxides, reduction of anhydrous chlorides and fluorides and fused salt electrolysis of rare-earth chlorides or oxide–fluoride mixtures. Preparation of pure rare-earth oxides may be carried out by selective oxidation, selective reduction, fractional crystallization, ion exchange and solvent extraction. Chlorides and fluorides are produced by transformation of rare-earth oxides (Gupta and Krishnamurthy, 2004; Mishra and Anderson, 2014).

A rare-earth deposit is valuable only when the metal values can be extracted in a cost-effective, efficient, sustainable and environmentally friendly manner. A large number of deposits have been explored worldwide; however, only a few are mined. The most prominent deposits mined are Bayan Obo (China), Mount Weld (Australia) and Mountain Pass (USA), which is currently not in operation as of 2015 (Haque et al., 2014; Li and Yang, 2014).

Bayan Obo is the largest exploited iron-rich rare-earth deposit. The ore contains up to 6% REO and 35% Fe. More than 90% of rare-earth elements in the ore exist as independent minerals, and about 4 to 7% of the total rare-earth minerals are disseminated in the iron-oxide minerals. The deposit contains bastnaesite and monazite with magnetite and hematite as the dominant iron minerals. This ore is beneficiated into a 65% REO concentrate with an overall rare-earth recovery of about 61% via a complex beneficiation process. The milled ore of 90–95% passing 74 µm is submitted to low-intensity magnetic separation (LIMS) and high-intensity magnetic separation (HIMS) which produce a mixed concentrate of about 9.78–12% REO. The latter is upgraded by flotation using appropriate reagents. Reverse flotation of iron minerals, an important step in this process, is carried out to produce the final iron concentrate while the cleaner tailings, containing most rare-earth minerals, is processed to obtain a rare-earth concentrate. This is followed by selective flotation to separate bastnaesite and monazite. Higher recovery, up to 75%, can be achieved at lower concentrate grade (Li and Yang, 2014).

The Mount Weld mine was recently opened and its ore contains the highest REO grade, averaging 15.4%. Its mineralogy has been described as a secondary rare-earth phosphate, but those phosphates (most likely monazite) are encapsulated in iron-oxide minerals (Haque et al., 2014). The Mount Weld process produces rare-earth and iron ore concentrates for further beneficiation.

Upgrading some of rare-earth ores was found to be challenging owing to the complexity of their mineralogy. This was the case for two specific iron-rich rare-earth bearing ore deposits in the Southern African region. Milling was not able to liberate the rare-earth minerals from the iron mineral matrix. This paper presents the results of an investigation into the processing of iron-rich, rare-earth-bearing ores from the Southern African region.

PROCESS OVERVIEW: IRON-RICH RARE-EARTH-BEARING ORES FROM SOUTHERN AFRICA

Brief Mineralogy and Implications

Two samples of iron-rich rare-earth-bearing ore from two distinct deposits in Southern Africa were evaluated at Mintek. Both samples contain a bulk of monazite minerals as fine grains of less than 20 µm in size; mineralisation occurs as coatings on hematite/goethite or grains disseminated in the hematite/goethite matrix. The ores also contains bastnaesite, synchysite, ancylite, pyrochlore, REE-apatite and rhabdophane. Quartz, jarosite, svanbergite, apatite, barite, mica, clays, crandallite, gorceixite, carbonates and nordite are present in the ores as gangue minerals. Monazite and ancylite

often occur as sub-microscopic inhomogeneous grains in association with Fe-oxyhydroxides. Ancylite is associated with calcite and apatite. The Fe-carbonate is dominated by dolomite/ankerite with minor calcite and Fe-oxyhydroxides. The gangue contains REE-minerals less than 10 µm in size. Fine milling to below 20 µm in size, to liberate the rare-earth minerals and selectively separate them from the rest (principally from hematite or goethite), was extensively investigated by Mintek's Minerals Processing Division. Application of most conventional physical separation techniques was found to be inefficient.

Globally, research efforts are directed towards the development of novel methods and new reagents to overcome these difficulties. Bulk leaching of these ores to extract rare earths inevitably solubilises iron minerals and increases acid consumption. Removal of the iron from the leachate requires excessive amounts of reagents such as lime. This, in addition to the discarding of iron precipitate, increases the cost of the process, usually beyond economic viability. Extracting rare earths without affecting the value of the iron minerals is required to improve the economics of the processing of these ores.

High-Temperature Processing of Iron-Rich Rare-Earth Ores

Separation of iron and rare earths is theoretically possible by carbothermic reduction and could be used as an alternative to physical upgrading of the ore. As shown in the Ellingham diagram for pure oxides in Figure 1, iron oxide can be selectively reduced over rare-earth oxides; however, in the present case, smelting would be required to enable efficient separation of iron metal from REO-bearing slag. REO are stable under the iron smelting conditions generally described by the temperature and oxygen partial pressure, therefore they will report to the slag phase whilst the iron will report to the metal phase. Although this separation may be possible, the most important step is the extraction of the rare earths from the slag generated. Hydrometallurgical methods were investigated for this purpose.

In practice, smelting of the ore and solubilisation of the REO by mineral acid leaching is affected by the ore mineralogy and impurities. Ore mineralogy affects the kinetics of both smelting and leaching.

Process Principle of the PyEarth Process

The PyEarth process flowsheet, as presented in Figure 2, incorporates smelting of ore, leaching of rare earths from the resulting slag and recovery of the rare earths from the leach liquor. Smelting of the ore to produce a concentrated rare-earth- containing slag and metal product and leaching of the resulting slag constitute the main critical steps of this process. In practice, there is more than one option available for smelting, leaching and recovery of rare-earths from the solution; however, for simplicity, the flowsheet considers using a direct current (DC) open-arc furnace to smelt the ore, leaching in a hydrochloric acid medium, precipitation of impurities, precipitation of rare earths and recovery of HCl. As opposed to smelting reactors such as blast and submerged-arc electric furnaces, the DC open-arc furnace is able to process ore fines without prior preparation and the complex nature of the mineralogy is easily managed by way of temperature control. This reactor would be an appropriate option for the smelting of mineralogically complex iron-rich rare-earth bearing ores.

Upgrading of the slag prior to leaching to remove elements other than REO is not included in the basic flowsheet presented in this paper; however, this process option may improve the efficiency of the hydrometallurgical flowsheet if physical upgrading (such as electrostatic and magnetic concentration, froth flotation or gravity concentration) is introduced post-taphole.

Results of preliminary smelting and leaching tests are presented and discussed in the next sections.

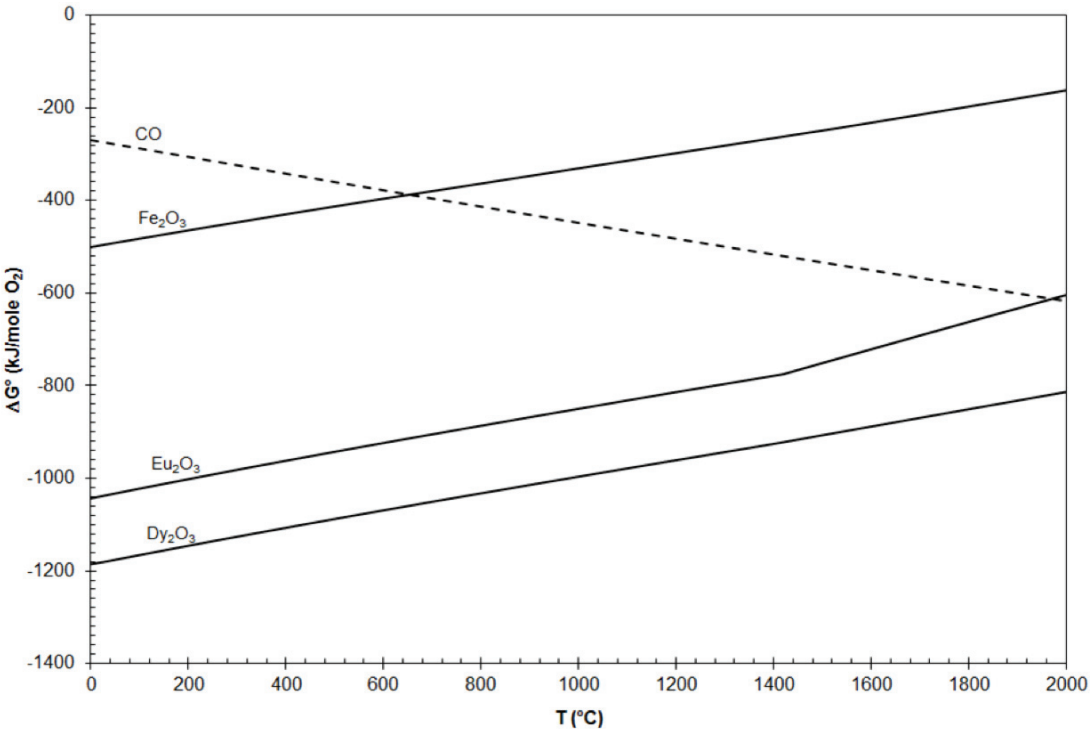

Figure 1. Ellingham diagram of iron and REO.

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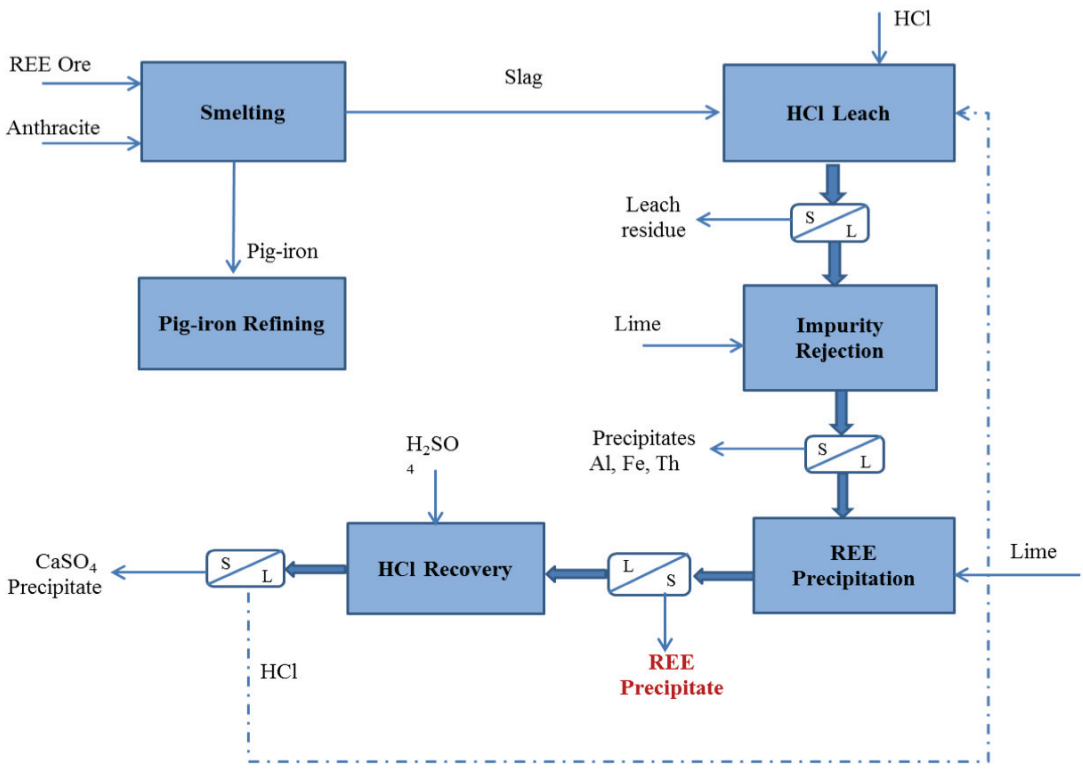

Figure 2. Proposed flowsheet of the PyEarth process.

SMELTING

Raw Materials

A sample of iron-rich rare-earth-bearing ore from a Southern African deposit was subjected to a series of laboratory tests. The sample was submitted for chemical and mineralogical characterisation. The bulk chemical composition was determined by a Varian Vista-PRO CCD simultaneous inductively coupled plasma optical emission spectrometer (ICP-OES). The morphological and phase chemical compositions were determined by Zeiss MA15 scanning electron microscopy (SEM) equipped with a Bruker energy-dispersive spectrometer (EDS) and Bruker D8 advanced X-ray diffractometer (XRD), respectively. The chemical composition of the ore is given in Table I. The proximate analysis of the anthracite used is given in Table II. High-purity CaO was used as a fluxing agent to improve the smelting process, particularly to decrease the slag liquidus temperature and viscosity—constraints that are exacerbated at small scale.

Table I. Summary of the bulk chemical composition of the iron-rich rare-earth ore 1

MgO (%)	Al_2O_3 (%)	SiO_2 (%)	CaO (%)	TiO_2 (%)	V_2O_5 (%)	Cr_2O_3 (%)	MnO (%)
1.13	6.48	6.08	2.06	3.87	0.11	0.07	9.09
FeO(OH) (%)	SiO_2/ Al_2O_3	SiO_2/ MgO	P_2O_5 (%)	La (mg/kg)	Ce (mg/kg)	Pr (mg/kg)	Nd (mg/kg)
46.9	0.94	5.38	1.77	6 060	10 200	921	3 900
Sm (mg/kg)	Eu (mg/kg)	Gd (mg/kg)	Dy (mg/kg)	Er (mg/kg)	TREE (%)	Th (mg/kg)	U (mg/kg)
478	145	435	166	100	2.34	221	72

Ho,Tm, Lu, Yb have concentrations less than 100 mg/L

Table II. Summary of the bulk chemical composition of the anthracite (mass %)

Ash	Volatile	Fixed carbon	Total sulphur
4.74	6.19	89.1	0.56

Experimental Procedure

Laboratory smelting tests were conducted in 30 kW and 60 kW induction furnaces. Figure 3 shows a schematic of the induction furnace setup. The raw material components for the test conditions specified in Table III were blended and packed in a graphite crucible. Tests 1 and 2 were un-fluxed tests conducted at stoichiometric anthracite addition and different temperatures. In this particular case, anthracite amount was calculated to fully reduce iron and manganese oxides. Tests 3 and 4 were fluxed with 0.5% CaO with stoichiometric anthracite addition and two different operating temperatures.

A packed crucible was placed at the centre of the graphite susceptor in the induction furnace. A B-type thermocouple was secured next to the crucible and argon gas was purged into the furnace to create an inert environment. The power was switched on and manually increased at a rate of 20–50°C/min until the target temperature was reached. Thereafter, the crucible was held for about 30 min at the target temperature. The furnace power was then switched off and the crucible was left to cool down in the Ar gas atmosphere inside the furnace. The cold crucible was removed, weighed and then broken to separate the metal and slag phases. The different phase samples were chemically and mineralogically analysed.

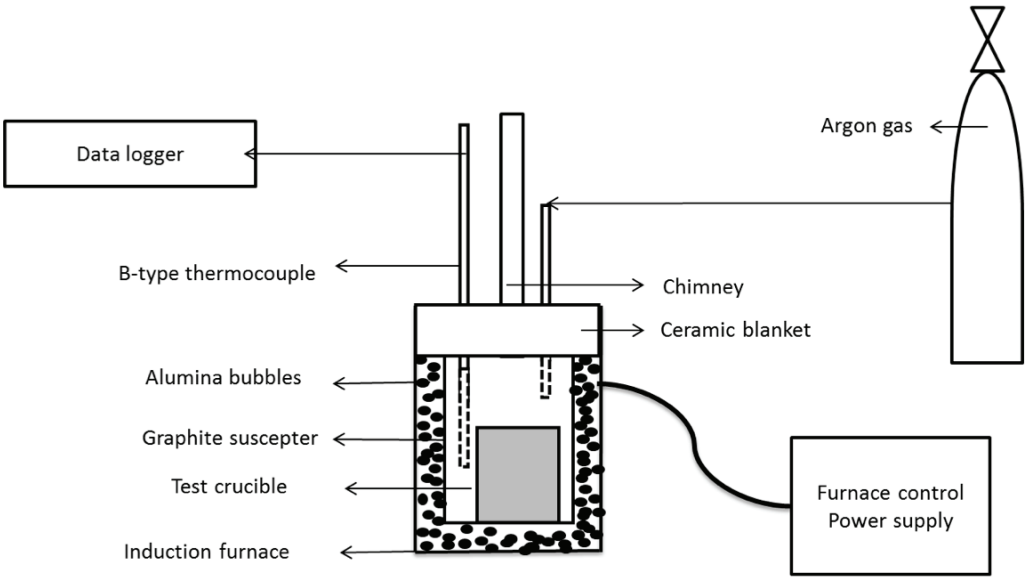

Figure 3. Schematic presentation of induction furnace experimental setup.

Table III. Conditions for laboratory smelting tests

Test	Anthracite (%)	CaO (%)	Temperature (°C)
1	100	0	1700
2	100	0	1800
3	100	0.5	1700
4	100	0.5	1600

Smelting Results and Discussion

The mass balance for each smelting test, including the metal and slag masses, are given in Table IV. Smelting produced a slag that was further leached in HCl medium. The smelting target was to achieve a slag with the highest REO grade, as well as achieve a clean separation between the slag and metal products.

Table IV. Mass balance for the smelting tests

Test	Ore (g)	Anthracite (g)	CaO (g)	Total mass in (g)	Products (g)				Total mass out (g)	T (°C)
					Alloy + slag	Alloy	Slag	Gas / LOI		
1	400.0	60.0	0	460.0	265.0	158.0	107.0	192.0	457.0	1700
2	400.0	60.0	0	460.0	249.0	164.0	85.0	211.0	457.0	1800
3	100.0	16.0	0.5	116.5	66.3	44.3	22.0	50.2	116.5	1700
4	100.0	16.0	0.5	116.5	70.8	42.1	28.7	45.7	116.5	1600

Slag produced per unit of ore smelted and total rare-earth oxide grade of slag product

The mass of slag generated per kilogram of ore smelted varied between 210 and 300 g. On average, a mass of about 250 g was produced per kilogram of ore smelted. As achieved by a physical method such as flotation, smelting is here principally a concentration step. Applying flotation principles to this case, the slag product may be called a concentrate and the metal called tailings (although it is expected that the metal product may, in fact, have substantial economic value on its own merit). The mass pull, expressed in mass percent, is the portion of concentrate produced per unit of ore processed. In this smelting case, a mass pull of 25% is achievable when processing a unit of ore. This mass pull would depend on the ore composition, particularly the iron, manganese and chromium contents, the loss on ignition (LOI) and the extent of smelting reduction. The value of 25% can be considered as a baseline

for the concentration through smelting for this particular ore. The chemical analyses of the slags produced in the smelting tests are given in Tables V and VI.

A total REO (TREO) grade of the slag of at least five times greater than the grade in the ore was achieved; which is a significant concentration step. Slags containing up to 13.8% TREO were produced from ore with 2.5% TREO. A higher grade of TREO in the slag is desirable because this will decrease the amount of feed to the leaching process and thus improve the process economics. Of course, this needs to be balanced against the cost of the smelting step.

The smelting conditions were optimised at laboratory scale with regard to the contents of FeO and MnO in the slag and amount of slag generated. This had an impact on the slag TREO grade. Within the scope of this testwork, an optimal slag TREO grade of 13.6% was produced by smelting the ore at the highest temperature of about 1800 °C. However, addition of 0.5% lime to the smelting recipe produced a slightly higher slag TREO grade of 13.8% at a relatively lower temperature of about 1700 °C. It is desirable to operate at lower temperature because this reduces furnace electricity costs.

Alloy quality

A carbon-saturated iron–manganese alloy was produced from these smelting tests, as can be seen in Table VII, for the conditions investigated. Table VIII presents the alloy rare-earth contents. It was observed (in other results not presented here) that the composition of the alloys is strongly related to the extent of manganese reduction. An increase in manganese reduction increases the manganese content of the alloy but decreases its iron concentration by dilution. Manganese oxide in the slag is not desirable because it is leached by the acid; increasing its content increases the acid consumption in the leaching step. Its reduction to the alloy in the smelting step is therefore desirable. The reduction of manganese oxide is driven by a combination of the reductant addition, temperature and slag basicity. The preferred alloy composition produced from Ore 1 was: 79–84% Fe, 10–12.5% Mn, 2–4% C, 3–6% Si and 0.7–1.3% P. This composition falls within the commercial manganese steel range, which comprises 11–13% Mn. The alloy may require P removal.

Recovery of rare-earth oxides to the slag

The recovery of TREO to the slag was calculated as the portion of REO of the ore that reported to the slag phase. The highest content of total rare earth elements (TREE) in the metal was about 600 mg/kg, while the highest slag TREO was about 13.8%. Within the scope of the present testwork, and considering the above values, a recovery of rare earths to the slag of more than 95% was achieved. This confirmed that REO were stable at the smelting conditions tested and fully reported to the slag phase, irrespective of the final slag grade achieved.

Smelting energy requirement

The theoretical amount of energy required by a large-scale smelting process was calculated using *FactSage* software. *FactSage* predicted furnace energy consumption in the range of 1.0 to 1.2 MWh/t ore. An electric arc furnace will be used to produce two products: the rare-earth-rich slag and the iron alloy. The plant finances may be structured in such a way that the smelting energy cost is entirely covered by the sale of the metal product.

Table V: REE chemical composition in the slag

Test	Concentration (mg/kg)																	Concentration (%)	
	La	Ce	Pr	Nd	Sm	Eu	Gd	Dy	Ho	Er	Tm	Yb	Lu	Y	Tb	Th	U	REE	REO
1	21 601	42 805	4 328	19 338	2 327	829	1 416	972	170	380	50	257	45	3 904	139	-	-	9.86	11.6
2	29 459	49 500	5 444	19 144	2 685	727	2 420	1 048	173	456	58	362	50	4 547	275	863	210	11.60	13.6
3	28 970	57 197	5 232	15 077	2 335	603	1 816	785	153	364	54	321	45	5 015	210	986	256	11.80	13.8
4	19 429	32 083	3 569	12 516	1 756	550	1 654	698	113	292	39	234	33	2 536	195	862	452	7.57	8.9

Table VI Chemical composition of the other metal oxides in slag, with total REO

Test	Concentration (%)											BI	SiO_2/Al_2O_3	SiO_2/MgO
	MgO	Al_2O_3	SiO_2	CaO	TiO_2	V_2O_5	Cr_2O_3	MnO	FeO	REE	REO			
1	5.24	24.4	24.9	9.97	10.5	0.10	0.08	5.81	3.16	9.86	11.6	0.31	1.02	4.75
2	7.74	29.6	18.3	13.6	4.40	0.11	0.09	4.69	1.39	11.60	13.6	0.45	0.62	2.36
3	6.82	25.8	13.2	14.1	5.10	0.09	0.07	1.67	3.04	11.80	13.8	0.54	0.51	1.94
4	5.74	19.9	22.8	10.1	11.4	0.09	0.07	5.24	4.64	7.57	8.9	0.37	1.15	3.97

* Traces amounts of CoO, NiO, CuO, ZnO, PbO <0.01%; BI (Basicity Index) = (%MgO+%CaO) / (%SiO_2 + Al_2O_3)

Table VII. Alloy analyses

Test	Concentration (%)												
	Si	Ti	V	Mn	Cr	Cu	Ni	Ca	Fe	Mg	Al	P	C
1	3.30	0.35	0.10	10.1	0.05	0.05	0.05	0.05	81.2	0.05	0.14	0.83	3.77
2	0.78	0.67	0.11	12.0	0.08	0.03	0.04	0.19	83.9	0.09	0.34	0.73	4.09
3	4.56	0.37	0.10	12.4	0.05	0.05	0.05	0.10	79.2	0.05	0.26	1.26	1.57
4	1.72	0.39	0.13	11.4	0.04	0.05	0.04	0.09	81.5	0.04	0.26	0.69	3.68

Table VIII. REE analyses of selected alloys

Element	Concentration (mg/kg)		Element	Concentration (mg/kg)	
	Test 1	Test 2		Test 1	Test 2
Ce	250	155	Sc	18	66
Dy	5	3	Sm	1	8
Er	2	1	Tm	11	1
Gd	16	7	Y	1	15
La	143	99	Yb	-	1
Nd	123	59	Th	7	9
Pr	31	18	U	50	32

RARE-EARTH ELEMENT RECOVERY FROM SLAG BY LEACHING

Selection of Leach Procedure

Two classical treatment routes as commonly used for REE recovery; namely, a caustic crack followed by leaching of the REE with HCl or a sulphuric acid bake followed by a water leach, were evaluated for treating this slag (Gupta and Krishnamurthy, 2004). The TREE extractions were relatively poor, yielding 49% recovery for the caustic cracking route at a reagent dosage of 2000 kg NaOH per ton of slag and 41% at a reagent dosage of 2200 kg H_2SO_4 per ton of slag for the sulphuric acid bake route. As a third leach option, a direct HCleach was evaluated. This yielded the best results, as discussed in more detail below.

Slag Characteristics

The slag produced in Test 1 was used as the feed for direct HCl leaching tests. The concentrations of REE and major impurities are listed in Table IX. Minor REE were omitted. The slag was milled to a grind of 80% −35 µm prior to leaching.

Table IX. Chemical analysis of slag submitted to HCl leach tests

Element	Concentration	Unit	Element	Concentration	Unit
La	25 120	mg/kg	Th	738	mg/kg
Ce	49 509	mg/kg	U	234	mg/kg
Pr	5 022	mg/kg	Mg	2.63	% (w/w)
Nd	18 387	mg/kg	Al	13.0	% (w/w)
Sm	2 722	mg/kg	Si	6.43	% (w/w)
Eu	986	mg/kg	Ca	6.32	% (w/w)
Gd	1 671	mg/kg	Ti	5.09	% (w/w)
Dy	1 118	mg/kg	Mn	3.03	% (w/w)
Er	428	mg/kg	Fe	7.39	% (w/w)
TREE	10.5	% (w/w)			

Experimental Procedure for Direct HCl Leaching Process

The milled slag was leached under four different process conditions, varying the acid dosage and the pulp density. Table X summarises the various leaching conditions. The leaching was conducted by pulping the slag in HCl whilst the pulp was agitated and the temperature controlled at 60 °C. Operating parameters of redox potential, temperature and pH were recorded at regular intervals. After a leach duration of 2 h, the pulp was filtered, the filter cake was washed thoroughly and subsequently dried, weighed and analysed. The solution volumes were measured prior to analysis of the product liquors.

Table X. Process conditions for direct HCl leach

Test number	5	6	7	8	9
Temperature (°C)	60	60	60	60	60
Residence time (h)	2	2	2	2	2
Solids content (% w/w)	20	10	20	10	10
Acid dosage (kg/kg of slag)	708	861	2201	3445	3445
Acid dosage (kg/kg of TREE)	6.4	7.8	20	31	31
Grind size (80% passing µm)	35	35	35	35	35

Results and Discussion

The operating conditions and solid-based leach extractions for REE and impurities are summarised in Table XI. TREE leach efficiencies ranged between 94% and 97%, and seemed insensitive to pulp density and acid addition. At the lowest acid addition of 708 kg per ton of slag, the REE recovery was lowest (94%); however, co-extraction of some impurities was also markedly depressed. Nonetheless, owing to the co-extraction of deleterious impurities, such as Al, Ti, Mn, Fe, Th and U, further treatment of the liquor prior to REE separation would be required. Although the TREE recovery was slightly lower, the test conducted at 20% (w/w) pulp density at an acid addition of 708 kg/t slag could be considered the most favourable, owing to lower lixiviant addition (and associated lower lixiviant costs) and decreased downstream neutralising reagent requirements for both the residual free acid and impurities. In addition, a higher pulp density would allow a smaller equipment size and thus reduce the capital costs for this step.

CONCLUSIONS

The PyEarth process was developed by Mintek to recover rare-earth elements from iron-rich rare-earth-bearing carbonatite ores of complex mineralogy from Southern Africa. PyEarth comprises two critical steps; namely, the smelting of ore producing typically an Fe-Mn alloy as a potential saleable by-product, and recovery and concentrating the REO to the resulting slag from which these are subsequently leached using HCl.

The investigation successfully demonstrated that the smelting process can be operated as a pre-concentration process for REE. In addition, the Fe and Mn are converted into a potentially saleable by-product, whereas in the classical treatment routes these would have to be discarded in a large quantity as tailings with economic and environmental implications.

In the smelting step, close to full recovery of the rare-earths to the slag is achieved, regardless of the final slag REO grade. A rare-earth recovery in excess of 94% to the PLS was achieved using this process. Although some deleterious impurities are co-extracted from the slag, these would be removed downstream by conventional hydrometallurgical means. Process optimisation is required and future plans include pilot-plant scale testwork.

The laboratory testwork showed that the PyEarth process is indeed a technically feasible, robust and viable process for the extraction of rare earths from the mineralogically complex iron-rich rare-eart-bearing ores from the Southern African region.

ACKNOWLEDGEMENTS

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Table XI. Summary of actual operating conditions and extractions of REE and impurities

Test	5	6	7	8	9
Final temperature (°C)	61	61	59	59	59
Residence time (h)	2	2	2	2	2
Solids content (% w/w)	20.0	9.4	20.0	8.9	8.9
Final pH	1.00	0.00	0.07	0.00	0.00
Final Eh	131	-144	-102	298	298
Acid dosage (kg/t of slag)	708	861	2201	3445	3445
Acid dosage (kg/kg of TREE)	6.4	7.8	20.0	31.3	31.3
Mass loss (%)	44.7	55.6	56.2	53.7	57.2
Grind size (80% passing µm)	35	35	35	35	35
Solid-based metal extraction, % (w/w)					
Mg	27.0	46.1	46.7	40.5	44.9
Al	57.6	69.7	69.5	66.6	69.4
Si	<1	<1	<1	<1	<1
Ca	79.2	90.9	86.1	86.7	87.7
Ti	27.6	45.1	49.1	44.9	49.3
Mn	38.1	64.5	61.7	60.6	63.1
Fe	93.4	95.7	94.5	95.4	95.4
Th	N/D	93.2	90.0	91.2	92.3
U	N/D	95.3	94.0	93.2	94.8
La	93.8	96.8	95.1	95.9	96.1
Ce	94.4	97.0	95.4	96.2	96.4
Pr	93.5	96.6	94.8	95.6	95.9
Nd	93.8	96.7	95.0	95.8	96.0
Sm	94.1	97.0	95.4	95.9	96.2
Gd	90.3	95.7	93.2	94.2	94.4
Dy	93.7	97.4	95.8	96.3	96.4
Ho	94.0	97.5	96.1	96.4	96.5
Er	92.5	96.9	95.2	95.5	95.7
TREE	93.8	96.8	95.1	95.9	96.1

N/D = not determined

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